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The Electrochemical Publishing Company, publisher of this journal, was formed ten years ago with the co-operation and support of the McGraw Publishing Company. All of the stock of the Electrochemical Publishing Company has now been acquired by the McGraw Publishing Company, which will, hereafter, be the publisher of METALLURGICAL AND CHEMICAL ENGINEERING. The stockholders of the Electrochemical Publishing Company have become stockholders of the McGraw Publishing Company. There will be no change in the editorial and business staff, or in the management, except that the staff will be able to call more freely than ever upon the professional and business resources of the McGraw Publishing Company.

I sincerely hope that this step, which was taken with the hearty and unanimous consent of the minority stockholders of the Electrochemical Publishing Company, will enable this journal to fulfill in future still more energetically its mission in technical journalism and to remain true to the ideals for which it has stood these ten years. To this end I pledge my full support.

JAMES H. MCGRAW.

General Business Conditions and the Metal Markets in 1912.

The expansion in the metal markets in the past year has exceeded the expectations of many and the hopes of a few. For, considered as far as output and prices go, 1912 has been a banner year in the metallurgical industry. Indeed, in the past two months it has been pushed to the extreme in making prompt deliveries in certain lines. As the metal markets are such a sure index of the growth of the business of the United States, in considering the past general aspects of the metal business, due regard must be paid to those fundamental commercial prime-movers, the crops and the psychological atmosphere. So, it must be clearly held in mind that all crops save cotton were in 1911 considerably below the average and that in 1912 all the crops were considerably above the average. In the ultimate analysis, the price of foodstuffs is the chief immediate factor in industrial life. It can be laid down as a fact, therefore, that the balting tendency manifest a year ago had its main cause in the poor condition of agriculture, and that the rush of business in the latter part of 1912 was due to the assurance that the farmer had been bountifully aided by Dame Nature and that there was a surplus of food products. This in point of fact counteracted the effects of the presidential campaign.

At the same time man cannot live by bread alone, and the profound influence of what people think and feel preponderates over the direct acting stomachic forces—potent as they are in their own sphere. The general mental and spiritual food that was consumed in America contained certain good elements and certain poisonous elements. Yellow journals and several of the magazines generate in one

a feeling of horror and disgust. Yet there are many millions of calm, sensible and sober men and women who form the real opinions of society and on whose mental balance and on whose self-control our entire national welfare and life depends. These people are the substance of America. The others are the excrescences. Now, life is derived from sentient intelligence and from real sources of subjective power.

With these conceptions in mind, let us turn to specific events of the past year. At first there was a braking effect on the wheels of industry because of the poor crops of 1911, and due to the usual apprehension of a presidential year. However, copper and spelter started to move fast and strong in the early spring. The great expansion in the electrical trade resulted in an enormous consumption of copper and as we have said before there cannot long be in this age of electricity too much copper.

For a considerable period in recent years, business has been dominated by a spirit of rational conservatism. This spirit has shown faith in the future and practically has exercised energy in present endeavor. Without this we would now be in a perilous condition. Since there was a good foundation of conservatism, when once the good crops of 1912 were harvested, something that approached a boom arose. The election over, a scarcity of labor and supplies was marked. At the same time the Balkan war came on and caused a general restriction of credit in Europe. Now, slowly and surely, the inhibiting forces manifest themselves—in traces only, to be sure, but certainly in a slight tendency to quiet. Labor and political troubles, phenomena closely connected with all industrial life, were manifest throughout the year. The fear of them is an inhibiting force. All such forces are much needed checks on extravagance and the actions of the unintelligent. An individual case now is the query of the business man, "Why should I build now when labor and capital and supplies are scarce, high, and dear?" No observer can deny that this is the case and no reasoning person can refuse to admit that a negative answer is the logical one.

Another inhibiting force is that large sums of money have been sunk in enterprises of sure merit, such as the Panama canal, whose return to capital must needs be long delayed. This state of affairs is not so pronounced as in 1905-7, but it is nevertheless a check to expansion. It would seem inevitable and natural to expect for several reasons soon a holding off in the demand for metals and a market much more favorable to the consumer. A feeling of uncertainty begets caution, for the emotion of doubt in an intelligent person becomes volitional caution.

Let us look at this phase broadly and consider the varying conditions of mental unrest that is so marked in the civilized world. Let us also remember the great ease and accuracy with which intelligence nowadays can be transmitted. From both it follows that we are to have short periods of expansion followed by equally short periods of retrenchment for the next few years as has been the case for the past five years. This is a healthy state of affairs, for it shows that moral forces are working quickly and surely in the industrial world.

Now the greatest fault with America is the widespread extravagance starting in with the so-called upper classes

that ought to know better but do not, and progressing thence by social migratory imitation to the working classes. This causes more trouble than all the wars, more than all vice, for it is fundamental in its perverse character. Fortunately or unfortunately, according to the viewpoint, this country is a young giant on whom excesses seem to make little effect. But the time has come for the country to put away childish things and to become a strong man under control.

This same extravagance engenders envy and is so the cause of political and social unrest. We can be sure that it makes for destruction, and, if it were not for the fact that it is not universal and that the great law of compensation acts against its increase, things would surely be pretty bad. Now, all the waste of extravagance prevents a larger healthy growth in the metal business.

Besides these far-reaching causes, there is, of course, the uncertainty that the coming tariff legislation brings. There seems to be a feeling that this country could profitably import some of its labor "in cans and boxes" and that the best way to stop labor troubles is by a gradual lowering of the tariff and that so immigration will be lessened and the nation will have time to absorb and assimilate the vast mass of immigrants already here so that we can reach a condition of social equilibrium. Moderate action in this way will be of great benefit, but any running amuck will distil over into business unrest.

After all has been said, however, we have certain facts to count on and to recite. The pig-iron production has reached the greatest annual total and the greatest daily average in its history. This is likewise true of steel. Prices of both have averaged fair.

Copper and spelter have made the greatest production as yet recorded and in the face of this prices have advanced. Lead has not made as satisfactory a showing. Apparently the copper business has not yet felt the incoming of several large producers. As the consumption both abroad and in this country has been enormous, it has been easy to keep copper prices at a good level. Whether this has been caused by the centralized control is, of course, a question. Undoubtedly the consumption has been so strong that such a process as that called "pegging" would have been easy. But, be that as it may be, 12.50 cents is altogether too low a price to be paid for the "red metal" if capital is to be attracted into the business at a rate needed for the demands of an electrical age. It is sure that new mines will not be developed unless the price of the metal is so high to make profits large enough to justify the risk. Still, all things considered, copper looks strong though much depends on European demand.

Spelter has likewise undergone an enormous expansion in the face of an increasing price. For the last six months spelter has averaged 7 cents per pound or over at New York, and has reached a price of 7.90 cents, thereby breaking all records. The zinc smelter has reaped a rich harvest, but as the prices of metal have been extremely high, prices for ore have been also high and the zinc mining companies have not suffered. Joplin, while still the most prominent ore center, does not hold the prominent place that it did formerly, and we have seen in 1912 Butte, the great copper district, becoming a producer of zinc ore on a large and serious scale.

For many reasons it can be concluded that there is present a spirit of caution both among producers and consumers of metal that is bound to check any decided increase of further expansion. But it can also be concluded that after this steadying influence has had its work, the fact that food-stuffs will be cheapened inevitably is bound to enlarge the purchasing power of the industrial classes. There will thus be a larger margin that can be spent otherwise and in the early spring we should see a stiffening of prices. There are born some good judges of metal markets, who believe that the latter tendency will be found to be so strong as to obliterate the evidences of the first. Later in the year, when the question of new crops comes up, there is another restricting factor that will come into play. So, eliminating the effect of chance events and allowing for the uncertainties of politics, the next year has a good outlook.

Last January we based our conservative views on the poor crops of 1911. Events have shown that our views were too conservative. This year our views may prove too optimistic. But, with much good feeling among sensible people of all classes and with the full dinner-pail that nature has provided the country, we reasonably expect that the metallurgical industry will receive the good treatment that it deserves for converting potential mineral wealth into the actual wealth of refined metals, which latter are modern civilization's prime requisites.

A Noteworthy Year in Iron and Steel.

The year which has passed into history was a most noteworthy one in the iron and steel industry, made so by commercial conditions as to demand which were not remarkable in themselves but which disclosed conditions in the industry deserving of special mention and in some respects of very careful thought.

There has been, as all know, a wonderful "revival" in the industry. This revival has not been produced by a spectacular increase in demand. In November, 1911, mills were making up rolling schedules, in many cases, from turn to turn, and were able to ship most products in from one day to one week after the receipt of specifications, while prices were dropping to their lowest point, as to finished steel since early in 1899 and as to pig iron since 1904. At the close of 1912 pig iron had advanced \$4 a ton and finished steel products an average of \$6.50 to \$7 per net ton, while the mills had actual specifications on books for three months of full operation, and contract business for from three to five months additional operation.

This was a wonderful reversal in the commercial position of the industry, yet the interesting fact is that in November, 1911, production was at a rate fully 75 per cent. as great as the rate in the three closing months of 1912. Consumption appears to have increased by one-third, which is not much, but there is reason to suspect that the consumption in ordinary channels has not really increased as much as that. In November, 1911, exports were at the rate of 2,250,000 tons per annum, and in the closing months of 1912 at the rate of 3,000,000 tons, an increase of 750,000 tons. Comparing the same periods there was an increase in the rate of iron and steel consumption by car shops of fully 1,000,000 tons per annum, and a considerable increase in the consumption of other railroad material. It seems fair to estimate that out-

side of the increases in exports and the increases in railroad consumption there has not been an increase in general demand of more than about 20 per cent at the outside. Of course, it must be recognized that the action of buyers in placing contracts far ahead, and in specifying material which cannot be delivered in less than three months is purely a result of the sold-up condition of the mills, and not a product of their own commercial position. In the condition in which these consumers found themselves in November, 1911, they would have contracted and specified just as far ahead had they observed a condition of the mills being behind in deliveries as they have been of late. Specifications accumulate when mills are slow in delivery and thus produce greater slowness until the movement plays itself out. That is the course in a free market, the roughly "unscientific," perhaps, but perfectly natural.

Much will be said in the current trade reviews of what caused this so-called wonderful revival in the iron and steel industry, and what further will come of it. With a free market, where what occurs is largely natural and inevitable, it avails little to discuss these points. Of much more interest, and of vastly more profit, is it to dwell upon the conditions in the alignment of the iron and steel industry which the developments of 1912 have disclosed.

First and foremost is the peculiar condition disclosed as to the supply of coke. We are familiar with lists of the beehive ovens of the country and statements as to the capacity of the by-product ovens. We have also seen compilations of the total blast furnace capacity of the country. Such compilations have a probable error which might be negligible when the compilation is used for some purposes, but which is altogether too great to make it worth while to devote time to a comparison of the relative capacities for coke-making and pig-iron production, equal to coke consumption. Far more conclusive is a study of commercial conditions, of market results, for there the conclusions are written in actual money, and "money talks."

From the low point, about January 1, 1912, pig iron in northern markets advanced an average of \$4 per ton. Contracts for Connellsville coke for the first half of 1912 were made in November, 1911, per net ton at ovens, at \$1.55 and \$1.60, and in December at \$1.05. At the close of 1912 the same coke was held, for the first half of 1913, at \$3.50, an advance equal to fully \$2 in the cost of making a ton of pig iron. Lake Superior iron ore was advanced, for the season of 1913 62 cents on Bessemer and 55 cents on non-Bessemer ores, equal to about \$1 in the cost of making pig iron.

The major operation is that of making pig iron, yet only one-fourth of the advance in pig iron stays with the blast furnace, which is an altogether unreasonable commercial movement. There are no coke ovens idle because there are not prospective customers for the coke, but there are many idle blast furnaces, with a pig-iron market awaiting them, and idle because they cannot secure coke at a suitable price.

The coke-making capacity of the Connellsville region is substantially the same as it was in 1906, six years ago. Meanwhile the blast-furnace capacity of the country has been increased almost one half, i.e. give each existing blast furnace the supply of raw materials and the market for pig iron, which existed in 1906, and about one-half more pig iron could be made. The commercial capacity is not

one-half greater, simply because the raw materials are not available, and market prices for pig iron are lower. Blast furnaces have been built in the past six years to the extent of about 9,000,000 tons annual pig-iron capacity, requiring 10,000,000 tons of coke and the coke supply has not been increased by anything like that amount.

Again, depending upon the market movement to disclose physical condition, one finds that billets and sheet bars have advanced \$10 per gross ton in this upward movement of about a year's duration, while the finished steel products made from them have advanced somewhat less than \$7 per net ton. A reasonable alignment would be that the finished product, the thing of greater value, should advance more per ton than the unfinished material. As outlined at some length in our current iron and steel market review, the condition arises largely from the fact that finishing capacity is usually in excess of steel-making capacity, through the expectation of builders that all branches of the finishing trade will not usually be fully employed throughout, the further fact that the transition from Bessemer to open-hearth steel has left Bessemer capacity unemployed when the attendant finishing capacity is equally available for rolling open-hearth steel, and finally the fact that there is a large demand for rails, while many of the billet and sheet bar mills are convertible into rail mills.

Obviously there is required a greater expansion in open-hearth steel-making capacity. The rate of new construction in this direction has been much less in the past two years than in immediately preceding years. Of the Bessemer capacity thrown idle by the change in demand towards open-hearth steel, there has been considerable utilization by adoption of the duplex process, but the developments in the building of pure open-hearth departments and the constitution of duplex departments has not been as rapid as the market movements show it should have been. That is, perhaps, not all of the story. The whole industry may be shaken any day by finding itself under the necessity of adopting a method of producing sound ingots, requiring little if any cropping. Comparison of steel ingot and rolled-steel production statistics show that there is being produced to-day in the steel works and steel rolling mills about 8,000,000 tons per annum of new steel scrap, practically all of which finds its way into open-hearth steel works, but a large part of which would suddenly disappear should a method of producing sound ingots be adopted. A large premium upon the non-scrap processes, like the duplex, would instantly be written, and those who had early adopted such processes could count themselves most fortunate.

This much for two clear facts which the movements in the year 1912 have disclosed, in that certain departments of the industry have not developed as they should. Perhaps it would be true to say that the whole industry has not grown as rapidly as it should have done. Five years ago—and the point is that many in the trade do not realize that it is as long as five years ago—men were saying that the industry was overgrown and that a period of years would be required for demand to grow up to capacity. For five years this demand has had a chance to grow. For two or three years following 1907, the year which saw that memorable slump in demand, capacity grew somewhat rapidly, but in the past two years it has grown but slowly. The country

is now making pig iron at the rate of only between 32,000,000 and 33,000,000 tons per annum, although apparently the existing capacity is strained, but in 1906 it actually produced 25,307,191 tons, and from the rate of growth which was shown by the increase from 1,205,663 tons in 1866 to 5,683,329 tons in 1886 and then to 25,307,191 tons in 1906, the tonnage for 1912 can be extrapolated as 40,000,000 tons, instead of the 29,600,000 tons estimated as the actual production of 1912 and the rate at the close of the year of only slightly more than 32,000,000 tons.

Metallurgy of Gold and Silver in 1912.

The past year was marked by many advances in the treatment of gold and silver ores. As might be expected, cyanidation held the premier place among processes, and strengthened its position as the prevalent method of gold recovery. No mention has been made of an attempt to revive chlorination, although one company in Utah is reported to have adopted a process in which a chloridizing roast is an important feature. Nothing of special importance has occurred in the way of electrolytic amalgamation, in spite of the fact that the records of the Patent Office indicate activity among inventors in this branch of metallurgy. It appears that there may be a legitimate field of operation for such processes, and we may look for tangible results from some of the proposals that have been made by competent engineers.

In the domain of amalgamation interest must center naturally in the radical departure at the Nipissin high-grade mill, in the Cobalt district, Ontario, Canada, where high-grade silver ore is given a combination treatment by amalgamation and cyanidation. Ore containing 2500 oz. silver per ton is ground to 20-mesh size and treated in a tube mill with 4 tons of mercury, 2 tons of 5 per cent cyanide solution, with a charge of 6 tons of pebbles. About 3 tons of ore is thus treated at once. The remarkable feature of the process is that over 95 per cent of the silver in the ore is recovered by amalgamation in the tube mill. The remainder is largely recovered from the cyanide solution.

From a business viewpoint, the most important event of the year in the cyanide industry was the decision of the U. S. Court of Appeals sustaining the validity and fundamental nature of the Moore vacuum filter patents. Vacuum filtration transformed the cyanide process as applied to slime, and contributed largely to the increase in gold production by cyanidation. It was natural that so important a process should be modified by other inventors, and equally natural that the originator should consistently defend his invention. To the victor belong the spoils, but we are of the opinion that a policy of moderation will be more effective than one of strict demand for rights. Doubtless the end of the litigation is not yet in sight.

The steady advance of continuous decantation during the past year has been another feature of the cyanide process. Conceding the great benefits which resulted from vacuum filtration, it is nevertheless true that some other method of treating slime has been earnestly sought. If one such could be discovered which would eliminate the capital expenditure and cost of operation for filters, another great step in advance would be taken. Judging from the variety of conditions under which continuous decantation has been success-

fully applied, it would seem as though the desired process had been found. Sound in theory and comparatively simple in operation, it seems destined to grow in favor. Certainly the filter decision referred to above cannot militate against the spread of continuous decantation; rather will it tend to call attention to the merits of a non-filtering system which is already a proved success.

Regarding the agitation of slime pulps in cyanide solution, there has been a tendency to adopt a system of successive treatments with intermediate change of solutions, placing thickeners between the agitators. This practice has resulted in better extraction than could be obtained in one agitation for a long period of time. There are indications that mechanical agitation, or rather a combination of mechanical and air agitation, is growing in favor as against air agitation alone. The combination offers the opportunity to apply the two features to the work for which they are best adapted. Another reason for seeking some other method than air agitation is the objection to the Pachuca tank. Its height is objectionable in mill design where all appliances must be inclosed in the mill building, and the power consumption is quite an item.

Zinc dust precipitation has grown in favor for obvious reasons, and many installations of this system have been made during the past year, even in small mills of, say, 100 tons capacity. Experiments have been made in the manufacture of a zinc ribbon as a substitute for zinc shavings, but while the material itself has apparently given satisfaction, mechanical difficulties in its manufacture in large quantities have not been overcome. The method tried was to allow molten zinc to run in small streams over a rapidly revolving water-cooled cylinder.

The use of warm cyanide solutions has been adopted in some places, particularly in continuous decantation plants, where it is quite as easy to add hot water as cold in compensating the moisture loss in the discharged pulp. The general impression has prevailed that warm solutions are beneficial in cyanide extraction, but doubt has been raised in at least one instance, where a tabulation of recoveries from various departments of the mill system failed to show the expected improvement in recovery by cyanide.

Taken as a whole the progress in cyanidation has been very gratifying. No radical changes have been made in the chemistry of the process, and improvement has been along the line of better mechanical application. None of the proposed modifications of the standard process has met with success, although several of them reached, and even passed, the Patent-Office stage.

Improvements in the Metallurgy of Lead.

The use of accessory processes in preparing lead ore for smelting, and the recovery of by-products, were the two most important lines of development in lead metallurgy in 1912.

The increasing quantity of concentrates offered to reduction works, and the corresponding decrease in production of crude lump ore, have necessitated the adoption of some method of preparing the former for the blast furnace. Under former methods of roasting concentrates the product was too fine to form a large part of the furnace charge.

The quantity of such material, however, was comparatively small, and by briquetting it with lime and flue dust it could be handled without undue loss; but with the great increase in quantity came an urgent necessity for preliminary roasting and agglomeration.

If we may judge from the wide adoption of the Dwight & Lloyd method of sinter roasting, this process is meeting the need fully. Not only has it proved beneficial in the smelting of concentrates, but it has demonstrated the value of "pre-digested food" for blast-furnace reduction. Furnace capacity is practically doubled when the charge is composed largely of sinter-roasted ore, and the smelting process runs more smoothly. Some Western plants are smelting in half the number of furnaces the same tonnage of ore formerly treated in all the stacks. This preliminary treatment has proved so satisfactory that one company has adopted the plan of crushing lump ore and sinter roasting it before smelting. At other plants where the Huntington-Heberlein process was adopted some years ago, the Dwight and Lloyd machines are being introduced. As to the relation of the Huntington-Heberlein process to the Dwight-Lloyd process some remarks by Dr. Heberlein in our columns devoted to Readers' Views and Comments in the present issue will be found of interest.

A commendable departure which was emphasized last year by the largest lead smelting company was the increased attention paid to the recovery of by-products, and the establishment of a technical research department. In the past, lead, copper, gold and silver have been the more tangible metals recovered, and these have received attention to the exclusion of others which have escaped unnoticed. Perhaps the centralizing of the industry, particularly the refining operation, has called attention to the minor quantities of other metals which gradually accumulate in flue dust, matte and bullion.

In any event, a number of metals are now being recovered as by-products from lead smelting and refining. Among these may be mentioned platinum, palladium, antimony, nickel, bismuth, tellurium, cadmium and arsenic. Some of these may not be of immediate commercial importance, and their cost may exceed their present value. On the whole, however, the indications are that the present policy of technical research will result in substantial economies in lead smelting, even if no radical changes in practice are evolved.

Developments in the Metallurgy of Copper.

Progress in the metallurgy of copper during the past year has been in the line of more careful dressing of ores, improvement in the basic-converting practice, and in the treatment of tailings.

In the dressing of ores much work has been done in the selection of the best type of machine for crushing jig middlings, the general practice being to reduce such material in either Huntington or Chilian mills. One company in Arizona, after a very extensive series of tests, has decided in favor of the use of Hardinge mills for this purpose.

Much work is being done in an effort to secure a suitable device for concentration of finely divided ore. The Wilfley

multiple-deck concentrating machine has been adopted in several plants. The Peck centrifugal slime concentrator has not made the success that was predicted for it, the difficulty being not in any defect in the principle of separation, but in the present inability to secure a machine that will "stand up" under the severe strain put upon it by the high velocity required for successful separation. A desliming device using the centrifugal principle, separating clayey material from finely divided silicious and sulphide material, makes a concentration of the sulphides practicable.

It is interesting to note in this connection the renewed use of the old round-table principle for the concentration of such fine material. The new tables are of a very modern type of construction, being made with frames of structural steel and in units of four decks.

Tailings from concentration of sulphide copper ores contain as a rule from one-half per cent to one per cent copper. Sooner or later efforts will be made for the recovery of the metal from this low-grade material. For many years, by leaching the old mine-dumps about Butte, many thousand pounds of copper have been recovered. This process, however, is very slow and requires for the best results a slightly acid water. The ferric sulphate usually present also is a valuable factor. Then, too, the mine-dump material is far more porous than concentrator tailings would be, the latter comprising material usually finer than one millimeter.

At one large plant a roasting-leaching method is being tried on a scale large enough to demonstrate its commercial value. The method consists in calcination in a McDougal furnace for the production of sulphate and oxide of copper, leaching this product with dilute sulphuric acid, and precipitating the copper by hydrogen sulphide formed by the action of sulphuric acid on low-grade matte.

The Hybinette process for leaching copper ores has been successfully applied in Norway. Ores which formerly were shipped to Sweden and to Germany are now treated locally. The ores are calcined with from 2 to 20 per cent sodium sulphate, the calcine is leached with dilute sulphuric acid and the copper electrolytically deposited from solution, insoluble anodes being used.

Wet methods have been in use many years, but without commercial success except in isolated cases. Progress such as that noted above give occasion for the hope that the many difficulties encountered by these processes may be overcome in the near future.

Only one or two plants in this country are now converting copper matte in acid-lined converters. Converting in basic lined converters has proved far cheaper, and, because of the much longer campaign made possible by the long life of basic linings, the capacity of a converting department is very greatly increased.

Duties on a single basic lining, in a converter approximately eight feet in diameter and twelve feet long, have been recorded as high as 20,000 tons, with 40 per cent matte. When these same converters were used with acid lining the average duty per lining was only about fifty tons. Steps are now being taken for the use of much larger converters. A converter recently put into use is 20 feet in diameter and has a capacity of over three hundred

tons of matte per day. What the duty per lining will be is hard to conjecture.

The thio-gen process for neutralizing smelter gases, by reduction of the oxides of sulphur with petroleum, aims at the root of the smoke problem, that is, the elimination of not only sulphur trioxide, which is accomplished by the Sprague and Cottrell methods, but also of sulphur dioxide, which is probably the principal cause of injury to plant life, particularly to conifers. If Mr. Young, who devised this process, proves it to be commercially applicable, he will have solved a very troublesome problem. Extensive tests were made in California during the past summer, but the results have not been made public, and so far as can be gathered there is still work to be done before the process can be considered a commercial success.

The Metallurgy of Zinc in 1912.

The situation with regard to the production of spelter has not changed greatly in the past year. In discussing processes for winning zinc from its ores, it must be kept always in mind that an ore-body of large size, such as the Butte and Superior, which is reported to contain 3,000,000 tons of 20 per cent zinc ore, can stand a great deal of costly experimenting, and a very expensive metallurgical plant, such as could not for a moment be considered for small mines.

It is safe to say that there is but one body of 15 per cent zinc ore in the United States with over a million tons actually in sight. There are not to exceed three with half a million tons each, and not to exceed eight with over one hundred thousand tons developed.

With large compact bodies of ore in this country more attention undoubtedly would be given to primary rather than secondary metallurgical processes, the aim being to produce spelter at the mine. There are so far but three primary methods to consider: retort-smelting, electric-smelting and electrolysis.

Practically all spelter now produced is made in retorts. That strong and well informed spelter manufacturers believe in the permanency of this condition is evidenced by their continued heavy investments in permanent smelting plants in the coal fields of Illinois. It is true that there have been two strong motives. First, the failure of the Kansas natural-gas supply, and, second, the possibility of marketing acid in the Central States; but permanent smelting and acid-making plants such as these are expensive affairs, and it may be taken for granted that the well-posted builders thereof have not made these investments without a most careful and thorough investigation of other possible methods. This is not the place to discuss the details of improvements in retort smelting. They are substantial but not radical.

Electric smelting of zinc ore is now being exploited on a commercial scale and at considerable cost in at least one plant. The information coming out from this side does not as yet offer much encouragement, although there is always hope for the electric furnace when its achievements in other lines are considered. Several of the best qualified experts in electric zinc smelting are continuing their development work quietly on an experimental yet gradually

increasing scale and many of the early serious technical difficulties have either been solved or brought near a solution. It is a slow, but apparently safe and sane development.

At least four serious attempts are being made by promoters of electrolytic processes to produce spelter commercially. In each case the electrolyte is a sulphate solution and insoluble anodes are used. With power at \$25 per horse-power-year or less, and with cheap fuel available, these processes can make a very high-grade spelter at a direct operating cost which compares favorably with that of retort smelting. The spelter produced is 99.9 per cent pure, which is one of the points strongly urged, since such material commands a premium in the market; but with any considerable tonnage of such material offered regularly the premium would be quite certain to disappear. Cheap fuel for roasting and cheap power for electrolysis are essentials, and they are not often found together in the immediate neighborhood of Western zinc deposits. A rather more serious objection to the processes is the heavy cost of plant, which is from two to three times that of a retort smelter of equal capacity. This probably limits the use of electrolysis to large ore bodies or to large custom works centrally located, where the plant cost can be distributed over a tonnage large enough not to raise total costs unduly. Perhaps the most attractive feature of these processes is the fact that a very high recovery of the gold, silver, copper and lead is made in a concentrated by-product suitable for subsequent lead or copper smelting at small cost. On ores rich enough in these other metals this recovery seems sufficiently important to give electrolysis a strong advantage. The coming year should add greatly to our positive knowledge of this subject, since thorough trials under the best conditions are quite certain to be made.

Considerable of the work done by the secondary, preliminary or preparatory processes brings us to the conclusion that here the small mine owner has apparently a much better chance. He finds such methods wasteful, yet profitable. Wet concentration is, of course, the most common of them, and needs no description. The progress of the year seems to be toward improved tables, better classification and large slime plants. Like the improvements in smelting these are substantial rather than radical.

Flotation is without doubt as well adapted to many Western ores as it is to those of Butte or Broken Hill, and will come into wide use as results become better known. The method has the advantage that it can be applied by the owner of the smaller mines.

Dry concentration has had a quiet year, but at least two new attempts will be made in 1913 on zinc ores; one in Colorado, the other in Arizona. The Colorado attempt will use the McKesson screenless sizer in the effort to get proper table feed, followed by Sutton, Steele & Steele tables. The Arizona attempt is not yet very fully worked out in detail. Several promising dry concentrating machines are being developed, but the difficulties in the past have not been so much that the jigs or tables failed to make products as that the machinery and methods for the preparation of the dry ores and handling of dust were not by any means satisfactory.

Magnetic and electrostatic separation continue to hold their own. Their field is comparatively narrow, being confined largely to the separation of certain sorts of mixed concentrates or middlings made by wet concentration methods, but this field they fill well. An interesting achievement is the success of the Huff electrostatic separator at Sunnyside mine, Eureka, Col., where 80 per cent of the material separated passes a 200-mesh screen, yet clean products are made at good capacity.

Leaching processes aiming to produce zinc oxide either for direct consumption as such, or for later conversion into spelter, are attractive on paper, but are no more successful metallurgically and financially than the hydrometallurgical processes for copper have been. If we had no other methods of winning zinc they could and would be used, but so far they have proved too expensive and troublesome.

At least two new proposals have been made during the year to extract the zinc as oxide from the slags of the American lead and copper furnaces. Results cannot be said to be satisfactory, but since such methods have been successful abroad it seems likely the troubles here are economic rather than metallurgical.

Preliminary fire concentration is being increasingly discussed and investigated in the West, particularly for the carbonate ores. There are places where fuel, labor and transportation conditions should apparently allow a low-grade zinc carbonate to be burned on grates to oxide, at a profit; such oxide to be shipped East for conversion into spelter. But here again the fact confronts the investor that such a plant could only succeed where backed by large and reasonably homogeneous ore bodies.

Just a Word to Promoters and Investors.

What Dr. J. E. Teeple says elsewhere in this issue on the necessity of early expert advice in the field of wood waste utilization, is true and timely and important far beyond the limits of that particular industry. It is a significant fact of the times that promoters and investors alike are beginning to think of chemical and metallurgical processes, besides mines, gas plants, electric lighting plants and trolley lines. It may have been simply a matter of taste which has invited this change of subjects or the public service corporations may have become troublesome. But whatever the reason, the fact is as stated.

This augurs well for the near future of chemical and metallurgical industries in this country. Both the honest promoter and intelligent investor are needed and needed badly. And the enthusiasm with which promoters and investors approach chemistry is delightful. But it is also dangerous. Too much is too often hoped for. As a single instance, plain miracles are sometimes expected by honest people from the electric furnace, like reduction without a reducing agent, etc.

A first advice to promoters and investors in their first venture in chemistry is, therefore, needed, and Dr. Teeple's article in this issue fills this want admirably. The quintessence of this first advice is that in order to promote a new chemical process to industrial life the first man to be seen is not a lawyer or a politician, but the very best technical expert who can be found.

Readers' Views and Comments

Sir Robert Hadfield on the Status of the Metallurgical Engineer.

To the Editor of Metallurgical and Chemical Engineering.

SIR:—With reference to the editorial in your November issue on "The Metallurgist in Steel," I very heartily agree with your conclusions, upon which may I be allowed to make a few remarks.

It has in the past been a puzzle to me to know why the "metallurgist" has not held a higher and better position in the scientific world than has apparently been the case. It has always been my object and aim to do my small share in raising his status. Is not the work done by the metallurgist even more valuable to the world than that done by any other profession from the point of view of adding to material comfort and convenience? Take away his work, and where, from the modern point of view, would those who live in the world be?

Startling as it may seem, the whole ground work of modern progress to-day turns upon the work done by the metallurgist. The mechanical and electrical engineer may design, but they one and all eventually have to come back to the men who produce material that is capable of meeting the required conditions of modern civilization.

In my presidential address to the Iron and Steel Institute I strongly emphasized this. In this connection I had some interesting correspondence with the able president of the American Institute of Electrical Engineers, Mr. Gano S. Dunn, who in his presidential address on the Relation of Electrical Engineering to Other Professions mentioned various classes of "human engineering," but omitted the metallurgical engineer. When I wrote him about this he answered that if he had had before him the facts mentioned in my presidential address, the metallurgist would have crowded out one of the other varieties of engineers. I take this as no little compliment, while at the same time being sorry for that class of engineer who was to be removed from his niche in fame. However, as his status is not mentioned there need be no broken hearts. We can each select the one who has least favor in our eyes. I suppose many of us would select the plumber.

Let me conclude by again saying how exceedingly valuable and just, so it seems to me, is your editorial on "The Metallurgist in Steel." It is most satisfactory to find your paper put forward such views on this subject.

London, England.

R. A. HADFIELD.

* * *

The Electric Steel Furnace and the Use of Titanium in Foundry Practice.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—I just have read the communication of Mr. Petinot in your November issue with reference to my article on "The Electric Steel Furnace in Foundry Practice." His main objection is due to a typographical error in the original article. Instead of "lost heats" one should read "lost heads," or better, "sink heads," that is to say, the part of the cast piece destined to prevent the formation of sinking holes by feeding the piece with steel still in liquid condition.

This answers Mr. Petinot's objections to the working conditions of electric steel furnaces, but his remarks suggest some more observations.

The electric steel furnace permits one to obtain perfectly deoxidized castings of higher quality than those made in ordinary open hearth or converter practice. This fact is at present agreed to by all metallurgists and is proven by the continually increasing number of foundries adopting the electric steel furnace for the manufacture of high-grade steel castings.

Mr. Petinot concludes from his experiments with several methods of heat in an electric furnace that the deoxidation by calcium carbide in the slag is not always perfectly well ob-

tained. I do not doubt his experience, but I beg to state that in the electro-metallurgical steel industry the earlier theories or views as to the exact action of the slag and the effects of calcium carbide in the slag are to-day already subject to criticism. Here I simply want to state that the operation of electric steel furnaces up to 12 tons capacity with many thousands of heats in foundry practice for production of steel castings of the highest quality as well as for all sorts of fine steels has proven that the electric furnace is exceedingly well suited for the turning out of first-class deoxidized material.

Mr. Petinot does not agree with me as to the usefulness of silicon-titanium alloy like that indicated by me. I am well aware of the fact that the great development of the use of titanium in steel practice is due to American metallurgy and I fully agree with Mr. Petinot that titanium has an excellent effect in cleaning the bath from impurities. But I cannot share his opinion regarding the addition of silicon-titanium compared to that of ferro-titanium and ferro-silicon, and without looking for a theoretical explanation, I beg to state that my works have decided to manufacture and to use the former complex alloy after a series of systematical experiments in foundry practice.

Silicon-titanium alloy is practically free from carbon and of sufficient density to be added in the electric furnace before the tapping and without remaining suspended in the slag. The silicon combined to the titanium in the form of the said alloy facilitates the formation, in the bath, of a slag of very fluid condition and separating easily.

In all our experiments with the electric steel furnace silicon has always been found to be very low, and with the exception of some qualities of special steels, it is always necessary to add silicon to the bath. It seems that actual electric steel furnace practice—as far as I know—is now sufficiently developed to do away with the fear of any influence of an acid furnace cover on the silicon contents of the bath.

Ugine, France.

PAUL GIROD.

* * *

The Huntington-Heberlein Process.

To the Editor of Metallurgical and Chemical Engineering:

SIR: Having come back from a three months' trip through the States and Mexico, I now have found time to read your extra edition reporting the proceedings of the Eighth International Congress of Applied Chemistry, in which you give an extract of the most important papers read and of the discussions which took place.

I find that the statement of certain facts regarding the so-called Huntington-Heberlein process may cause a wrong impression of the merits of this process as compared with the Dwight-Lloyd process (see page 596, Vol. X., No. 9 A.).

What I intended to state and did state after Mr. Dwight's very explicit speech on the progress of his process was that the Dwight-Lloyd machine giving a roasted product of a relatively thin stratum, say 4 to 5 in. thick, did not cause further cost of labor for breaking, and for this reason in connection with lead ores also offered an advantage from a sanitary point of view.

I mentioned that during the last years, of course, Mr. Huntington and myself had also further developed and improved the Huntington-Heberlein process so as to treat up to 40 and more tons of material in 24 hours in a single apparatus, and that by wetting the roasted product and letting it fall from a higher level the inconvenience of heavier expenses for breaking the cakes and of the development of obnoxious dust was greatly diminished.

Regarding the physical quality of the product I did not and cannot admit the superiority of the Dwight-Lloyd product over the Huntington-Heberlein roasted agglomerate.

Frankfurt a.-M., Germany.

F. HEBERLEIN

The Rôle That Zinc Ferrites Play in Hydrometallurgy.

To the Editor of Metallurgical & Chemical Engineering:

SIR: At the present time there are many attempts being made to treat complex zinc sulfides by chemical leaching and subsequent precipitation. These attempts can be divided into two classes.

The first class used chlorine gas and makes a water soluble chloridized product. The second class avails itself of the familiar metallurgical roasting operation and then leaches with sulfuric acid or sulfurous or some caustic solution. Most of these proposals of either class use electrolytic decomposition of the solution for the preparation of the metal.

Whether or no the felicitous phrase "damn a wet process" and so consigning to the infernal abysses these numerous sincere and industrious strivings to increase our natural mineral wealth by applying the methods of the chemical laboratory to the works can be legitimately applied to the hydrometallurgical trials is, of course, a question to which the future alone can give a pragmatic answer. Possibly time will prove the contention of the advocates of the electric zinc furnace that it has certain innate metallurgical points of superiority over leaching apparatus. But, be that as it may, the great activity of experimenters in this country, Canada and England demand respect from all metallurgical engineers.

The question of roasting of complex zinc sulfides has a most direct and practical bearing on the hydrometallurgy of any wet process. The "old chloridizing" or sulfating roast with or without mixtures of salt or crude acid sodium sulfate has been known in the old Henderson process, in the Ziervogel process, and in the hyposulfite leaching of chloridized silver ores. In any process that depends on desulfurizing partially or completely with the idea of making a product soluble in any reagent, the physics of the process and the changes in the physical microstructure vitally affect the ease of solubility of the product. Indeed we are here in the borderland of physics and chemistry and the problem can be stated truly to be one of physical chemistry or of "physical metallurgy."

In heating up any stable substance such as fireclay, it assumes, according to temperature and time, a denser and harder structure or in short the play of atomic and molecular forces results in a configuration of braced and interlocked dynamism. This can be true either of the intra-molecular stresses or of the extra-molecular stresses. At any rate as a matter of practical fact the soluble quality of the roasted product will vary greatly and on holding a uniform quality it will be found that the whole question of practical success of the process will hang.

One of the chief of these chemical compounds that can be definitely isolated is zinc ferrite, $\text{ZnO} \cdot \text{Fe}_2\text{O}_3$ or $\text{Zn} \cdot \text{Fe}_2\text{O}_3$. This may be formed in the wet way or it can be formed in the dry way. In the wet way it is probably formed by the double precipitation of the oxides of iron and zinc out of a strongly ammonium chloride solution and drying in vacuo. Franklinite, a well-known ore of zinc is a zinc ferrite with part of the iron and zinc replaced by manganese. In the dry way it is simply made by heating an intimate mixture of zinc oxide and iron oxide together.

The writer found in 1903 that the solubility of this product in such reagents as ammonium chloride, ammonia, sulfuric acid, diminished greatly as the time and temperature of heating and the rate of cooling varied. Indeed by heating at about 1300 deg. C. for 12 hours I made a hard dense zinc ferrite that was practically inert chemically. I lay stress on the factor of time as affecting the solubility.

Now zinc blende which is the raw material which all of these processes use is always associated with some mineral form of iron sulfide. In the roasting of such ores preparatory to leaching the tendency is to make zinc ferrite and the more this tendency is allowed to exert itself the greater will be the difficulty of solution.

In a recent meeting of the Faraday Society there was a symposium on various phases of magnetism. In one of the

papers the author, Prof. E. Wedekind, pointed out how different salts where ferric oxide played the rôle of an acid, such as zinc ferrite and of the type $\text{M}_2\text{Fe}_2\text{O}_4$ like magnetic iron oxide or iron ferrite (Fe_2O_3 or ferrous ferric oxide) had certain marked ferro-magnetic properties. A long series of such salts as cobalt ferrite, calcium ferrite were given.

It is a long way from Broadway to the copper river district of Alaska, but modern commerce is so related. It seems a long way in the physical chemical world between the magnetic qualities of iron compounds to the hydrometallurgy of zinc. But the case of zinc ferrite suggests the possibility that calcium ferrite and numerous other ferrites are formed in roasting.

It seems a positive dictate of both experimental and theoretical chemistry that successful leaching of zinc ores can only be obtained by a careful investigation of this rather new field of physical chemistry—the formation of new insoluble compounds in the dry way. What little is known about it almost haunts one with the idea of what might be known of the subject—of the move of heaven and hell—than is dreamt of in our philosophy.

Hartford, Conn.

WOOLSEY MCA. JOHNSON.

* * *

Code of Ethics of American Institute of Chemical Engineers.

To the Editor of Metallurgical & Chemical Engineering.

SIR:—In your December issue you published a letter from Dr. Joseph W. Richards regarding the Code of Ethics which was then before the American Institute of Chemical Engineers, and which since then has been adopted by the Institute. I was chairman of the committee that drew up the Code of Ethics, and Dr. Richards attended one of the meetings. At that meeting we tried to meet most of the objections which Professor Richards raised. The code having been adopted, it is perfectly proper for Professor Richards to advocate in a proper manner and place a reconsideration of the action by the Institute or to amend the code in such a way as seems desirable. I would like, however, to call attention to the following points:

It is a great pleasure for the members of the Institute to realize that Professor Richards does not require a Code of Ethics. It is to be hoped that no member of the Institute requires a code, but that is not the important question.

The important question is: Will a Code of Ethics be of assistance to the members of the Institute? Will it help to build up a higher ethical standard among them, and will it help to build up in the mind of the public a better appreciation of the moral excellence of the members of the Institute and the profession generally?

Some of the features of the code are undoubtedly unnecessary except as being a part of the logical presentation and development of the code. In such a presentation we naturally start with axioms which might appear gratuitous. As we read over the features of the code, it seems that the code offers valuable suggestions of methods of building up the profession. It urges mutual helpfulness and a general attitude of defense of chemists. It declares against sensationalism in advertising and against unreasonably low charges for professional work. It emphasizes the desirability of chemists giving credit to others wherever such credit is due. These and many other points are positive in their suggestiveness. Is any of us so good that he can overlook or ignore these positive suggestions? Are we all of us so helpful to our fellow chemists; do we always so avoid sensationalism; do we always so give credit to others for professional work, that it is unnecessary to be reminded on these points?

The Code of Ethics of the American Institute of Chemical Engineers gives backbone to our moral purpose. We can hardly conceive of it as limiting anyone's proper freedom of action. The most pronounced individualist will be as free under the code as he would be without it; but it adds to

such freedom a recognition of the profession as something of an entity—something that we should try to improve and exalt. I am confident that many members of the Institute joined it because they believed that it would raise, both ethically and intellectually, the standards of the chemical profession and believed that the indirect benefits derived from the Institute would be of infinite value.

The adoption of the Code of Ethics puts an emphasis on the ethical phase just as its educational requirements put an emphasis on the intellectual phase. Both are important parts of its reason for existence—neither should be ignored.

Brooklyn, N. Y.

G. W. THOMPSON.

Perkin Medal Awarded to James Gayley.

At the regular annual meeting of the Perkin Medal Committee the medal was awarded for 1913 by a unanimous vote to Mr. James Gayley for his dry-blast process in pig iron manufacture.

The Perkin Medal Committee consists of delegates of the Society of Chemical Industry, the American Chemical Society, the American Electrochemical Society, and of their New York Sections, and recommendations are made by the various national chemical societies of this country. The presentation of the Perkin Medal to Mr. Gayley will be made at the regular January meeting of the New York Section of the Society of Chemical Industry.

Electric Steel Furnaces.

As an indication of the present status of the electric furnace in the steel industry the following list of licenses of the Girod furnace will be found interesting:

Rubery Owen & Company, Darlaston, England; Gib. Ansaldo & Company, Genoa, Italy; Putiloff Works, St. Petersburg, Russia; Bethlehem Steel Company, South Bethlehem, Pa.; Krupp & Company, Essen, Germany; Oehler & Company, Aarau, Switzerland; Société John Cockerill, Seraing, Belgium; Stotz & Company, Kornwestheim, near Stuttgart, Germany; Stahlwerk Becker, Krefeld, Germany; Gutehoffnungshütte, Oberhausen, Germany; Ternitzer Stahlwerk, Schöeller, Ternitz, Austria; Danner & Company, Judenburg, Austria; Ungarisches Staats-Stahlwerk, Diosgyor, Hungary; Simonds Manufacturing Company, Lockport, N. Y., and two other American works whose name is not divulged at this time. One of these American works will install a 3-ton Girod furnace to be used exclusively in the manufacture of steel castings.

Thirty-Third Annual Report of the Director of the United States Geological Survey.

In the advance sheets of the director's report for 1912, Mr. George Otis Smith calls special attention to the need of a new building for the Survey. In considering the general unfitness of the buildings now occupied by the Survey, the director finds that they are inadequate from the point of view of the health and efficiency of the employees, as well as offering great risk from fire. The offices are crowded, and in the summer season some of them are too hot for occupancy. There is lack of fresh air and light, and too much noise and dirt from without; and the prospect of increasing the efficiency of the Survey under present conditions is remote.

The risk from fire is a menace second only to the existing danger to the health of the employees. The Survey's papers, records, maps, reports, etc., have an inventory value of \$4,840,000. If these were lost, it would be impossible to replace them without incurring an expense equal to their present value, and even then it would require observations extending over long periods of time to replace some of them. The director urges the need of adequate quarters, and states that plans for a building have been drawn and referred to the House committee on public buildings and grounds, of the Sixty-second Congress.

Oxygen from Liquid Air.

The Linde Air Products Company of Ohio has commenced the installation of Plant 5 in Detroit.

Preliminary work is now complete for the establishment of Plant 6 which will go forward at once in Boston. The new Boston plant will have a capacity of 15,000,000 cu. ft. of oxygen per annum and will involve an investment of approximately \$500,000.

The Linde Company is expanding its capacity in advance of requirements with a view to supplying local needs as rapidly as they develop at so low a price that the largest possible industrial use of oxygen will be made probable.

The use of commercial oxygen is strictly limited by small differences in selling price and the business must necessarily depend upon a very large output with a rather small margin of profit.

This calls for large investments and the Linde Company has the courage of its convictions.

Transactions of Eighth International Congress of Applied Chemistry.

There seems to be some misunderstanding among members of the Eighth International Congress of Applied Chemistry as to its printed report. This report will consist of 26 volumes of original communications, one volume of discussions, one volume on the Congress meeting itself and an index, or a total of 29 volumes or about 6500 pages. Prior to the Congress 24 of the 26 volumes of Original Communications had been printed; the two additional volumes of Original Communications have since been set up in type and the manuscript for the discussions and for the volume on the Congress itself has been practically completed since November 4, 1912, but is still awaiting manuscript from a few belated but important contributors, both here and abroad; the index cannot be prepared until all matter is in type and ready for printing, but the most of the work for that is finished for the first 26 volumes.

The date originally set for delivery of the entire Report was December 31, 1912. In view of the delay above described delivery to transportation companies cannot begin until toward the end of February, 1913; this will be automatic and will not be hastened by special inquiry; it is therefore requested that such inquiries be not made.

Bound volumes will not be delivered until the end of May, 1913; orders for bound volumes will not now be accepted. Members will be advised by letter at their last known address of the time their volumes left the Rumford Press at Concord, N. H., and by what transportation line. All inquiries as to reprints, corrections in papers appearing in volumes 1-24, imperfect volumes or sets and lost shipments, as well as all statements of change of address should be addressed to the Rumford Press, Concord, N. H.

The office of the Secretary of the Congress has been removed to 90 William Street, New York City, and all other communications should be so addressed.

Patent Reform.

Resolutions of American Institute of Electrical Engineers.

The Board of Directors of the American Institute of Electrical Engineers has adopted the following resolutions:

"Whereas, there are pending before the Congress numerous bills affecting and greatly modifying the Patent System in the United States, and

"Whereas, the Patent System has been, and is, a tremendous factor in building up the present industrial prosperity of this country, thereby greatly contributing to the prosperity of the country as a whole, and

"Whereas, any untoward change in the patent situation might disastrously affect this condition of industrial and general prosperity, and the conditions contributing to their continual augmentation, and

"Whereas, in view of the intimate relation of the Patent System to the general welfare, no action looking toward any radical change in the Patent System should be taken without most careful consideration, and

"Whereas, in our opinion, proper consideration of such important changes as are proposed can be had only by an unbiased, non-partisan commission, made up of men from various walks of life and not from any one vocation, or interest,

"Be It Resolved, that the American Institute of Electrical Engineers, acting through its officers and board of directors, respectfully urge the Congress of the United States that they provide for a Commission, made up of unbiased, independent, non-partisan men of such national standing as will command the respect of the whole country; and chosen from different walks of life; and not more than one from any one calling or interest; and serving without pay. Such Commission to hold public hearings, and otherwise, as may appear to them best, to make a thorough and careful study of the American Patent situation, and to prepare and submit a comprehensive report and recommendations to Congress for such changes, if any, as may, as the result of their study, appear to them expedient, whether in the Patent Office, in the method of court procedure, or in the organic Patent Law, and recommendations as to the Legislation they would propose for effecting said changes. And that we further respectfully urge that the Congress make ample provision for the expenses of said Commission, and

"Be It Resolved, that we respectfully urge the Congress of the United States to hold in abeyance all proposed Legislation affecting the Patent System in whatsoever way until such time as the said Commission shall have had ample opportunity to hold the said hearings, and make the said study and report, and

"Be It Further Resolved, that these resolutions be printed and a copy be sent to each Senator and Representative of the United States who is a member of the Senate or House Committee on Patents."

Mr. Ralph D. Mershon is the president and Mr. F. L. Hutchinson is the secretary of the American Institute of Electrical Engineers.

The Iron and Steel Market.

The slowing down which usually occurs in the iron and steel market at the close of the year was observed in December, but was less marked than usual in that the pressure for deliveries continued heavy. Buyers were as insistent as ever in urging the mills to better deliveries. In forward buying there was a notable decrease, except as to railroad buying, which continued heavy. In specifications against contracts already placed the month made a good showing, though the total tonnage specified was not as great as that of November, the decrease being probably due to the fact that some contracts, nominally written to last to December 31, had already been worked out.

There were no important price changes during the month. Effective December 16 the American Steel & Wire Company advanced its prices on wire products \$1 per ton, putting nails on the basis of \$1.75 per keg. Such an advance had been made by two or three independents about the middle of October, in the expectation that a general advance would thus be precipitated.

The scarcity of crude steel became still more acute in December, a point being reached at which there was practically no quotable market. The regular large producers had long been out of the market, and only occasional lots could be picked up from brokers or from outlying mills. An Alabama interest, for instance, offered a moderate tonnage in the North for next year's delivery, and succeeded very quickly in selling on the basis of about \$20, Pittsburgh.

The famine in unfinished steel has greatly disturbed many of the independent finishing mills, particularly sheet and tin mills, which must depend upon purchased steel for their operation. The prospect is that some of them will lose time in the first quarter of the new year from not having steel. The

scarcity of unfinished steel, however, is a natural outcome of conditions. In the first place, the normal alignment of the steel industry, the alignment that has prevailed ever since the steel industry reached important proportions is that of their being an excess of finishing capacity over steel-making capacity. In the laying out of an individual plant, with diversified product, it is always regarded as desirable to have an excess of finishing capacity, for if perfect balance could be secured in mechanical operation, such balance would be impossible in the market. A mill cannot be certain of selling its output in precisely the proportions the different finishing mills would make, if operated full. It is easy to slow down or to close a given finishing mill, whereas to slow down the steel-making unit is expensive, there being little reduction in overhead. When the individual plant is naturally planned for more steel finishing than steel-making capacity, so much more is the alignment in that direction when the steel-making unit is under one ownership and the steel finishing unit is under another. When, therefore, there is a market which would involve the operation of all the finishing capacity, there is not enough steel to supply it.

Another cause of the famine in steel is the continued sharp trend in demand toward open-hearth steel. The greater portion of the finishing capacity of the country chances to be so arranged that it can roll either Bessemer or open-hearth steel, whereas there is practically no flexibility in the steel-making units. The Bessemer plants can make only Bessemer steel and the open-hearth plants only open-hearth. For the idle Bessemer plants to relieve the scarcity involves their making billets or sheet bars for the market, and this they are not all equipped to do. We said there is "practically no flexibility" in the steel-making units. There is none at the moment, but there is some by rearrangement, whereby Bessemer vessels may sometimes be made to form the first link in the practice of the duplex process, either by using open-hearth furnaces already built, or by building them. A recent instance of the latter is the erection in the past year of two 250-ton tilting furnaces by the Lackawanna Steel Company, to practice the duplex process with existing converters.

Finally, billets and sheet bars have been made scarce by the heavy demand for rails, since many of the mills are interchangeable as to billets or rails, sheet bars or rails, or the whole three products. Rails are a finished product, sold to consumers, and a producer having the choice of selling rails or sheet bars, or rails or billets, chooses the rails apart from considerations of price and cost of manufacture, regarding rather the general principle that the actual consumer, rather than a detached finishing mill, presents the trade which in the long run it proves best to serve.

Pig Iron.

The pig-iron market was practically stationary as to prices during December, and was rather slow as to transactions. There was a strong undertone to the market, which did not depend upon current demand, but rather upon the facts that the operating furnaces were already well sold up, and that on account of the very high prices ruling for coke it was difficult to start any of the idle furnaces. The only advances over quotations in last report are of 25 cents in foundry iron in the valleys and in eastern Pennsylvania. We quote: Bessemer, \$17.25; basic, \$16.50; No. 2 foundry, \$17.50 to \$18; forge, \$17 to \$17.50; malleable, \$17 to \$17.50, f.o.b. valley furnaces, 60 cents higher, delivered Pittsburgh; No. 2N foundry, delivered Philadelphia, \$18.50 to \$18.75; No. 2 foundry, f.o.b. Chicago furnaces, \$18; No. 2 foundry, Birmingham, \$14 to \$14.50.

Steel.

The market has become purely nominal, as already noted. Prices last quoted, which were based upon actual transactions, and which would probably prevail had the large mills any tonnage to dispose of, are as follows: Bessemer billets, \$27; sheet bars, \$27.50; open-hearth billets, \$28; sheet bars, \$28.50, f.o.b. maker's mill, Pittsburgh or Youngstown. Under special cir-

cumstances, sales of open-hearth billets have been made at \$29, Pittsburgh. Rods are \$30, Pittsburgh, with sales at that figure.

Finished Steel.

Regular steel prices showed no important change in December, there being simply an advance of \$1 a ton in wire products. On the majority of products premiums ruled for early deliveries, but such prices cannot be quoted, as the premiums increase the smaller the order, with no definite limit, and also increase with the nearness of delivery. In general there are many products which have commanded premiums of \$2 to \$4 a ton in carload lots for delivery within one to three weeks.

Regular prices for forward delivery are given below, prices being f.o.b. Pittsburgh unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.50 cents.

Shapes, 1.50 cents.

Steel bars, 1.40 cents.

Common iron bars, 1.65 to 1.75 cents, Pittsburgh; delivered Philadelphia, 1.67½ to 1.75 cents; f.o.b. Chicago, 1.60 to 1.65 cents.

Steel hoops, 1.50 cents.

Sheets, blue annealed, 10-gage, 1.65 to 1.75 cents; black, 28-gage, 2.25 to 2.35 cents; galvanized, 28-gage, 3.40 to 3.50 cents; painted corrugated, 28-gage, 2.45 to 2.50 cents; galvanized, 3.45 to 3.50 cents.

Tin plates, \$3.60 for 100-lb. cokes.

Merchant steel pipe, ¾ to 3-in., jobbers' carloads, 80 per cent. off list.

Steel boiler tubes, ¾ to 4½-in., 71 per cent. off list.

Standard railroad spikes, 1.85 to 1.90 cents.

Button head structural rivets, 2.20 cents; cone-head boiler rivets, 2.30 cents.

Tubular Goods Revision.

Under the lead of the National Tube Company, which on November 27 announced to its customers and competitors the revised lists it would work under beginning January 1, 1913, a general change has been made in merchant pipe and boiler tubes.

The old merchant pipe list carried the weights per foot and thicknesses of wall based upon wrought iron, and when steel was adopted, with about 2 per cent greater specific gravity, the mills departed from the standard thicknesses and adhered to the standard weights per foot. The present revision restores the old thicknesses, thus increasing the weights about 2 per cent, but the computations are more exact than formerly. Next, the list prices per foot are revised so that they conform more closely to the principle of making the list price produce a price of 10 cents per pound. Until now there have been made two weights in merchant pipe, 6-in. and under, the "card weight" conforming to the card, and "merchant weight," a weight indefinitely lighter, and sold in the market at one point discount (about \$2 per net ton) less than the price of the card weight.

The revision of weights and list prices made on the whole a slight advance in prices per foot, though a slightly greater reduction in prices per pound, and at unchanged discounts the net prices per foot would be slightly advanced. To avoid the appearance of making any profit out of the rearrangement, when it is made solely in the interest of improving the quality, the manufacturers have made the old discount on merchant weight apply to the new full weight pipe. Thus the price per foot of card weight is slightly reduced, and a buyer formerly taking merchant weight secure full weight at a very slight advance compared with the much greater weight furnished.

The boiler tube list was somewhat similarly revised, but as list prices were somewhat reduced and as contracts were on back orders well into the new year, the discounts were reduced at one point, thus obtaining a slight advance.

The Non-Ferrous Metal Market.

General dullness has characterized the non-ferrous metal market for December. Copper has been stagnant, lead has been greatly reduced in price, tin is offered freely without premium for immediate delivery, and spelter has been offered liberally at low prices without finding buyers.

Copper.—Since our last report Lake copper has had only a nominal price, and little business has been transacted. The principal sellers have maintained the price, and outsiders have sold the metal at concessions. Moderate sales have been made in electrolytic. The impression prevails that consumers cannot much longer stay out of the market. The latest quotations are: Lake, 17½@17¾ cents; electrolytic, 17.35@17.40 cents.

Lead.—Twice since our last report the principal producer cut the price, first to 4.50 and later to 4.35 cents. In each case the cut was met by other sellers, who reduced their prices correspondingly. In spite of the low prices prevailing, little business has been transacted. The metal is quoted at 4.15@4.17½ cents, St. Louis, and 4.30@4.35 cents New York.

Tin.—Lower prices prevailed in the London market, and this developed a fair business with American consumers. The premium on spot tin has disappeared, and December tin was quoted at 49¼ cents.

Spelter.—Prices have continued to decline, and some business has been effected at 7.10 cents. The prospect of importations of foreign spelter has affected the domestic market, and removed the customary differential between New York and St. Louis markets. The latest quotations are 7.05@7.10 cents, St. Louis, and 7.10@7.20 cents New York.

Notes.

Metal shipments from Nome, Alaska, for the navigation season of 1912 have been made public by R. W. J. Reed, deputy collector of customs. Placer tin mined in the vicinity of Nome, 173 tons; placer tin shipped, 129 tons; gold dust and bullion shipped, 160,079.69 oz. valued at \$2,931,478.26. The gold output for the season probably will be increased by about \$100,000 by dredges operating during November.

Pebbles for the Hardinge regrinding mills at the Tomboy mill, Telluride, Colo., are prepared at the mill from selected pieces of mine rock composed of rhodonite and quartz. The rock is broken into rough spherical shapes and charged into the mills at the rate of about 170 lb. per day. Some of the worn pebbles show ridges of the hard quartz above the general surface of the rhodonite. The pebbles give satisfaction, and are used in preference to imported pebbles which would cost about \$28 per ton delivered at the mill.

The Bureau of Mines has recently issued the following publications of interest to mining and metallurgical engineers: *Technical Papers*, No. 22, Electrical Symbols for Mine Maps; No. 24, Mine Fires; No. 25, Determination of Water in Petroleum and its Products; No. 26, Determination of the Sulphur Content of Fuels, Especially Petroleum Products; No. 28, Ignition of Gas by Standard Incandescent Lamps; *Bulletins*, No. 43, Fuel Values of Gasoline and Denatured Alcohol for Internal Combustion Engines; No. 44, First Mine-Safety Demonstration; No. 45, Investigation of Explosion-Proof Motors; No. 47, Notes on Mineral Wastes. These publications can be had on application to the Director, Washington, D. C.

Timber Preservation at the Homestake.—The cats and tanks at the Homestake mill, South Dakota, are of California redwood, with the exception of a few thickening tanks which are of cypress. They are painted on the inside with P. & B. paint, and on the outside with a mineral paint containing more or less red oxide of iron. A timber-treating plant is installed for impregnating wood with creosote or dead oil of coal tar. The usual treatment is a hot bath of two hours' duration at a temperature averaging 216 deg. Fahr., followed by a half-hour contact at 180 deg. Fahr. or 190 deg. Fahr. The cost is about ten cents per cubic foot. Railroad ties and 2-in. plank for mortar blocks are thus treated.

Notes on Chemistry and Metallurgy in Great Britain.

(By Our Special Correspondent)

The Iron Market.

The effect of the steadily increasing demand for iron, both at home and abroad, which set in before the middle of this year, has been to send up prices to figures which have not been reached since 1900; and there is every probability that the rise will continue, because the demand seems likely to become still greater. Many manufacturers are showing disinclination to make further purchases of pig-iron at the present high rates, whilst the pig-iron makers assert that the price of manufactured articles has risen proportionately higher than that of the raw material—a contention which certainly appears to be supported by a study of the quotations during the last few months; and, in any case, the advance unquestionably began with manufactured products before the price of pig-iron increased.

At the close of 1911 Cleveland warrants stood at 50s. 10d., and the stock at Middlesbrough amounted to rather more than half a million tons, whereas at the end of last October it was only slightly over a quarter of a million tons, and the quotations touched 67s. Since the coal strike both coal and ore have increased in cost, and the replenishment of the stock depleted as a consequence of the strike could only be possible at greater cost; but had there not been any coal strike the increased activity in the iron trade would have been felt only a little earlier, and an uninterrupted production of pig-iron would not have materially affected the advance in price during the last three months or even sooner.

There are now very few idle furnaces in England or Scotland—although the total number now in blast is only some six more than at the end of last year—but their output is inadequate to meet the requirements of the iron and steel works, which are in turn unable to keep pace with the demands of the consumers. It is not magnifying the truth to say that every iron and steel works is being run at the fullest capacity; many works are being extended to cope with the increasing influx of orders; and it is anticipated that unless the supply of raw iron can be considerably augmented the inquiry for the home market will, at no very distant date, be such that it will absorb almost the whole of our production and very largely reduce the amount available for export when still heavier shipments than are being made now will in all likelihood be called for.

The question is how the increased consumption of pig-iron is to be met soon enough. The output from the furnaces now in operation cannot be enlarged to any great extent, and the provision of new ones in sufficient numbers is not practicable in a few weeks, while the localities in which electric smelting could be quickly put into practice are very few and even in these the working costs would be prohibitive for iron ore reduction in England. The old proverb, "No pleasure without pain," is emphasized by the embarrassing situation resulting from a most gratifying revival of the iron trade.

Poisonous Gases from Oilfields.

A paper on this subject was read by Mr. H. S. Shrewsbury before the Society of Public Analysts at the meeting on Nov. 6. Recently a negro was killed by breathing the gas at the bottom of a pit under an oil derrick close to the Pitch Lake at La Brea, Trinidad. The pit was some 10 ft. deep and surrounded the casing of the tubular oil well, from which the gas most probably overflowed into the pit. While working in the pit the man became giddy and had to be helped out. Twelve hours later he was dead.

A sample of gas taken from the top of the oil tube was analyzed by the author, who found that its composition was: sulphuretted hydrogen, 0.1 per cent; carbon monoxide, 1.1; saturated hydrocarbons, 4.4; unsaturated hydrocarbons, 2.5; oxygen, 8.7; carbon dioxide, 11.9; hydrogen, 18.8; and nitrogen, 52.5 per cent. Assuming all the oxygen and some of the nitrogen to be derived from the air, the amount of air present was 43.4 per cent, and the composition of the gas as it issued

from the ground was: sulphuretted hydrogen, 0.2 per cent; carbon monoxide, 1.9; unsaturated hydrocarbons, 4.4; saturated hydrocarbons, 7.8; carbon-dioxide, 20.9; hydrogen, 31.4; and nitrogen, 33.4 per cent.

A Wynter Blythe states that 0.05 per cent of sulphuretted hydrogen in air is the maximum that can be breathed with safety; that 0.08 is dangerous, and 0.1 to 0.15 fatal to human life. According to Haldane, more than 0.15 per cent of carbon monoxide is dangerous, and 0.4 per cent would kill in a moderately short time. The gas as it issues from the ground is extremely poisonous; and when diluted with 43 per cent of air would kill in a short time—as indeed it did.

The author visited the Pitch Lake and observed that the surface is covered with little pools and channels of water through which gas is constantly bubbling, and gas also issues from the pitch itself. This gas has exactly the same odor as the sample described—the small percentage of sulphuretted hydrogen mixed with the hydrocarbons producing a sweet and tar-like smell. The author also witnessed the interesting fact that small fishes live in the pools through which the gas bubbles, notwithstanding its poisonous character. It appears to your correspondent that air containing 11.9 per cent of carbon dioxide is sufficiently poisonous without the CO and H₂S.

The Granular Structure in Metals.

It is a matter of common knowledge that reheating steel at temperatures below 900° C. renders the structure of the metal granular and induces brittleness. F. Robin (Comptes. Rend., 1912, 155; 716-719) states that he has found that other metals are similarly affected, and that hammering, deformation and compression likewise tend to develop a coarse, granular structure. In the case of local deformations, such as bends, the granules formed in the deformed area appear to mutually limit their development; while just outside this area their development is greater and extends gradually into the unaltered portions. When an annealed metal is locally hardened by forcible deformation, such as shearing, punching, or bending, and is re-annealed at various temperatures (presumably below the change point) for about half an hour, the invasion of the unchanged area by large granules is distinctly noticeable after five minutes. Such granules appear to be subject to much the same conditions as those produced by annealing, and they extend as far as 7 to 12 mm. into the unaltered region.

The extensive manufacture of articles from stamped, bent and perforated sheet metal invests this subject with a considerable degree of industrial importance. The remedial measures suggested by the author are, for iron, heating to above 900° C., and for non-ferrous metal, the addition of other metals.

Electrolytic Tanning.

A process of tanning by electrolysis has been devised by Dr. L. A. Groth, and a small plant for demonstration purposes on a commercial basis is being put down at Mr. C. Smith's tanyard, Kidderminster, by the Local Electric Lighting and Traction Co., under the direction of the inventor.

The time required for tanning heavy hides by the ordinary processes extends to several months; and Dr. Groth claims that by his method they can be purely and effectively tanned in six weeks, yielding a leather of very high quality.

The operation of the plant is practically automatic, and the necessary attendance throughout is furnished by two lads; whilst the cost of production is said to show a very considerable saving as compared with that of tanning by the usual processes. An ingenious system of automatic control is designed to prevent injury to the skins or the tan-liquor, and any departure from normal conditions is at once signalled to the attendant by bell and the lighting of a lamp at the spot requiring attention.

A Reinforced Concrete Water Main.

At Luton, Beds., a ferro-concrete rising main about a mile long and having an interior diameter of 27 in. has been constructed by a London firm. The steel reinforcement measures

$2\frac{1}{2}$ per cent of the volume of the concrete, which has a thickness of 3 in. and is without lining of any description. The main stood a continuous test at a pressure of 60 lb. per sq. in., and when completely filled the loss after standing twelve hours was only four gallons. As the valve at the lower end was not readily accessible and might possibly not have been quite tight, the main was reported as thoroughly water-tight. The lowest tender for cast-iron pipes of the same capacity was £2-2-7 per yard, and the cost of the concrete main, delivered at Luton, was £1-7-7 per yard.

Castner-Kellner Alkali Co.

The annual meeting of this company was held late in November in London. The chairman, Mr. G. W. Balfour, announced that the net profit on the year's working was £172,062, against £178,853 for the previous year; but the reduction was due to special circumstances. The depreciation reserve against patents, land, buildings, etc., valued at £737,000, now amounted to £322,500, out of which sum £60,000 had been invested by reducing their debentures.

The general reserve of £177,500 was invested in readily realizable securities, and could be employed in any way the directors chose. Given continued prosperity, the directors might consider the advisability of reducing the sum carried to general reserve in future and increasing the dividends. A final dividend of $1\frac{1}{2}$ per cent was agreed to, making 20 per cent for the year.

Alby Carbide.

The sixth annual general meeting of the Alby United Carbide Factories, Ltd., was held on Nov. 28 in London, under the presidency of Mr. A. E. Barton, chairman of the company, who said that the profits for the year ended June 30 showed an increase of practically 100 per cent on those for the previous year, which in turn were 58 per cent greater than in 1909-10. The balance sheet shows that the net income for the 12 months June 30, 1911-1912 was £72,702, and this sum, with £493 brought over, suffices to write off £6,514 for depreciation; £2,000, the balance of the Atkins Acetylene Dry Generating; £37,012, the entire amount of preliminary expenses, and to provide a final dividend of 3 per cent on the ordinary shares, making 6 per cent for the year, leaving £4,097 to be carried over. The dividend for the last financial year on the shares held by the company in the Meraker Company amounted to nearly twice the investment.

The present output of the Odda factory is 32,000 tons of carbide per annum, and in order to enlarge the works for turning out 80,000 tons per annum, the capital has been increased during the past year to £550,000 by issue of 150,000 $\frac{1}{2}$ per cent accumulative convertible preference shares. The company's interest in Alby Carbide Works (Sweden) has been acquired by Nitrogen Fertilizers, Ltd., for 142,380 £1 shares in that undertaking, and by the purchase of 50,000 additional shares the company now holds 192,380 shares out of a total of 220,000 in Nitrogen Fertilizers, Ltd.

Engineering Imports and Exports.

The returns issued by the Board of Trade again show substantial advances in the value of both imports and exports in the ten months ended on October 31 as compared with the same period of last year. Imports of iron and steel, including manufactures, totalled £10,489,072, an increase of £1,380,752; and exports amounted to £39,523,957, an increase of £3,719,742. Imports of other metals, including manufactures, are valued at £25,588,083, an increase of £2,582,495; whilst exports reached £10,014,596, an increase of £944,294. In electrical goods the imports are put at £1,176,199, a rise of £32,307; and exports went up to £1,715,449, an improvement of £1,458,481. Machinery was imported to the value of £5,645,823, an increase of £804,204 and was exported to the amount of £27,502,255, an increase of £2,077,878. Imports of new ships were £26,652, a fall of £4,441, but exports were £5,673,942, a rise of £1,260,879. The figures for October solely show increases all round. Imports of iron and steel were £349,706 higher and exports

were £1,022,072 better than last year; imports of other metals rose £1,082,431, and exports improved by £284,910; imports and exports of electrical goods went up £8,754 and £250,563 respectively; machinery shows increases of £219,113 in imports and £537,435 in exports; and imports of new ships rose £1,267, whilst exports were £867,310 better.

The aggregate increase in the value of the imports for the ten months is £4,798,848, and that of the exports is £9,441,265. Taking the decrease in the imports of ships as equivalent to a similar increase in exports, the net excess of increase of exports over imports is £4,676,758.

Market Prices for November.

Copper has been quiet on the whole, with tendency to rise about the middle of the month. Starting at £74.10 it had reached £77.10 by the 19th, and was over £78 on the 20th. On the 28th it only showed £77.15 and closes £76.10.0.

Tin opened at £229.5 and was fairly active, being at £229.15 at middle of month. A week later it was £227, and closes at £226.

Pig Iron (Scotch) has been quiet, with harder tendency towards the end of the month. Starting at 72/- it dipped a shilling on the 6th, recovering on the 7th. It commenced to rise on the 12th, and had reached 73/- by the 19th, and closes 73/10 $\frac{1}{2}$ d.

Cleveland has been in sympathy with Scotch pig, opening at 66/-, dropping on the 6th and rising on the 12th, it reached 67/9d. by the 28th, and closes at 67/2d.

Hematite has been slowly but steadily rising. Opening at 80/-, it reached 81/10 by the 19th, and closes at 82/-.

English Lead has fallen throughout November. Starting at £19.5, it was under £19 by the 5th, and continued to decline, reaching £18.15 on the 13th and £18.12.6 on the 21st. It closes at £18.10.0.

	£ s. d.
Aluminum, per ton.....	82 to 85. 0.0.
Alum, lump, loose, per ton.....	5.15.0.
Antimony, black sulphide powder, per ton.....	21. 0.0.
Borax, British Refined Crystal, per ton.....	17. 0.0.
Copper Ore, 10 to 25 per cent, per unit.....	14/- to 14.6
Copper Sulphate, per ton.....	25. 2.6.
Carbolic Acid, liquid 77/99 per cent, per gal.....	0. 1.5.
Creosote, ordinary good liquid, per gal.....	3.
Camphor, 1-oz. tablets.....	1.8.
Caustic Soda, Ash 48 per cent, ordinary, per ton.....	5.10.0.
Hydrochloric Acid, per cwt.....	5.0.
India Rubber, Para, fine, per lb.....	4.6.
Naphtha, solvent, 90 per cent, 160 degs. C., per gal.....	1.2.
Petroleum, Russian spot.....	8.
Quicksilver, per bottle.....	7.12.6.
Sal Ammoniac, lump, first, delivered U.K., per ton.....	44. 0.0.
Sulphate of Ammonia, f.o.b. Liverpool, per ton.....	13.17.6.
Sulphur, recovered, per ton.....	5. 5.0.
Shellac, Standard T. N. Orange Spots, per cwt.....	4. 4.0.
Tin Ore, 70 per cent, per ton.....	150 to 152. 0.0.
Zinc, Vieille Montague, per ton.....	30. 5.0.
Platinum, nominal, per oz.....	0. 5.0.

The differences on the month are thus:

Higher.	£ s. d.	Lower.	£ s. d.
Antimony	1. 0.0.	Creosote, gal.	74.
Copper Ore.....	5 $\frac{1}{2}$.	Quicksilver	7.6.
Carbolic Acid	1.	Zinc, V.M.I.	1. 5.0.
Camphor, oz.	34.	Tin	3. 5.0.
Petroleum, gal	34.	Lead	15.0.
Tin Ore	2. 0.0.		
India Rubber	2.		
Copper	1.10.0.		
Hematite	2.0.		
Scotch Pig	13 $\frac{1}{2}$.		
Cleveland	2.		
Sulph. of Ammonia	0.6.		

London, England.

The Electric Furnace in the Production of Iron from Ore

A critical discussion of the present status and a comparison of the Scandinavian and Californian practices, by D. A. Lyon

AS is well known, the electric furnace, under favorable circumstances, may be used in the iron and steel industry for the production of metal from the ore, and later for the refining of this same metal into the grade of steel desired.

Conditions Favorable to the Use of the Electric Reduction Furnace.

It was pointed out by the Canadian Commission in their Report of 1904, that the prime pre-requisite for the use of the electric furnace in the reduction of iron ores, is cheap electric power. The large amount of experimental and practical work which has been done since that time has only emphasized this fact. The electric furnace, therefore, has not been developed as a competitor of the blast furnace, but, as is the case in Norway and Sweden, for the purpose of keeping alive an industry which finds itself confronted by conditions unfavorable to a continuation of blast furnace practice, due to the increased cost of charcoal; or for the purpose of making possible the development of the iron industry, as is the case in California, where the iron industry has never been developed, due to the cost in that section of the country of both charcoal and coke.

In each country, considerable experimental work was first done and then the furnace was tried out on a commercial scale. As was to be expected, the problems which presented themselves at the outset of the work were not solved at once, and there are still problems which are awaiting a more satisfactory solution than has so far been attained, but satisfactory progress is being made in this direction. That the electric iron ore reduction furnace has entirely passed the experimental stage is evidenced by the situation in California and Scandinavia.

The Scandinavian Practice.

Due to the large number of articles descriptive of the work in Sweden, which have previously appeared, it will not be necessary in this connection to do more than to refer to them and to state briefly the status of the electric reduction furnace in Scandinavia at present. (See also Oct. issue, 1911, p. 525.)

As a result of the successful operation of the furnace at Trollhättan, three other iron electric reduction furnaces are now being operated in Sweden¹ which together with the furnace at Trollhättan represent a total of 12,000 hp that is being used for electric reduction furnace work in that country; in

Norway there is one 3500-hp furnace in operation and one 3500 hp and three 3000-hp furnaces in the course of construction, while in Switzerland one of 2500-hp is being built.

As was likewise pointed out in the September, 1912, number of this journal, the Uddleholms Company, at Hagfors, Sweden, are increasing their electric furnace installation by the addition of three more furnaces and will probably substitute electric smelting altogether for their blast furnace work. It is also a significant fact that the plant at Trollhättan, which was built by the Jern-Kontoret, of Sweden, purely for the purpose of determining the feasibility of attempting to reduce iron ores in an electric furnace, has been sold by the Jern-Kontoret to the Degerfors Iron Company, and will be worked by the latter company as a commercial proposition from now on. The Degerfors Company is an old-established company, who make very high-grade material, and so it is not a case of a new concern taking up with a new venture, but one with long experience in the business, who have had ample opportunity to carefully study the feasibility of electric reduction furnace work under the conditions prevalent in Sweden, and who, as a result of their study, have satisfied themselves that the electric furnace will meet their requirements. In fact, the old saying, "the proof of the pudding is the eating thereof," would seem to apply as regards the use of the electric reduction furnace in Sweden.

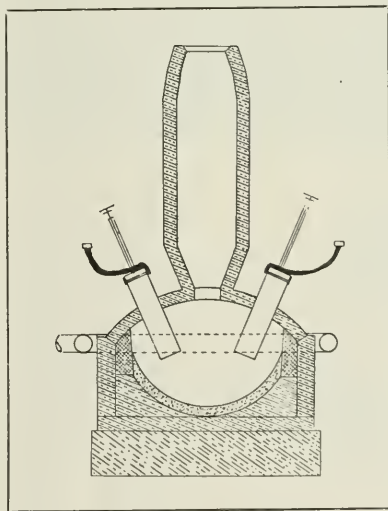


FIG. 1.—SECTIONAL ELEVATION OF TROLLHÄTTAN FURNACE.

Some of the Difficulties That Had to Be Solved.

If a study be made of the drawings of those furnaces which have been tried out experimentally, it will be noted that in most of them an attempt was made to construct a furnace somewhat similar to a blast furnace, and to introduce the electrodes into the stack of this furnace at that point where the tuyeres are located in the former. All those who tried this soon learned that it was practically next to impossible to maintain the furnace walls adjacent to the electrodes. There was thus evolved as a matter of necessity the type of furnace construction which was finally adopted by the Swedish experimenters and such as is shown in Fig. 1². By referring to the same it will be noted that the furnace consists of two separate parts, namely, the crucible or hearth, superimposed above which is a shaft or stack. One of the most serious difficulties that had to be overcome in the perfecting of the electric reduction furnace was the maintenance of the crucible walls and roof. Such a construction as shown in Fig. 1 solved the difficulty as it meets the

¹A list of electric reduction furnaces now in use and in the course of construction in Europe will be found on page 539 of Vol. X of this journal.

²Those caring for further information in regard to the work of the Swedish experimenters are referred to the article of Lars Vingstom on "The Electric Shaft Furnace at Dönnarriet, Sweden," which appeared in this journal, Vol. VIII, page 11.

necessary pre-requisites, namely, keeping the end of the electrode dipping into the charge, as far removed from the side wall as possible, and second, keeping that part of the crucible out of contact with the charge at the point where the electrode enters the crucible. The reason for having to observe these precautions is due to the fact that the heat is excessive at that point where the electrode comes in contact with the charge, and hence this point should be as far removed as possible from both the side walls and roof of the crucible, in order to prevent the destruction of same through fusion.

The Electrode Problem.

When the demand was first made upon the manufacturers of electrodes for large electrodes of say 20 in. in cross section and 72 in. long, they experienced considerable difficulty in furnishing an electrode which would not go to pieces when put into commission. This caused considerable loss and annoyance to those attempting electric reduction and steel furnace work. Even if the electrode did not go to pieces, as soon as one-half to two-thirds of the same had been consumed it became too short for further use, and those unused portions represented a considerable item of expense. At the present time it is not only possible to obtain quite satisfactory electrodes, but also to

SiO ₂	2.40
P	0.011
S	0.009

Although at one time or another the Shasta Iron Company has planned to make pig iron from this ore, nothing definite was done in regard to the same until the summer of 1906. At this time the possibility of smelting this ore by electricity was brought to the attention of Mr. H. H. Noble, president of the Northern California Power Company. Due to Dr. Heroult's connection with the experimental work which had been done at Saulte Ste. Marie, he was commissioned to construct a plant for the purpose of treating the ores above mentioned. The place selected was at a point on Pitt River, near the mines. This place was named Heroult and is still known by that name.

In July, 1907, the first furnace having been completed, experimental work was begun. This furnace was a 1500-kw, three-phase furnace of the resistance type. It was soon found that the type of furnace first used presented mechanical difficulties which made its commercial use impracticable, and so it was closed down. A sketch of this furnace is shown in Fig. 2. In construction the crucible resembled a long, elliptical-shaped iron box. The bottom was made of heavy cast-iron plates, into which had been cast cast-iron lugs or pins, and upon the iron plates was tamped a carbon bottom made of carbon paste. The bottom plates were connected with the bus bars in such a manner as to form a neutral in the three-phase system.

The electrodes passed through the roof, as shown in the sketch. They were suspended from copper holders which were water-jacketed. The charge was fed into the crucible through the pipe (B), and the gases from the crucibles passed up around (B) into (A), the idea being to thus preheat the charge in (B) by burning the gases around (B), air being admitted for that purpose through slots in the outer pipe just above the roof of the crucible. These pipes, however, proved too small, and the charge in them became so hot as to hang, and so they had to be done away with. It was also found impossible to maintain a roof over the crucible, and so the furnace was finally operated as an open-top furnace. Several tons of A No. 1 foundry iron were made in this furnace, and later it was used for making ferrosilicon, but was finally closed down and removed in order to make room for the type of furnace subsequently adopted. After the closing down of this furnace, experimental work was conducted in a 160-kw single-phase furnace, and from the results obtained in this furnace and its methods of operation a 1500-kw, three-phase furnace was designed and built. This furnace was practically the same in construction as is the furnace shown in Fig. 1. In other words, there was developed simultaneously in Sweden and in California a similar type of furnace, although neither the Swedish inventors nor those engaged in the work in California knew of the other's work. The development of this type of furnace was continued until the spring of 1911.¹ At that time, acting upon the advice of the consulting engineer then in the employ of the Noble Electric Steel Company, a radically different type of furnace was constructed. The object aimed at in the new design of furnace was to secure one which would be as easily controlled, so far as the regulation of the nature of the charge was concerned, as is the present open-hearth steel furnace. This furnace was not a success.

Another type was then hit upon, and this type has since been developed into a commercially successful furnace. It is shown in Fig. 3. By comparing Figs. 2 and 3 it will be seen that the furnaces shown in each are quite similar, but in a recent article descriptive of the plant of the Noble Electric Steel Company it is stated that "this resemblance is only general."

Electric Iron Reduction Furnace in California.

The following information in regard to this furnace and its operation has been furnished the writer by the secretary of the Noble Electric Steel Company, and is chiefly taken from a re-

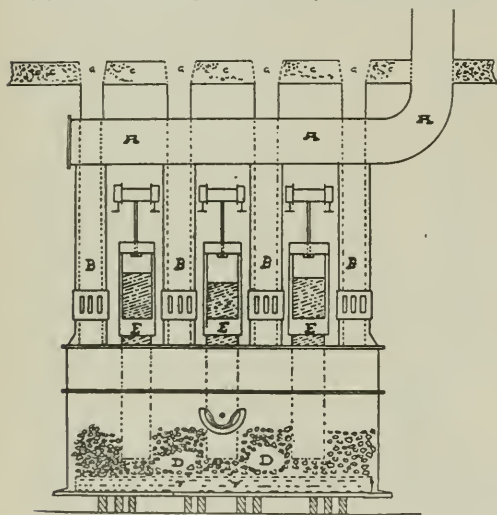


FIG. 2.—HEROULT THREE-PHASE FURNACE FOR SMELTING IRON ORE.

avoid the waste above mentioned. This is done by joining a new electrode to the butt end of an old one, as is done at Trollhattan² and at various other places.

So far our remarks have been applicable alike to the California practice and to the Scandinavian practice. It will, however, at this point, be necessary to discuss each practice separately, as they are at the present time radically different from each other. In order to understand the situation more clearly, it may perhaps be well to briefly outline the development of the electric reduction furnace as it has taken place in California.

The Development of the Electric Reduction Furnace in California.

For twenty-five years or more a company known as the Shasta Iron Company has owned a deposit of magnetite, located about seven miles from the mouth of the Pitt River, in Shasta County, California. This ore is a very pure magnetite, having on an average the following percentage composition:

Fe	69.90
MgO	0.10
MnO	0.18

¹For account of the work done in this type of furnace in California, see this Journal, December, 1911, page 631.

cent article by R. W. Van Norden on "Electric Iron Smelting at Heroult, Cal."

The furnace is housed in a heavy steel frame building, covered with corrugated-galvanized iron. It is rectangular in shape, 120 ft. long and 75 ft. wide, and has a height of about 60 ft. The building is in two sections or bays divided by a line of columns. The more southerly of these is the pouring floor, on which the pigs are cast. Throughout this bay is

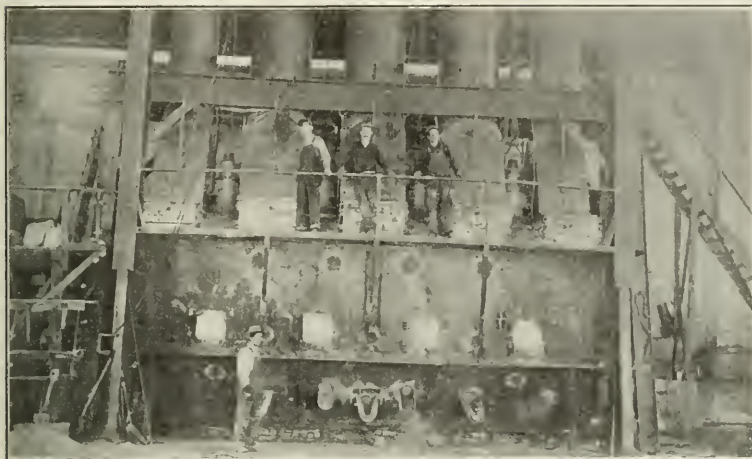


FIG. 3.—VIEW OF ELECTRIC IRON REDUCTION FURNACE AT HEROULT, CAL.

operated a 50-ft. Northern Engineering Company traveling crane. This is electrically operated and has two hoists, one of 20 and the other of 5-ton capacity. A Cleveland Electric Controller & Manufacturing Company lifting magnet, having a capacity of 2000 lb., is used in picking up the pigs. This magnet will lift about 1000 lb. of hot pigs.

The bay to the north of the center columns is given to the electric furnaces, their transformers and the necessary control. While but one furnace, having a capacity of 18 tons of pig iron per twenty-four hours is at present in use, two more of slightly greater capacity are practically completed. There is space reserved, and it is intended to immediately construct three more, thus bringing the total number of furnaces in this building to six.

The Furnace.

The crucible is contained in a steel shell 27 ft. long, 13 ft. wide and 12 ft. high. The shell is so lined with refractories that the upper half of the box is rectangular, but in the lower half the sides taper toward the center of the foundation, and the ends slope toward the middle to facilitate the flow of the molten bath, when the furnace is tapped. The tap hole is in the center and front of the crucible. The roof of the furnace is arched, and opening into the same are five stacks through which the charge is conducted into the crucible. In the four spaces between the stacks at the center of the dividing arches are inserted the graphite electrodes. The electrode jackets and the arched roof are water-cooled. The stacks or pipes through which the ore is charged are 24 in. in diameter and extend upward to a distance of 15 ft. from the roof of the crucible. These stacks are used only for the purpose of charging, no reduction being attempted in them. A charging platform is built along the top of the stacks and carries a track which runs to the mixing platform. Dump cars with the charge are run on the charging platform and dumped directly into the stacks. Except when receiving a charge, the stacks are closed with a tight-fitting cap.

The Electrodes.

Graphite electrodes are used. They are cylindrical in form, 12 in. in diameter and 4 ft. long. The upper end has a tapered male-threaded nipple, while the lower end has a corresponding socket with a female thread. As the electrode is fed into the charge a new one may be fastened to it, making a continuous feed.

An electrode lasts in continuous operation thirty days. Occasionally an electrode breaks in the furnace. This was formerly a serious matter and caused considerable delay and annoyance in the operation of the furnace. It was hard to determine just where the break had occurred, although it was generally at the threaded joint. A novel means has been devised by which the furnace men are now able to ascertain whether an electrode is broken or not. A bar is driven into the furnace opposite the electrode, through one of the several peep holes, and at the same time the electrode is tapped on top with a hammer. The man on the bar simply listens for the tapping. The tapping is then done on the bar and the man on the top listens. A break may be accurately located in this manner.

Transformers.

In the rear of each furnace, and as close thereto as is possible, are the three service transformers which supply three-phase current at from 40 to 80 volts to the electrodes. These transformers are oil-immersed and water-cooled and have a capacity each of 750 kw. The low-tension connections to the electrodes consist of eight pieces of flat copper bar, $\frac{3}{8}$ in. thick and 5 in. wide, bolted together. On the 2400-volt primary side there are brought out eight current taps for voltage regulation.

These are taken to an oil-immersed, individual solenoid-operated switch group, which, with auto-transformer compensator, gives fifteen steps for voltage variation.

Electric Control.

Electric control is through a switchboard, there being a panel for each furnace. As the current and power-factor in each phase must be under observation at all times during operation, separate meters are installed in each phase. The requirements for one panel are three ammeters, three voltmeters, three wattmeters, three power-factor meters and three recording wattmeters. These are mounted across the panel in rows of three each. Under the first four sets named are three hand wheels to control the voltage variation, and under these three switches, which control the entire load, and still under these are the recording wattmeters.

For operating the voltage control and the main circuit breakers there is a 7½-kw. motor-generator set, comprising a 125-volt direct-current generator, directly connected to a 10-hp induction motor. This set has a small panelboard mounting a circuit breaker, ammeter, voltmeter, and two single-pole knife switches. The transformers, switches, and control were supplied by the General Electric Company. In the event that line voltage should fall, or for any other cause, the direct-current supply should become deranged there is a National Storage Battery Company's set, having a capacity of 7½ kw, which may be instantly switched in, and thus prevent the furnaces from cutting out in the case of low voltage.

*Reprint from Journal of Electricity, Power and Gas, Nov. 23, 1912.

Operation.

In operation the furnace is continuous. After a period of eight hours the hearth contains a full bath of molten metal, and this is, therefore, tapped three times each day. Charging is done at regular intervals and the current is not shut off at any time.

During the period of smelting the change in electrical conditions is interesting. At the beginning of the charge the power-factor is almost unity. This gradually lowers as smelting continues, until with a full bath of metal a power-factor of about 5 per cent is reached. If coke is used in place of charcoal, or if a mixture is employed a different set of power-factor conditions exists. By studying these conditions it is possible to know the exact condition of the charge by looking at the meters. The load is, of course, a function of the voltage, and with half voltage the load will drop one-quarter.

In charging, the ore cars are run on the mixing floor, which is at the same level as the charging floor. Cars with charcoal come into the lower part of the building and are hoisted on an electrically operated elevator to the mixing floor. Lime and quartz for flux are also brought in in cars, and each material is dumped into bins. The mixing is done in a car which is run on a platform scales. The charge is placed in layers, the proportions depending upon the tests made in the laboratory by the chemist. Following is a representative charge:

- 500 lb. iron ore (magnetite);
- 135 lb. to 150 lb. charcoal;
- 3¼ lb. lime (well burned);
- 12½ lb. quartz.

Comparison of the California and Swedish Furnace and Process.

As to the construction of the two types of furnaces, the difference is so apparent that no comment is necessary. As to the process, the California practice differs from the Swedish practice very decidedly in the following respects, namely:

1. No attempt at reduction in the stacks of the furnace.
2. Non-circulation of gases.
3. The use of limestone calcined outside the furnace.

In other words, the California practice is just the reverse of the Swedish practice, for in the latter they attempt reduction of the ore in the shaft of the furnace, they circulate a part of the gases escaping from the top of the shaft, and they do not use burned limestone.

Reduction in Shaft.

It would seem that the ideal condition in electric reduction furnace work would be to have the charge in such a condition as to only require melting by the time it comes in proximity to the electrodes. If this be done, the electric current is then only called upon to furnish the additional heat necessary to bring about the melting of the hot, previously reduced oxide. Of course, the heat for the reduction must come from some source, and its origin would be in the crucible and as a result of the heat developed during the melting of the charge, but, as will be stated presently, the excess heat so developed can be transferred by the circulation of the gas to the charge in the shaft and not carried away by radiation and an excessive amount of cooling water.

Gas Circulation.

In the Swedish practice, a portion of the gases rising from the top of the shaft are returned to the crucible and made to pass through the same and up through the charge in the shaft. This is done for the following reasons:

1. To cool the superheated brickwork.
2. To carry up through the stack a sufficiently large volume of gas which has been raised to a high enough temperature to bring about reduction in the stack.

As has been previously pointed out in this article, one of the most serious difficulties that had to be overcome in the development of the Swedish type of furnace was the destruction of the crucible due to local heating in the vicinity of the electrodes. Therefore by returning to the crucible a portion

of the gases resulting from the reduction of the ore, it is possible to prevent excessive local heating and at the same time to make use of the heat instead of wasting it. This matter has been the subject of much discussion, and has its objections, but the writer is of the opinion that its advantages in the way of economy of operation far outweigh its disadvantages. Perhaps it will in time be found practical not to use gas circulation in the electric iron reduction furnace, but, on the other hand, we believe that reduction previous to melting will be considered the proper practice, which means stack reduction, and the present California practice will not find application except in those instances where the local conditions warrant the same, as is the case at present. As has been pointed out, the present demand in California is for a soft grey foundry iron. The furnace now used gives that product, but this does not mean that such a construction and such a process are absolutely necessary for the production of such an iron. As a matter of fact, many tons of the same kind of iron were made at Heroult in a furnace of the Swedish type, and the Swedish inventors claim that it is possible to operate their type of furnace so as to reduce regularly an iron of the same character as is now produced at Heroult. They state, however, that in operating the furnace for this purpose the amount of electrical energy consumed per ton of pig produced is increased.

As to why the California type of furnace is better suited to the production of a soft grey foundry iron than is the Swedish type of furnace, it is probably due to the fact that reduction is performed solely by solid carbon in the crucible and as silicon is only reduced by solid carbon, its reduction to silicon takes place at the same time as that of iron oxide to metallic iron. The silicon is dissolved in the iron and by its presence causes the carbon to precipitate out as graphitic carbon, and there is thus obtained the soft grey iron desired. We are also of the opinion that in a furnace having a rectangular-shaped crucible, such as has the California furnace, the temperature of the charge in the crucible as a whole is much higher than is the case in a furnace of the Swedish type, and this higher temperature is more favorable to the reduction of silica to silicon, the latter, as before stated, being largely the controlling element in the production of soft grey foundry iron.

The Use of Calcined Limestone.

This has been repeatedly suggested, and theoretically it is the proper thing to do, but when operating a furnace of the Swedish type there are two objections to its use, namely:

1. It increases the percentage of fines in the charge.
2. It makes the charge hang.

As to the former of these objections, the idea seems to be more or less prevalent that there is no limit to the percentage of fines that may be used in the electric reduction furnace. We are not definitely informed in regard to this point with reference to the California practice, but judging from observations made upon the smelting of black sands, when the furnace consisted simply of a crucible and no shaft, more or less difficulty was experienced on account of the charge being made up entirely of fines. In the Swedish practice, the matter of the proportion of fines that can be used in a charge was definitely determined in the work at Trollhättan, and it was found that a large proportion of fines was detrimental to smooth running and good results, but Mr. Leffler, one of the engineers in charge of the work at Trollhättan, states that he is of the opinion that calcined limestone may be advantageously added to the charge if no reduction is attempted in the shaft.

Summarized, then, we may say that the California practice as compared to the Swedish practice presents the following advantages:

1. Permits the use of limestone calcined outside the furnace.
2. Does not require the circulation of gases.

As to this latter point, and also as to reduction in the shaft, we are not sure but what the California practice would prove more efficient if both were done. Although a very full and

¹"Gas Circulation in Electrical Reduction Furnaces," J. W. Richards, Transactions, American Electrochemical Society, Vol. XXI, page 403.

complete record of the working of the furnace at Trollhättan has been published, such is not the case as regards the operation of the furnace at Heroult, and so it will not be possible to make a definite comparison of the two practices until it is possible to receive as definite information in regard to the working of the furnace at Heroult as is now available in regard to the operation of the furnace at Trollhättan.

Lines Along Which the Electric Furnace Will Probably Be Improved.

The Size of the Unit.—In Scandinavia the furnace has gradually been increased in size until now there are 3000-hp furnaces in operation and A. B. Elektrometall have completed a design for a 7500-kw furnace. Naturally the problem in connection with a furnace of the Swedish type is to get such an arrangement of the electrodes as to insure a practical uniform heating of the whole contents of the crucible, and at the same time maintain the electrical equilibrium of the power system.

It is quite certain that the Swedish type of furnace will be improved along this line, namely, increase in size of the unit, and it is for this reason that results obtained from the operation of a furnace designed for 7500 kw will be awaited with great interest.

As for the California type of furnace, it will no doubt be possible to indefinitely increase the length of the same, just as has been done with the modern rectangular copper blast furnaces.

Efficiency.

The efficiency of the furnace will also be increased as time goes on, and probably by improvements along the following lines:

- (a) The utilization of the waste gases.
- (b) The securing of a high power-factor.
- (c) The correction of induction losses.
- (d) The further study of the single-phase furnace vs. the three-phase furnace.

As regards the latter, inasmuch as practically all large power installations are three-phase, it would seem that the only logical thing to do is to use three-phase current in electric furnace work, but from data submitted by Catania it would seem as if the mono-phase furnace is more efficient than the poly-phase furnace.

Summary.

Summarized then, we may say that the electric iron reduction furnace has passed the experimental stage; that it is meeting the requirements in those localities where it has been introduced, and that it will no doubt find extended application in those countries where the conditions are favorable to its introduction, namely, where electric power is comparatively cheap and where coke and charcoal are comparatively high. Inasmuch as the cost of electric power is largely the governing factor in determining its adoption, and as the cost of electric power is constantly decreasing, we may expect to find an extended application of the electric reduction furnace in the future.

Pittsburgh, Pa.

Bakelite Billiard Balls.—Another milestone has been reached in the victorious course of bakelite in its introduction into the most varied industries by the invention of the bakelite billiard ball. Ivory lacks uniformity and is seriously affected by changes in temperature and climate. As a result of years of scientific experiment a composition ball has now been perfected which is of the same weight and resiliency as the best ivory ball, but practically indestructible and absolutely unaffected by heat or cold, dryness or dampness. This composition ball is the invention of Mr. J. W. Hyatt (the inventor of celluloid) in co-operation with Mr. Charles F. Burroughs. The basic material of this ball is bakelite, Dr. L. H. Baekeland's well known invention. The bakelite ball is made by the Hyatt-Burroughs Billiard Ball Company, 141 Commerce street, Newark, N. J.

The Billiter Alkali-Chlorine Cells.¹

By A. J. Allmand, D.Sc.

The most important advances made in late years in the technical electrolysis of alkaline chlorides for the production of caustic alkali and chlorine are perhaps due to Dr. Jean Billiter, privatdozent in the University of Vienna. Some years back he designed a diaphragm cell which is being exploited by Siemens & Halske, and recently he has invented a modified form of the well-known bell-jar cell, which is undoubtedly, taken all in all, one of the most efficient cells now operated technically. The Billiter-Siemens diaphragm cell I propose to mention briefly only, as full accounts of it are available elsewhere,² and shall devote most attention to the Billiter "membrane" cell, the first installation of which started work early this year.

Billiter-Siemens Cell.

Most diaphragm cells in technical use have vertical diaphragms (e. g., Hargreaves-Bird, Townsend, Finlay). Billiter however, has returned to the use of horizontal diaphragms such as were formerly employed by Le Sueur and by Carmichael. He has compared elsewhere the relative advantages and disadvantages of the vertical and horizontal arrangements.

A horizontal diaphragm is more slowly attacked chemically, as it remains more or less saturated with the alkali which migrates upwards from the cathode situated immediately underneath. This is particularly important when making strong alkaline liquors. Further, the hydrostatic pressure on it is uniform at all points, not varying with the height as in vertical diaphragms. This means a more uniform flow of liquid through, and consequently a more efficient checking of the movement of the OH⁻ ions towards the anode.

There are corresponding structural disadvantages. The floor-space needed is greater. Further, the insides of the cells are less easily accessible and the anodes are necessarily of more complicated construction.

The active diaphragm material in the Billiter-Siemens cell consists of a mixture of asbestos wool and BaSO₄, spread out evenly on top of a sheet of woven asbestos cloth, which in its turn rests on the iron-wire net-work which constitutes the cathode. The function of the asbestos cloth is merely to support the BaSO₄ mixture and to prevent its being displaced by the lively hydrogen evolution. The cathode makes contact with the iron bottom of a trough, the sides of which are lined with cement, and the anode chamber is formed by a non-conducting bell-jar which is securely cemented around the edges of the diaphragm. An important feature of the cell is the arrangement for heating the electrolyte by means of earthenware pipes through which hot liquids run, these being immersed in the brine at the top of the bell-jars.

Cells have been made carrying up to 3,000 amperes at a current density of 4-6 amp. per sq. dm. Working at 85-90 deg., 12-16 per cent. NaOH or 18-20 per cent KOH can be made at a 95 per cent cathodic current efficiency, employing 3.5 volts.

An interesting laboratory study of the Billiter-Siemens cell has been recently published by Mühlhaus.³ He finds that the BaSO₄ used for the diaphragm is best precipitated hot. No change of size of grain when in the hot liquors of the Billiter cell is then likely to take place, and a regular and uniform liquid flow is thus assured.

The highest current density permissible is about 6-7 amp. per sq. dm.

Above this limit, disintegration sets in. Using 30-32 per cent brine at ordinary temperatures, the following figures were obtained

¹A paper read before the Faraday Society in London on November 26, 1912.

²Billiter, Die elektrochemischen Verfahren der chemischen Grossindustrie, II, 215 ff. Die electrolytische Alkalischlöriderzeugung mit starren Metallkathoden, I, 119, 191. See also Allmand, Principles of Applied Electrochemistry, p. 375. Also Chemiker-Zeitung, 1911, p. 373. Der Papier-Fabrikant, 1911, Parts I, 11, 33, Wochenblatt für Papierfabrikanten, 1911, Paris 19, 42.

³Dissertation (Dresden, 1911).

Concentration of Alkali.	Current Efficiency. Per Cent.
3n.	98-99
4n.	95-97
5n.	80-85
6n.	75

The chlorine gas fell in purity from 99.5 per cent to 92 per cent as the alkali concentration was increased. The oxygen produced exceeded the amount of CO_2 . The solid caustic resulting contained considerable quantities of NaClO_3 if produced from liquors stronger than 4n. The best results on the whole were given when making 4n. NaOH at a current density of 4 amp. per sq. dm. Raising the working temperature to 70-80 deg. improved both energy efficiency and purity of alkali.

The Billiter-Siemens process is now operated as follows:

(a) In Aschersleben, Saxony, by the Kaliwerke Aschersleben A.G. Ten 2,000-ampere units.

(b) In Höchst, by the Farbwerke Höchst.

(c) In Brückl, Bosnia, by the Bosnische Elektrizitäts A.G. Sixty-four 2,500-ampere units.

(d) In Krumau, Bohemia, by Ignaz Spiro und Söhne. Thirty-two 500-ampere units.

(e) At Niagara Falls, by the Niagara Alkali Company. A 1,000-kw installation of 3,000-ampere units.

A number of plants are also being installed or enlarged. The Niagara Alkali Company (which has succeeded the old Roberts Chemical Company), is at present increasing the size of its plant from 1,000 to 3,000 kw. It has also been made known that a new factory for caustic alkali and bleach is being erected, commencing with 78 2,000-ampere units, and that a textile factory is installing 24 100-ampere units.

Billiter-Leykam Cell.

In this cell⁴ Billiter has successfully modified the bell-jar process, placing the cathodes (hooded to collect the hydrogen) immediately underneath the bell-jar, and not around its sides, as in the Aussig cell. Similar constructions had been formerly suggested by Richardson⁵ and by Bein,⁶ but had achieved no technical success.

By doing this, Billiter gains an important advantage. In the ordinary bell-jar cell, both flow of liquid and flow of current change in direction at the lower edge of the bell-jar. This results in non-uniformity of current density and liquid flow in the different parts of the bell-jar section. At some points liquid will be flowing more quickly than is necessary to overcome the OH^- migration towards the anode, at other points the contrary will be true; the movement of the OH^- ions will predominate, and the current efficiency will fall. This last statement particularly holds for those parts of the bell-jar section furthest removed from the walls, where the current of liquid is exceedingly slow. The larger the bell-jar, the greater proportionately is this disturbance. It also increases rapidly with current density. The consequence is that the units (bell-jars) at Aussig are very small, and can only be worked at a low current density, furnishing weak alkali. Such a plant requires much floor space for its output and the labor charges are high.

In the Billiter cell, this change of direction and the resulting irregularities are avoided, the motion of the brine being directly opposed to that of the current lines until it has passed the cathodes. The consequence is that high current densities can be used, stronger caustic liquors produced, high current efficiencies obtained, and large units constructed. At the same time, the advantage of the bell-jar cell—the absence of diaphragms—is essentially retained. For though the current must all pass through the hoods which cover the cathodes, this is not true of the electrolyte. Most of it passes directly through the space between neighboring cathodes, and is charged with caustic by diffusion from the concentrated alkali which fills

the cathode hoods. Not only does this alkali protect the hood material (in practice, woven asbestos) against chemical action of chlorine compounds, but suspended matter present in the brine is not filtered out, as happens in diaphragm cells, and it is not necessary to purify the brine in any way. The asbestos hoods are made of as coarse a structure as is consistent with their function of leading off the hydrogen gas, and affect the voltage to a very slight degree only. Finally, the cell is heated in a similar fashion to the Billiter-Siemens cell, and the voltage thus lowered.

The essential parts of the Billiter-Leykam cell are shown diagrammatically in Figs. 1 and 2. A is the bell-jar inverted in the cement trough B, and provided with anode outlet for chlorine, inlet for brine and heating arrangement (not shown). The cathodes C are arranged below the bottom level of A and consist in practice of T-iron. They are enclosed separately in long woven asbestos tubes which are suitably braced in order to avoid deformation by the evolved hydrogen, and are slanted sufficiently to allow the gas to stream away readily. These cathodes project out into a side compartment of the cell as shown, from which the hydrogen can, if necessary, be collected.

The causticised brine flows off through D.

The Austrian rights to the Billiter bell-jar cell have been purchased by the paper manufacturers Leykam-Josefsthal A.G. The first plant has been erected in their factory at Gratwein, a village on the Austrian Southern Railway, a few miles from Graz, has been in continued operation since early 1912, and has given every satisfaction. Some 1,700 to 1,800 kilos of active chlorine are produced daily, and the whole plant uses about 400 P. S.—350 P. S. for the cells and 50 P. S. for auxiliary pumps, fan, etc. Power is purchased from a company at 150 kr. (\$31.25) per P. S. year. After converting into direct current, and reckoning all losses in leads, etc., the cost comes to 180 kr. (\$37.50) per P. S. year.

The plant consists of 561,200-ampere units, 50 of which were working at the time of my visit last July. The cells are 5'5 metres long, 1'1 metres broad (internal measurements), and about 1'3 metres high. The side walls commence to slope inwards about 70 cm. below the top of the cell. They are built of reinforced concrete, the upper part being lined with earthenware bricks set in cement.

They are provided with three cement lids, each of which cement lids, each of which carries eight anodes.

The anodes are of Acheson graphite (1'0 x 0'18 x 0'05 metre), each being led through its lead by means of two graphite rods. The lids are cemented on to a shallow shelf cut out of the wall of the cell by means of ordinary putty, which stands the action of chlorine very well and always remains fairly soft. Putty is also used at the points where the anodes pass through the lids, as the weight of the anodes is not borne by the latter, but by the copper leads above. Each anode takes 50 amperes at a current density, assuming lower and side surfaces to be active, of about 1'6-1'7 amp. per sq. dm.

The cathodes are of T-iron and enclosed in woven asbestos tubes. These taper down slightly towards their closed lower ends, and the whole is braced by the insertion of a wedge-shaped strip of asbestos cement. The tubes pass through holes in the dividing wall into the side compartment of the cell as indicated in Fig. 1. Two iron rods attached to each T-iron cathode take the current to the copper leads above. The hydrogen is allowed to escape. The cathodes are, of course, about 1 metre in length, and the mean breadth of the asbestos tubes is about 5 cm. Between each tube there is a space of 5 to 8 mm. The upward slope of these tubes is about 2 cm. per metre.

The brine (moving from a reservoir under gravity) is led to a distributing box situated above the cell. From this it proceeds by way of a series of short capillary tubes (1.5 mm. bore) to 22-in. tubes, which pass through the cell lids, and is

⁴British Patent, 11,693 (1910), *Proceedings of Applied Electrochemistry*, 373.

⁵British Patent, 5,694 (1893).

⁶Chem. Abstr., 7, 18 (1895).

⁷The P. S. (Horse Power or German Horse Power) and P. S. year are of course about 1/3 per cent. smaller than the corresponding H. P. and H. P. year units.

thus fed in between the anodes. Regulation of the main supply to the cell is effected by means of a rubber tube and clip. This arrangement acts on the whole very satisfactorily, though the capillaries sometimes get stopped up. The causticised brine is drawn off from one side of the cell by means of siphon overflows. Three are provided, but one only is used in practice.

The heating arrangement used by Dr. Billiter in his diaphragm cell, though available for all other installations of his "membrane" cell, cannot be used at Gratwein. The cells are, therefore, provided with a heating arrangement which employs steam, the details of which, however, I cannot divulge here. When I was at Gratwein the cells were being mostly worked at room temperature, whilst those few which were being heated were working at 65 deg. to 70 deg. The cost of the steam used is very small.

The cells are arranged in four rows, two rows being worked in series, and are insulated by glass resting on concrete supports. Beneath the cells are arranged the chlorine and brine pipes, and a channel for the caustic liquors. The chlorine pipe-line is constructed of cement, junctions being made by covering the ends with a cement box and filling up with as-

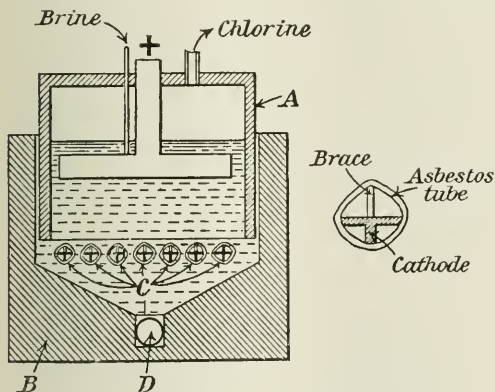


FIG. 1.—SECTION OF BILLITER-LEYKAM CELL

phalt. The current leads are of copper, those connecting the cells of circular cross-section and 3 cm. in diameter. They are all painted with a composition stable against chlorine and acid fumes, as are also the tin tubes and rubber connections referred to above. The cells are worked under slightly reduced pressure ($\frac{1}{2}$ cm. of water), and, as a matter of fact, no smell of chlorine was noticeable. Ventilation can be effected, if necessary, by blowing air in through the agency of a fan driven by a horse-power motor. A traveling-crane is provided to facilitate lifting off the tops of cells, etc.

Saturated brine is used, prepared in the simplest way. A long cement trough is filled with large lumps (20-40 cm.) of crude salt. Water enters continuously at one end, and saturated brine leaves at the other. It is not purified, and can be sent straight to the cells. The average cathode liquors produced contain 12-13 per cent NaOH and about 20 per cent NaCl. By decreasing the rate of flow of brine, 16 per cent NaOH can readily be made. Up to 350 litres of brine per cell are causticised in 24 hours. The liquors are first evaporated in two vacuum pans, furnished by the Skodawerke A.G., Pilsen, and similar to those used in sugar refineries. In these they are successively brought to 30 deg. and 40 deg. B. Then by means of a Kaufmann rapid evaporator they are brought to 50 deg. B before going to the melting-house. The final product, which will probably be later somewhat improved upon, contains 97 per cent NaOH, 1.2 per cent Na_2CO_3 , and 1.8 per cent NaCl. The chlorine is diluted and absorbed in milk of lime.

Cells which are worked at room temperature take about 4 volts. Those I saw working at 65-70 deg. require 3.4-3.5 volts. Long continued experiments carried out with a 250-ampere unit have shown that at the normal working temperature of 85 deg. the voltage falls to 3.1 volts (*plus* in all cases the voltage drop in the leads). The cathodic current efficiency is about 92 per cent for 3n-4n alkali.

It remains to be added that wear and tear in the cells are very small. Under normal working conditions the anodes lose about 1 mm. in thickness at their bottoms and sides in two months. Suspended matter in the brine settles on the bottom of the cells, and the asbestos hoods remain unaffected. Finally, when once matters are adjusted, the whole system works automatically. The actual cell-room in Gratwein is worked by shifts of one workman only, whose duty it is to adjust the rates of flow in the different cells. Moderate fluctuations in rate of flow of brine or in current density result in a change in the alkalinity of the liquors produced, and in a corresponding *small* change in current efficiency. The cells at Gratwein work at a current density between the electrodes of about 2 amp. per sq. dm. only. With cells working at 85 deg. the most convenient figure, according to Billiter, is 4 to 6 amp. per sq. dm.; but, by suitably varying the other conditions, 14 amp. per sq. dm. can be employed.

The withdrawal of 350 litres of catholyte per cell per 24 hours corresponds to a movement of the brine of about 0.0007 cm. per second. The OH' ions in that part of the catholyte between the electrodes will have a velocity anodewards of just about the same magnitude. (E.g., assuming the cell to work at room temperature, the conductivity of the electrolyte would be about 0.5 recip. ohm per cu. cm., the velocity of the OH' ion at a voltage gradient of 1 volt per cm., 0.0018 cm. per sec., and the actual velocity $0.02 \times 0.0018 = 0.5 = 0.00072$ cm. per sec. This close approximation between the two opposing velocities, with the consequent straying of OH' ions towards the anode, is of course the chief cause of the 8 per cent deficiency in cur-

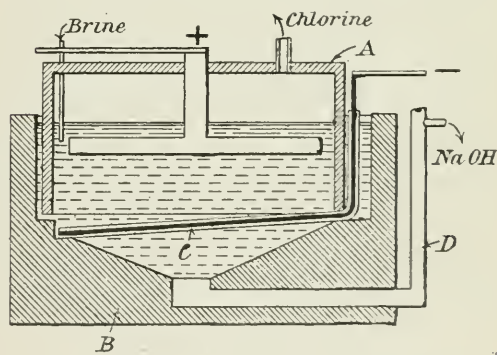


FIG. 2.—SECTION OF BILLITER-LEYKAM CELL.

rent efficiency, as the amount of chlorine which dissolves in the concentrated brine, and which would thus tend to be carried to the cathode, is very small.

The second installation of the Billiter membrane cell is at the present time being erected in Saxony.

In conclusion, I will again emphasize the chief characteristics of the Billiter-Leykam cell. They are—

- (1) Freedom from diaphragm troubles.
- (2) Use of impure brine.
- (3) Automatic working, great adaptability, and low attendance charges.
- (4) Alkali (3n-4n) strong compared with most other non-mercury cells.
- (5) With heated cells, lowest energy consumption per ton of product of any of the electrolytic cells now working on a large scale.

*The patent rights are not there available.

Concentration of Scheelite.

There is but one commercially important deposit of scheelite in Canada. It lies in the central part of Halifax county, Nova Scotia, 12 miles from the Atlantic coast and 34 miles from the nearest railroad point. It is 2 miles west of the old Moose River gold mines. The ore consists of quartz, scheelite, ankerite and arsenopyrite. Not a particle of either of the black tungsten ores¹ has ever been found with the scheelite. In a paper read before the Colorado Scientific Society by Victor G. Hills, the method of concentrating this ore is described, as follows:

Construction and Operation of Mill.

On March 1, 1911, I took charge of the property to develop the mines and build a mill. Foundations for boilers and air-compressor were begun by blasting in frozen ground. The lumber for building was made with a portable saw-mill from trees cut in the nearby woods. There was neither clay nor sand for brick, mortar, or concrete. Brick was hauled in on sleighs at a cost of \$30 per thousand; also some sand was hauled 18 miles by sleighs. Later, when the river was clear of ice, sand was made out of quartzite boulders by crushing in an old water-power gold stamp-mill found 2 miles distant. A wagon road had to be built into the camp when the ground was thawed out enough to permit it. The best English Portland cement and Scotch fire-brick were readily obtainable at Halifax, but the 34 mile wagon haul for these things and for the machinery made construction tedious and slow, and the year 1911 was consumed before the mill began producing concentrate.

A modest but substantial mill was built to treat one ton of crude ore per hour. I outlined a flow-sheet to take advantage of every inexpensive device to prevent sliming. The ore was dumped over a grizzly before going to the crusher. The roll feed was washed by a spray so that only the clean coarse material went to the coarse rolls. Rolls were used for crushing, as being undoubtedly better than stamps or any other form of screen-faced pulverizing mill to avoid sliming.

While the scheelite is more friable than either the quartz or the ankerite and, as usual in the concentration of all tungsten ores, the main loss is in the slime tailing, yet this loss with scheelite is somewhat less than with our Colorado black ores. The walls of the vein are slate which makes an enormous amount of slime and augments the difficulties of handling this portion of the pulp. This must be remembered when considering the loss in slime tailing.

In milling, since scheelite and mispickel have the same specific gravity—6.0—they together form the concentrate from the tables. The pyrite is just enough lighter to enable it, for the most part, to be thrown out with the tailing.

The raw concentrate consists of some 45 per cent mispickel coming partly from the vein and partly from the slate wall-rock. To get rid of the arsenic it was necessary to roast and treat with a magnetic separator. This phase of tungsten milling, of course, does not appear in our Colorado work.

Roasting for Magnetic Separation.

For roasting a Wilfley revolving hearth furnace was built. It proved quite satisfactory. For treating separately numerous small lots of both coarse and fine material, and particularly where the furnace can not be kept in steady operation, I know of no roaster that would be more desirable. The literature on the subject of ore-roasting seems to be notoriously meager in reference to the degree of heat used in connection with the time of exposure. I found that this mispickel was decidedly better prepared for the magnetic separator by a high temperature and short exposure. Thus, 1500 deg. F., for 50 seconds gave better results, for example, than 1100 deg. for 90 seconds. The tests which had been made on roasting this ore at the Kingston School of Mines had evidently proceeded on the idea of burning off as much of the arsenic as possible; but this idea was all wrong. The best separation of the arsenic can be made by the

least possible roasting that will render the iron magnetic. The arsenic can then be removed with the iron.

A Dings single-magnet separator made an excellent, clean separation. The magnetic part carried an average of 0.72 per cent WO_3 , with some lots as low as 0.42 per cent. The non-magnetic product carried only 0.34 per cent arsenic. And these returns could, if necessary, be further reduced by using a two magnet separator or by running through a second time. The separator operated at 8 amp.

The finished product carried, besides the 0.34 per cent arsenic, a little sulphur and phosphorus. Otherwise the concentrate contained no deleterious element. While no other tungsten ore in the world compares with our Colorado product for freedom from undesirable elements, this Nova Scotia product is quite within the general market requirements and compares favorably with the tungsten ores from most parts of the world.

With my trial run the mill saving was 86.8 per cent. Remembering that, besides the excessive amount of scaly slate gangue, augmenting the slime loss, this concentrate has to be subjected to a second loss from flue dust in roasting and a third loss from magnetic separation, I regard this as a good saving. The market product carried 70 per cent WO_3 . That this is higher than the average of our Boulder county product is, of course, due to the higher tungstic acid content of the lime tungstate. The coarser table products at Scheelite would run as high as 73 per cent WO_3 . A test on this ore made at the Kingston School of Mines reported a saving of 89.5 per cent; but the ore was sorted nearly free from slate and the method used was impracticable to apply in a small and simple mill. Additional mill equipment, introducing coarse jigs, regrading of middlings, and more slime-settling capacity, would materially increase the saving. It is the old mill problem; to determine whether the additional saving would pay for the outlay necessary to produce it.

Production of Copper and Nickel in the Electric Furnace.

At the first general meeting of the recently formed "Gesellschaft Deutscher Metallhütten und Bergleute" (Society of German Metallurgical Engineers, essentially restricted in its scope to non-ferrous metallurgy), Mr. M. Stephan gave an account of some experiments made by him on electric smelting in non-ferrous metallurgy. Mr. Stephan is the superintendent of the Girod electric steel works in Ugine, which are familiar to our readers from previous descriptions.

Copper.—Formerly the Keller-Lecloux Company, of Livet, France, have experimented with a Chilean copper sulfide ore containing 5.1 per cent Cu producing a matte of 40 per cent to 50 per cent Cu and a slag with 0.1 per cent Cu. The problem brought before the Society An. Electrometallurgique *Procédés Paul Girod* was to reduce a copper oxide ore in the electric furnace by means of solid carbon as reducing agent and with a certain limestone as flux.

The ore came from the Belgian Congo, being mined by a Belgian-English concern. In five different analyses given the CuO varies from 21.01 per cent to 5.73 per cent, SiO_2 from 28.48 per cent to 78.55 per cent, with from 4 to 13 per cent Al_2O_3 from 4 to 16 per cent Fe_2O_3 , and besides smaller impurities from 2 to 7 per cent CoO , no nickel.

The moisture in the ore varied from 7 per cent to 32 per cent. It was not removed in order to meet the conditions of practical operation and to carry the experiments out under most unfavorable conditions.

Charcoal, coke, and anthracite were used successfully as reducing agents. Charcoal will probably be the cheapest under the special circumstances. Electrodes were primarily carbons of Girod's own make, and also bottom electrodes of soft steel or water-cooled copper; 500 kw were available for the experiments, for which not over 200 kw were used at from 30 to 150 volts.

Ores of the kind described are difficult to work with in the

¹Milling practice on Colorado tungsten ores was described in this journal, July and August, 1911.

water-jacketed copper blast-furnace. They do not offer any trouble, however, in the electric furnace. Within the limits of the highly refractory furnace lining it is possible to regulate the temperature, viscosity and composition of the slag to suit any condition.

Electric furnaces similar to the Girod steel furnace type were used and dimensions changed in several runs within wide limits in order to secure enough data for the construction of a large furnace for the same work. Electrodes were used suspended from the top and inserted in the bottom, also in the sides for heating by radiation, and a system was adopted of heating with a smothered arc mainly by the resistance of a thick layer of slag over the conducting metallic charge. The temperatures were measured with LeChatelier, Wanner, and Fery pyrometers.

A slag of the composition

SiO ₂	Al ₂ O ₃	CaO	MgO	Fe ₂ O ₃	MnO	CuO	CoO
51.90	11.31	16.83	13.71	3.55	0.94	0.46	0.87

begins to melt at 1250 deg., begins to flow in a pasty condition at 1350 deg., and is liquid enough at 1400 deg. to allow the little copper balls to completely settle. At 1550 deg. the slag is liquid enough to flow off freely; 1920 deg. were necessary to render the slag liquid when the highly acid ore was smelted without any fluxes.

The pig copper (Schwarzkupfer) produced analyzed in six different runs from 65 per cent to 95 per cent Cu, containing from 1 to 21 per cent Fe and from 1 to 11 per cent Co. The purity of the product will be the higher the lower the melting temperature.

The lower the temperature the smaller will be the chance for reducing any of its impurities, but at the same time more copper will also be retained in the slag so that the output is decreased.

It will be a matter of commercial calculation, according to the composition of the ore and other special conditions, whether a high-quality product or a maximum output will be desirable.

A continuous run for several days with the same ore aiming at slags of about the above mentioned composition required 1000 to 1200 kilowatt hours per ton (of 2205 lbs.) of ore.

These figures are high, as a result of the amount of heat required to keep the very viscous and anomalous slags in liquid condition.

With an easily fluxible ore the specific power consumption was only about 500 kilowatt hours.

The electrode consumption averaged 8 kg. or 17.6 lbs. per ton of ore, using an electrode of 1.8 specific density and 3500 microhm resistivity (per sq. cm section and cm length); it was operated with 4 amperes per sq. cm (25.8 amperes per sq. inch).

Coal for reduction was used at the rate of 25 per cent of the copper in the charge.

The best lining to withstand the severe conditions of the furnace was tamped from fire-clay with 80 per cent SiO₂ and 15 per cent Al₂O₃.

Nickel.—While in older experiments of his firm and in the experiments described in the paper read by Walter Morrison before the American Electrochemical Society last year the product was a nickel with high Si contents. Mr. Stephan was able to obtain a ferro-nickel with only 3 per cent to 5 per cent Si, as for instance:

Ni	Fe	Si	Al	C	S
41.50	51.61	4.33	0.81	1.34	0.04 per cent

from an ore containing 8.33 per cent NiO. A small furnace of 220 kilowatt treated 3 tons of ore in 28 hours and produced 350 kg. of metal. From this the author concludes that on a commercial scale the specific power consumption would easily be reduced to 1200 kilowatt hours per ton of ore and that the electrothermic smelting of nickel ores will pay in districts when power is cheap.

The complete paper by Mr. Stephan may be found in *Metal and Erz*, vol. 10, pp. 11-17 and 84 to 86, 1912.

Wood Waste Utilization.

The Industry in Its Relation to Expert Advice

By John E. Teeple, Ph.D.

Here is an industry expanding and changing so rapidly that it is difficult to keep abreast of its progress. Every time I attempt a general review, its aspect has changed.

Processes and patents without number loom above the horizon. Most of them die without ever seeing the light of day in a commercial form, many never advancing beyond the paper and ink stage. Some start life hopefully, only to draw out a weary existence suffering from the disease known as "annual deficits." They approach so near death that when existence ceases and the scrap heap receives them we hardly notice the change.

A few, either sturdy by nature or well provided with nurses and doctors, live and grow to the mature form known as "dividend payers." These are rare, much rarer than need be if proper skill and experience were only called in at the birth of all. Like other infants their death rate is very high in the early period of existence. Fortunately, in this case, the doctor has no compunctions about letting an unpromising one die, or drowning it if need be for the good of the community.

The net results to date have been disappointing. The rewards have been few. The number of plants that have been on a real dividend paying basis for the last two years could be counted on the fingers of a one-armed man. On the other hand the crop of white hopes is large. The prospects of the industry as a whole were never better than they are to-day. A very respectable number of plants have recently started or are in process of construction. Most of these efforts show a very decided advance over those we have seen spring up in previous years.

In nearly every case, actual construction of the plant has been preceded by an investigation of the process by competent and experienced experts, and also by tests on a semi-commercial scale. Then there has been careful study of the type of raw material to be used, of the finished products and their marketability as judged by men who were thoroughly familiar with those markets, and finally an intelligent adaptation of the design and construction and location of the plant, both to the raw materials and to the finished products.

These may seem like too childish or self-evident precautions to receive special mention here, but it must be confessed that in the past, often many and sometimes all of them have been overlooked in constructing plants. Too often the process has been the whole thing. The man with a process and a long array of figures showing what wonderful products in amount and value he could get from a cord of wood, has frequently carried the investor by storm. Without waiting for the advice and experience of men who have spent years in the industry, as he would normally do in any other line, the investor plunged. There must be something fascinating about the industry, like there is about buying a mine, but nowadays even the mine buyers have an expert examination and report before buying.

A man with a process and a man with money sat at a mahogany desk. The process and the usual array of figures were there. The figures showed that from a cord of wood, kind and quality not specified, you could get wood alcohol, acetone, acetic acid, acetate of lime, turpentine, rosin, tar, pitch, oil of tar, creosote oil, sheep dip, disinfectant, wood preserver, cable coating, pine oil, liniments for man and beast and rheumatism, cough syrups, patent medicines, embalming fluids and gaulacol, and high-priced drugs of the pharmacopoeia.

Of course, it was all more or less true. The beauty of it is that you *can* get these, but the question is, do you want to? This particular investor did want to. He verified some of the prices from "Merck's index" and the "Oil, Paint and Drug Reporter" and added the long column of figures.

The fact that if you used up your raw material in making some of the products you couldn't have it left to make the others too, didn't occur to him. Further, he forgot that few of the products could be obtained from all woods, that some

would certainly be obtained from certain woods in an entirely unsalable condition, and others would cost more to make and refine in most cases than they could be sold for; still others would glut the market from one day's run of his plant, still others would probably be so different from standard articles, that their use would be restricted to special industries at much lower prices.

Overlooking all these things he followed a common practice of investors in arriving at a conclusion. He took the huge total of returns and deducted 50 per cent, then he deducted 50 per cent of the remainder. After providing this factor of safety, the total was still so large that it seemed certain a handsome profit would result no matter how it was handled. In fact, it seemed a shameful waste to use mahogany desks for office furniture when such rich returns could be obtained by distilling them. After the plant was built it was my painful duty to clear away the sad remains before it had produced a thousand gallons of anything or sold a cent's worth.

This was years ago, but I could cite many almost as bad in more recent years. There was a plant that ran intermittently almost a year without ever producing a single gallon of really marketable goods, and the men in charge did not know tar from pitch, and still no one was called in during that whole period to advise them.

There was another plant with a marketable process where the plant was designed and built complete, and only after the initial operation was started did they discover a glaring mechanical defect that made the plant as constructed, an impossibility.

There was a plant built to handle mill waste where the yields were small but a profit possible because the raw materials cost nothing, and then the whole was laid out so that every stick of wood had to be *handled twice by hand* before reaching the plant.

There was a plant constructed where raw material in the shape of lightwood was "countless as the stars," and investigation showed later that if lightwood of the kind desired was meant, the stars must have been counted on a very cloudy night.

But why continue the enumeration? It seems like a record of one case after another of useless waste. And we wonder about the high cost of living. Literally the amount unwisely expended in this industry amounts to millions. And it is not because we are lacking in competent chemical and engineering information and experience here. It is chiefly because the idea of the *process* has been overrated.

The man who can invent and think out a real idea suitable for a process has done his share. In nearly every other industry we realize that the type of mind capable of inventing is not the type most suitable for designing and constructing or managing a plant or for developing and criticizing his own process.

Why in this industry do we so magnify process and expect the inventor to be more superhuman than we do elsewhere? Why do we load him with the whole responsibility of success or failure instead of calling in the best man available to his assistance? And why do we invest in this industry on an interested man's mere word without calling in competent advice?

Within this present year I talked with an inventor anxious to interest capital in his process of recovering turpentine and rosin from wood. The conversation developed his supposition that no one else was doing this or had workable methods for doing it, and that the only thing he had to compete with was the destructive distilling type plant of thirty years ago. If the inventor, supposed to be expert in his subject, thinks this, why should we be surprised if he convinces the investor?

The process is one element of success; a good process is necessary, yes, but it is only one element, and there are many others. The character and cost of the raw material is most important and deserves an article by itself. The marketability of products has been overlooked again and again on the assumption that turpentine is turpentine and rosin is rosin. Sometimes they are, and sometimes they are not. The fact that John Jones sold his druggist a gallon of pine oil for ten dol-

lars doesn't give you a clue as to its price and consumption in tank car lots. If you read that light oil or heavy oil is quoted at fifteen cents per gallon, that does not indicate that you can sell light oil or heavy oil from white pine at that price, or necessarily at any price.

There is really just one way to determine the marketability and price of a product and that is to sell it or products identical with it on the open market. And then, as mentioned before, the location of plant as related to raw material and market, the design, construction and management should all be done in consultation with men who have been handling this same type of installation as chemists and engineers.

A frequent mistake lies in placing the problem before an analytical chemist. He may be an exceptionally good chemist, and may give accurate reports that will be entirely misleading through no fault of his own. I have before me now such a report on a process made by a thoroughly competent analytical chemist. The report so far as I can see was correct in all respects. Unfortunately, however, from lack of specific information which he could not be supposed to possess, the report indicated success to the chemist and to the investor. A single reading of it would have convinced a man experienced in the industry that nothing but failure lay ahead, unless many things were radically changed. The report actually led to one more useless expenditure of money.

It seems probable that nearly every possible error has been made at some place or time. Most of the mistakes could have been avoided if competent experts, of whom there are a sufficient number, had been kept in touch with the work from its inception. Unfortunately in many cases the man of training and experience in this particular industry has never been called in at all. In other cases he comes to give decent interment or shorten useless suffering. Why not call him in at the beginning, in time to announce to the proud father the arrival of a bouncing baby boy?

New York City.

Wet and Dry Grading.—In testing ore for a metallurgical process, dry screening will give erroneous results if the actual practice is to be wet screening or hydraulic classification. The test should conform to the proposed practice. The following table will illustrate the point. It shows the grades obtained by wet and dry screening an ore which was tested by Morris Green, who gives his results in the September *Journal* of the Chemical, Metallurgical and Mining Society, of South Africa. The ore evidently contains some material sufficiently brittle to be broken by the impact of dry screening.

Mesh.	Dry Grading.	Wet Grading.
—30	22.15%	25.7%
—30 —60	31.55	34.9
—60 —90	9.70	9.0
—90 —120	3.20	2.9
—120 —200	9.13	7.2
—200	24.27	5.2
Slime		15.1
	100.00%	100.0%

Flotation of molybdenum ore by the Minerals Separation process has given the following results: The ore assayed 1.82 per cent MoS_2 , from which by treatment in five successive units, a concentrate was obtained containing 92.08 per cent MoS_2 , with a recovery of 76 per cent of the molybdenum in the concentrate. The ratio of concentration was 100 to 1.66.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in September, 1912, was 63. The total quantity of ore milled during that period was 2,131,593 tons. There were 9970 stamps in operation, with an average duty of 8.36 tons per stamp per 24 hours. Tube-mills in commission numbered 273. The yield for the month was 747,893 fine ounces gold, which was slightly lower than for four preceding months. Fewer stamps and more tube-mills were in commission than at any time during the year.

Cyaniding Slimy Ore by Continuous Decantation

A review of the status of this important recent departure in cyanide practice, by H. C. Parmelee

THE early history of cyanidation is replete with instances of complete failure or great loss when treating ore which tended to form impermeable slime. Leachable sands were treated successfully by percolation, and in some instances it was possible to mix a portion of the slime with the sand and thus avoid a total loss of the precious metals contained therein. In other cases, however, it was temporarily more economical to discharge the slime and impound it against the time when some successful method of treatment should be devised. In this connection the revolutionary influence of agitation and vacuum filtration is a matter of common knowledge.

But great as was the effect of vacuum filtration in overcoming the obstacles in slime treatment, and important as it still is, and will be, in the cyanide industry, the more modern system of continuous decantation¹ is proving of almost equal importance in handling slime. The cyanide process is now the prevalent method of gold-ore treatment; and as the simplicity, economy and ease of operation of the continuous decantation system become better known, it is safe to predict its wider use. It has the advantage of low cost of installation and operation, and it can be combined with existing milling plants without great difficulty. In some instances it displaces filtration, other than clarifying; and where filters are used it permits the delivery of a low-grade pulp to the filter, thus minimizing the loss of dissolved gold in the filter tailing. It will be apparent, also, that the use of continuous decantation preceding filters will increase the capacity of a given plant of the latter by avoiding the time required for washing with barren solution. Eliminating this step would also decrease the cost of a filter-plant. Further, the operation of continuous decantation is simple and flexible, lending itself to a variety of conditions which may be reasonably ascertained in advance.

Continuous Decantation at Ophir.

A case in point is the recent cyanide practice at the remodeled mill of The Ophir Gold Mines, Milling & Power Company, Ophir, San Juan district, Colorado. The company is operating a group of mines, locally known as the Suffolk property, which has a record of production extending over 3 years, and aggregating over \$2,000,000 in value. Considering the situation of the property and the prevailing metallurgical methods it was natural that during the early years of its history, only high-grade ore was mined and milled. Recovery was effected al-

most wholly by amalgamation, with some concentration. The value of the ore treated ranged from \$15 to \$100, and the recovery probably did not exceed 60 per cent. Under the present system of cyaniding, without amalgamation or concentration, ore averaging \$3 gross value per ton can be mined and milled at a profit, with a recovery of over 90 per cent.

Nature of the Ore.

The Suffolk ore consists of quartz, decomposed porphyry and andesite, and contains gold to the value of from \$8 to \$12 per ton. Silver is found only in negligible quantities, and practically no other metals are present. Analyses show over 90 per cent insoluble. The ore is soft, a great deal of it being in the form of clean clay resulting from the decomposition of the porphyry. This condition readily explains the failure of former attempts at cyaniding by leaching. It also gives rise to much colloidal slime in the present system of treatment, but no difficulty is experienced in handling it.

Preliminary Experiments.

Preliminary experiments revealed certain ideal qualities for cyanidation not found in all ores. There are no reducing agents present and the ore is only slightly acid. Chemical consumption of cyanide is only 0.1 lb. per ton, and of lime 1 lb. per ton. Being soft, the ore is readily ground to slime. This is an important characteristic because the gold is very finely divided, and, after crushing, is found for the most part in the fine slime. Despite the fact that the ore contains a great deal of colloidal matter, and therefore does not settle readily, no difficulty has been experienced in handling it by the present method of continuous decantation.

An interesting feature disclosed in the tests was the fact that the quantity of readily soluble gold varied directly with the gross value of the ore, while the quantity of gold requiring time and agitation for its solution was almost constant regardless of the initial value of the ore. Thus it was found that

on all grades of ore the gold went into solution almost immediately to a point where the residues had a uniform value of \$2.40 per ton.

These residues yielded their gold on further agitation as is indicated in the time-solution curve shown in Fig. 1. From this it will be seen that solution goes on quite rapidly, giving residues containing twenty cents gold per ton after 36 hours agitation.

This is the standard now maintained in mill operation, although as a matter of fact "trace" tailings frequently are obtained.

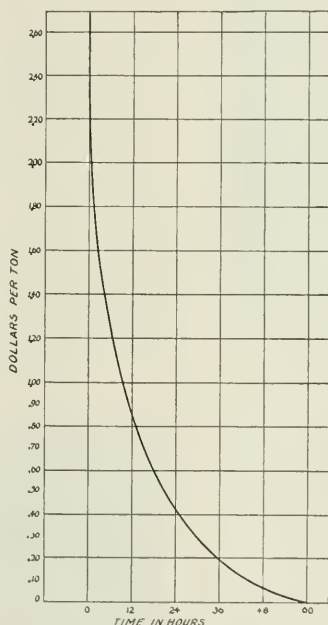


FIG. 1.—SHOWING DECREASE IN GOLD CONTENT WITH INCREASED TIME AGITATION WITH CYANIDE SOLUTION. 1 LB. KCN AND 2 LB. CAO PER TON SOLUTION. TWO PARTS SOLUTION TO ONE OF ORE. LOSS KCN 0.1 LB. PER TON ORE. LOSS CAO 1.0 LB. PER TON ORE.

¹Articles relating to continuous decantation have appeared in this journal for July, 1911, pp. 365 and 373; Sept., 1911, pp. 436 and 439; Nov., 1911, p. 605; Jan., 1912, p. 27.

Crushing in Cyanide Solution.

The present milling system, based on many tests and calculations, is shown concisely in the flow-sheet, Fig. 2. In this connection it will be interesting to note that the results obtained in practice checked closely those indicated by the calculations. The flow-sheet shows the strength, value and tonnage of solution and pulp at different points in the process, so that it will be readily understood without much description. The figures are based on an assumed ore-value of \$5 per ton.

The milling process begins by crushing the ore in cyanide

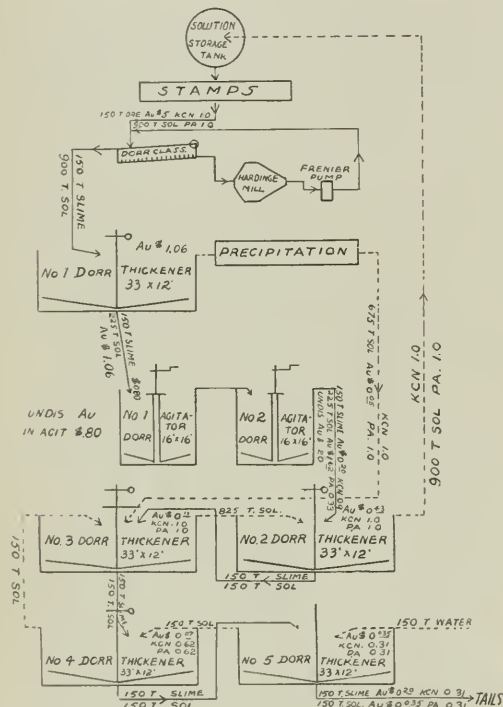


FIG. 2.—FLOW SHEET OF OPHIR GOLD M. & P. CO.

solution with twenty 850-lb. stamps. The height of drop is from 6 to 7 in. at the rate of 104 drops per minute. With $\frac{3}{4}$ -in. screens on the mortars the twenty stamps have a capacity of 150 tons in 24 hours, and 30 per cent of the discharge will pass a 200-mesh screen.

Considering the nature of the ore, the stamping is not so much a problem of crushing as of disintegrating and washing out the clay.

The stamp product is separated into sand and slime in a Dorrl classifier, the sand being reground in an 8-ft. Hardinge conical mill. The latter carries a 4-ton charge of pebbles and runs at 31 r.p.m. Its discharge flows to an Abbe-Frienier pump which returns the pulp to the classifier. The classified slime is elevated by centrifugal pump to the first Dorrl thickener, 33 ft. diameter and 12 ft. deep, running at the rate of one revolution in 12 minutes. Solution of most of the gold occurs so rapidly in the stamp-mortar and tube-mill circuit that the overflow of this thickener is clarified and precipitated by zinc dust. The consumption of the zinc amounts to 0.15 lb. per ton of solution.

The rich solution is measured before precipitation, by means of an all-iron Worthington direct-displacement water-meter. An automatic sample is taken both before and after precipitation. The difference between the assay-value of these samples,

multiplied by the tonnage registered on the meter, gives the amount of gold precipitated during a given period.

Agitation.

The thickened slime, containing $1\frac{1}{2}$ parts solution to 1 of solids, flows to the agitators which are shown in Fig. 3 as they appeared before being enclosed in an addition to the old mill building. The agitators are of a new type, combining air and mechanical agitation, patented by Mr. J. V. N. Dorr. The mechanism is applicable to flat-bottom tanks of ordinary proportions; those at Ophir are of wood, 16 ft. high and 16 ft. diameter. The driving gear is supported on a frame-work at the top of the tank as shown. The central air-lift pipe is connected with the vertical driving shaft and revolves with it, being suspended in the tank. The open bottom of the central pipe reaches within 8 in. of the bottom of the tank and is directly above the inlet-valve for compressed air. Attached to the air-lift pipe near its lower end are the agitator arms to which are riveted angle-irons so placed as to draw pulp from the periphery to the center of the tank during the revolution of the mechanism. The arms are adapted to be lifted out of the pulp in case operation is discontinued for any length of time. The pulp discharged from the air-lift is directed into inclined radial launders attached to the top of the central pipe and revolving with it. The bottoms of the launders are perforated to distribute the discharge over the surface of the pulp.

In operation, therefore, there is a combination of air and mechanical agitation. The arms, which in this case revolve only once in 54 seconds, continually draw thickened slime from all parts of the tank and deliver it to the air-lift for circulation and agitation. This agitator is giving entire satisfaction. It avoids the inconvenience in mill construction due to the great height of Pachuca tanks, and it overcomes the difficulties arising from accumulation of sand, which frequently occurs in cone-bottom agitators.

It is recognized that in many types of agitators the intensity of agitation is determined more by the mechanical necessity of keeping the pulp in suspension rather than by the chemical requirements of the ore under treatment. This frequently results in a waste of power; and where air-agitation alone is used, the excess of air may readily be a disadvantage. This agitator



FIG. 3.—AGITATORS, OPHIR MILL.

maintains the pulp in suspension under every condition, and allows the intensity of agitation to be regulated to suit the needs of the ore.

Continuous Decantation.

The agitated pulp is subjected to a system of continuous decantation and counter-current dilution as outlined in the flow-sheet. The final tailing is discharged from the last thickener

without filtration. The loss in dissolved gold at this point is but three cents per ton, which compares favorably with similar loss from filters, the latter ranging from four cents up as high as twenty cents or more per ton. The discharged pulp contains from 48 per cent to 54 per cent solids. In applying this system, a vital thing to be observed is the point at which barren solution is added; and this is governed by the relative value of the pulp in cyanide and dissolved gold, and the proportion of each that may be most economically discharged in the final tailing.

It will be noticed here that in the present case the barren

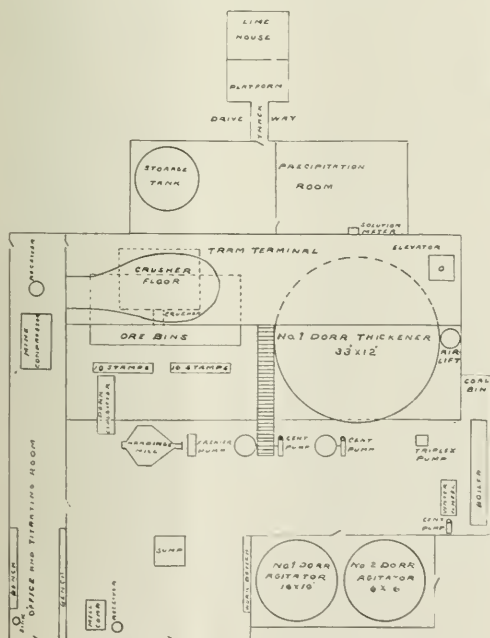


FIG. 4.—PLAN OF MILL, OPHIR GOLD M., M. & P. COMPANY.

solution is added in the third of the five thickeners. This is based on experiments and calculations designed to give the lowest combined loss of cyanide and gold, and is not necessarily the exact scheme which would be followed with other ores and solutions.

By reference to the flow-sheet it will be seen that the flow of solution through the entire mill system is in a closed circuit, except for that portion of solution discharged as moisture in the final tailing. This is compensated by the addition of hot

water from the heating system, so that the solution is kept at an average temperature of 85 deg. F. The quantity of hot water thus introduced is governed by a float in the mill-solution storage tank, which operates a throttle valve in the hot water line.

The float is placed in this tank because it is the only one in the entire system in which the level of solution varies. It is also the only tank in the system besides the treatment tanks. Sump-tanks are not needed.

Air-Lifts for Transfer of Pulp.

An important feature at Ophir is the convenient arrangement and control of the air-lifts used for transferring pulp from one thickener to another. These lifts are similar to those used at Real del Monte, Pachuca, Mexico. The four thickeners are symmetrically arranged as shown in Figs. 4 and 5, and in the central space are grouped the air-lifts and boxes which receive the various pulps and solutions. This arrangement greatly facilitates inspection of the entire system of continuous decan-

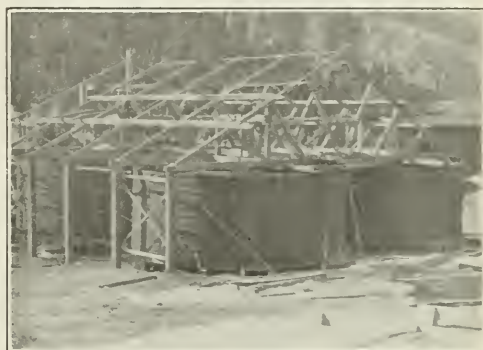


FIG. 5.—SHOWING ARRANGEMENT OF THICKENERS IN CONTINUOUS DECAN- TATION DEPARTMENT. OLD MILL IN RIGHT BACKGROUND.

tation. The thick underflow discharged through a nozzle, and the overflow to be mixed with it, flow into a sump-box, and the mixture is elevated to the proper tank by an air-lift.

The construction of the air-lifts, of which there are three, is shown in the drawing, Fig. 6. The thickened pulp and diluting solution which are to be transferred to any one of the thickeners are received in one compartment of a box which is 2½ ft. square and 3 ft. deep. Attached to the bottom of each compartment, and extending downward into a concrete-lined well, is a 9-in. pipe, 23 ft. long, capped at the bottom. This gives a total submergence of 25 ft. below the normal level of slime maintained in the box. Inside the 9-in. pipe, and extending downward within 4 in. of its bottom cap, is a 5-in. pipe which rises also 13 ft. above the surface of the slime in the box and discharges the elevated pulp to the feed box of the desired thickener. The inner pipe is supported on legs resting on the cap of the outer pipe. A 1-in. compressed-air pipe is introduced into the top of the 5-in. pipe and extends downward within 10 in. of the bottom of the 5-in. pipe.

In each compartment of the box referred to is a float rigidly connected by an iron rod with a throttle valve on the air line. As the pump accumulates in any compartment, the float rises, opens the valve and operates the air-lift. When the level of slime in that compartment has been lowered, the float automatically closes the air valve and the elevation of pulp is temporarily discontinued. These pumps have proved effective in pulp transference and are economical in air consumption. They require little or no attention, and the only cost is for power.

Air is supplied at 30 lb. pressure, and under this condition the consumption of air is 30 cu. ft. per minute for the three lifts. In this connection it may be stated that throttle valves proved superior to butterfly valves, as the latter did not close as

tightly nor as quickly as the former, thus allowing an escape of air following each discharge of pulp.

In order to prevent the violent discharge of the air-lifts from agitating the settling pulp in the thickeners, a wooden float is placed in each feed-box to break the force of the incoming stream. The central feed-box is 3 ft. square and 5 ft. deep, dipping about $3\frac{1}{2}$ ft. below the surface of the pulp. The wooden float clears the sides of the box by a few inches and has a centrally-bored hole through which the driving shaft of the thickener passes.

Another system for transferring pulp may be noted here.

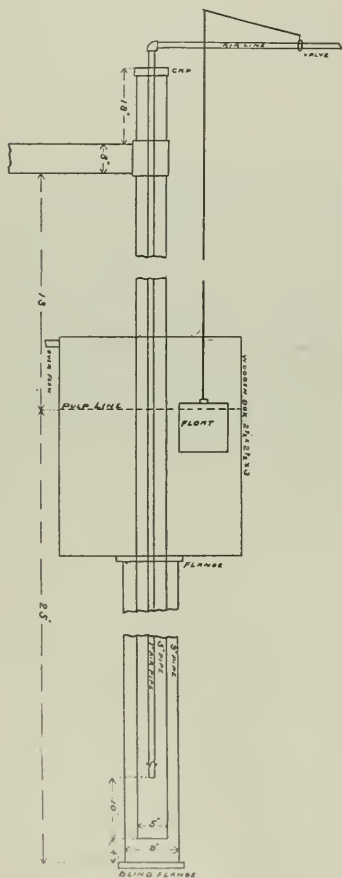


FIG. 6.—AIR-LIFT FOR TRANSFER OF PULP.

This consists in withdrawing the thick pulp by means of a diaphragm or other displacement pump set near the top of the tank. The pump discharge is then mixed with the proper overflow which flows by gravity from an adjoining tank.

This method gives closer regulation of the thickened discharge, as the withdrawal of the same volume in unit time means the delivery of more solid if the pulp becomes thick, and less as it becomes thin. The disadvantage of this system is the wear of diaphragms, which is partly compensated by the smaller power required.

Power, Labor and Sampling.

The property is favorably situated for the use of inexpensive water power, of which from 800 to 1000 hp. can be developed. At present two Pelton wheels are generating all the power required by the mill and the mine compressor. One 6-

ft. wheel with two $3\frac{3}{4}$ -in. nozzles is operating under a head of 100 ft., and one 3-ft. wheel with a 2-in. nozzle is running under a head of 375 ft. The mill is consuming about 110 hp. when treating 150 tons per day.

The labor normally required per shift in the mill is two men, with a repairman on the day shift, in addition to which there is a mill superintendent.

An automatic sampling system has been installed, controlled by a master-clock. The samplers are placed to receive the battery discharge, agitator discharge and final tailing from the last thickener. Samples are taken simultaneously for one second of time at half-hour intervals.

The work at this mill is worthy of note on account of the simple system of ore-treatment, which gives the best results at a low operating cost. The unit cost of installation, which in the last analysis must be distributed as an additional operating cost over the total tonnage to be treated, is considerably lower than would be the case for the usual filtration system.

We are indebted to Mr. George B. Pickett, general manager for the company, for permission to publish these data; and to Mr. Justin H. Haynes, general superintendent, and Mr. F. L. Veatch, mill superintendent, for supplying the details of the process as they have applied it to local conditions.

The Determination of Phosphorus in Vanadium Steels.

By E. W. Hagmaier.

The determination of phosphorus in the presence of vanadium has been a bugbear ever since vanadium steels have come into extensive use. The methods have either been long and tedious or not very reliable. The following method besides being short has proven accurate under all conditions in which it has been used:

The sample of steel is dissolved in aqua regia, taken to dryness and baked. When all the acid is driven off it is then taken up in strong hydrochloric acid; when complete solution is obtained the silica is filtered. The filtrate is then reduced with sulphurous acid. When entirely reduced 5 cubic centimeters of acetic acid 90 per cent and 10 cubic centimeters of cerium chloride saturated solution are added. Ammonium hydrate (about 1 part concentrated ammonia to 3 of water) is added drop by drop with constant stirring, until turbidity is shown. The solution is then heated to a boil, allowed to settle, and filtered. The cerium phosphate will filter rapidly, especially through a black ribbon paper. The precipitate is washed five or six times with hot water and then dissolved off the paper with hot (1-1) nitric acid, and the phosphorus precipitated as in other cases with ammonium molybdate.

Precautions:—It is better to filter off the silica before precipitating the cerium phosphate, for the presence of the silicon causes the cerium phosphate to filter slowly.

The point where most caution is required in the whole method is in adding the ammonium hydrate. This must be added very slowly, for if once too large an amount is added it seems impossible to obtain proper conditions by coming back again with hydrochloric acid. If by accident too much ammonium hydrate is added, the best plan is to filter the precipitate, redissolve it in hydrochloric acid, pass in sulphur dioxide, and reprecipitate the cerium phosphate.

This method has been tried with vanadium from 0.166 up to 5 per cent, and it has been found that in all cases above 0.5 to 1 per cent one reprecipitation of the cerium phosphate should be made, and with 1 per cent to 5 per cent vanadium two reprecipitations.

The following experiments were carried out by the above method, and in no case where the necessary precautions were taken did the ammonium phosphomolybdate precipitate show the orange color characteristic of vanadium contamination. The ammonium phosphomolybdate precipitation was also taken, redissolved and tested for vanadium, and no trace of vanadium was found.

In all of the following experiments the molybdate precipitate was filtered on a weighted paper and the phosphorus calculated using .0163 as the factor.

One gram of a steel having 0.60 phosphorus and 0.16 vanadium added. The following results were obtained: No. 1, 0.633 per cent P; No. 2, 0.596 per cent P; No. 3, 0.595 per cent P; No. 4, 0.614 per cent P.

One gram of steel having 0.60 per cent phosphorus and 0.767 per cent vanadium. The following results were obtained: No. 1, 0.585 per cent P; No. 2, 0.570 per cent P; No. 3, 0.625, per cent P; No. 4, 0.640 per cent P; No. 5, 0.572 per cent P; No. 6, 0.619 per cent P.

Ten grams of steel having 0.025 per cent phosphorus and 0.166 per cent vanadium. The following results were obtained: No. 1, 0.023 per cent P; No. 2, 0.018 per cent P; No. 3, 0.025 per cent P.

One gram of cast iron having 0.532 per cent phosphorus and 0.956 per cent vanadium added. The following results were obtained: No. 1, 0.539 per cent P; No. 2, 0.526 per cent P.

Pittsburgh, Pa.

Notes on Metallurgy in British Columbia in 1912.

By E. Jacobs.

The quantity of ore and concentrate smelted at the Consolidated Company's works at Trail was between 300,000 and 320,000 tons, which was less than in the fiscal year to June 30, 1910 (487,000 tons), and the next following fiscal year (389,000 tons). The decrease was caused by loss of the Snowshoe mine ore, of which there was 268,000 tons, received in the two fiscal years quoted. Improvements included an addition to flue chamber, to secure better results in settling of flue dust from copper ores; rebuilding of matte-handling plant, the method of treating low-grade matte having been changed; alterations to lead-sampling plant, made necessary by increase in quantity of coarse lead ore received; more electrolytic tanks in lead refinery, to provide for larger output of refined lead; and numerous changes, chiefly to facilitate handling materials. Experiments in reduction of zinc-lead ores were carried out, but no provision was made for production of spelter on a commercial scale.

Copper Smelting.

Last February, at the Trail smelter, experiments were commenced to smelt the raw matte pyritically, to take the place of the former more expensive method of roasting and sintering, and if possible without any considerable alteration to the furnaces already in use. The smallest furnace, dimensions 42 by 240 in., was blown in on a charge consisting of matte 2,500 lb., silicious ore 1,500 lb., lime rock 12 per cent, and coke 3 per cent. This furnace was blown out after having been in blast 12 hours, as it ran so fast the matte-tappers could not keep the matte down in the settlers. It was decided to cut the furnace in two, which was accordingly done, making the area at the tuyeres 42 by 120 in. The smaller furnace was then blown in on a similar charge, the coke being shoveled and the remainder of the charge dumped in from the side and distributed along the furnace by the locomotive in the same manner as for the ore-furnaces. The matte on the charge was coarse, about 6 in., while the silicious ore available was very fine, 50 per cent passing through a $\frac{1}{4}$ -in. screen.

The furnace was run on this charge for two days but never got properly heated, owing to the looseness and shallow depth of the charge. The next run was tried with a heavier charge and with matte crushed to 2 in., and, while this acted better, the sides of the furnace rapidly crusted, owing to the fine nature of the silicious ore. After several attempts with long-nosed cars and distributing plates, to try to get the charge into the center of the furnace, the ends were taken out of the hood and the furnace fed along the center with bottom-dump cars. This method proved successful. Two charge cars are used, and the charge of 4,200 lb. matte, 2,000 lb. silicious ore, 13.5 per cent lime rock, and 4 per cent coke is divided equally between the two.

The coke is fed by shoveling and the remainder of the charge is weighed in the cars—matte first, then ore, and lime rock on top. If the furnace shows signs of crusting the matte is weighed on top of the ore for a time, thereby throwing more matte along the sides, which invariably has the desired effect of cutting out the crust.

The matte is concentrated twice in this furnace, the first concentration running from 15 to 20 per cent copper, equivalent to a reduction of from 2.5 to 3 into 1. This matte is returned to the furnace and smelted with sufficient low-grade matte to keep the grade about 40 per cent copper for the second concentration. All the matte is tapped into Kilker's $\frac{1}{2}$ -ton matte-tapping cars (five of which are used), cooled, and hauled by electric trolley locomotive to the crushing plant.

The pans of the matte-cars dump directly into a 10 by 20-in. jaw-crusher and thence to a pan conveyor which discharges into three iron bins, whence the matte is drawn into V-shaped cars and hauled to the charge bins or shipped to Tacoma, Washington (for converting), according to grade.

Slag from the ore-furnaces assays, approximately: FeO, 14.5 to 16; SiO₂, 40 to 47; CaO, 18 to 22; MgO, 1.5 to 3; Al₂O₃, 14 to 16.5 per cent; S, trace. Slag from the pyritic furnace runs about FeO, 45 to 50; SiO₂, 35; CaO, 12 per cent.

This information was kindly supplied to the writer by James Buchanan, superintendent of the Consolidated Mining and Smelting Company of Canada's smelter at Trail.

The chief change at the Granby Company's smelter was in the substitution of water-granulation of slag for dumping it molten. By a system of conveying belts, operated by electric motors (already described in this journal), the dewatered slag is conveyed up a long incline to a height of 120 ft. and then dumped, this arrangement providing dump room for all slag that will be made for six or seven years. The quantity of ore from Granby mines treated in 1912 was more than 1,200,000 tons. One notable feature of the year's work was that the full battery of eight blast furnaces was operated all the year, one run having been for 156 consecutive days without interruption. The company is purchasing materials and plant, and other equipment for a 2,000-ton smelter, hydroelectric power station, shipping dock, etc., at Granby Bay, Observatory Inlet, where its new mine is being developed with promise of large productiveness.

The British Columbia Copper Company operated its smelter throughout the year. Ore receipts totalled between 650,000 and 700,000 tons. High efficiency was secured, monthly totals of ore smelted having ranged as high as 65,000 tons, with three furnaces in blast. The greater part of the ore smelted came from the Mother Lode and Rawhide mines. There is in the company's Lone Star mine a large amount of ore available, but owing to its high silica content, it has been deemed inadvisable to endeavor to smelt it with that from the company's other mines. For some time past concentration tests have been made with the object of determining how best to eliminate the excess of silica. The ore presents unusual obstacles to concentration, but notwithstanding this the problem is now in a fair way toward being successfully solved. Favorable results will have an important bearing upon the question of ore supply for the company's smelter. In value the general tenor of the ore is higher than that of the average of Boundary district ores.

Most of the copper concentrate and the ore shipped crude from Coast district mines was treated at the smelting works at Tacoma, Wash. This includes the comparatively large output of the Britannia mine.

Gold Milling.

While a large proportion, nearly three-fourths, of the lode gold produced in the province is obtained by smelting ores containing copper, the remaining one-fourth is recovered by stamp-milling and auxiliary cyaniding. The largest stamp-mill operated is that of the Hedley Gold Mining Company, at Hedley, Similkameen, where approximately 70,000 tons of ore was crushed in 1912, as compared with 58,000 tons in 1911. No gold is saved on plates, but fully five-sixths is contained

in concentrates from vanners and tables, while the remainder is recovered in cyanide bullion. Approximate value of gold recovered in 1912 was \$763,000, or an average of rather less than \$11 a ton; in 1911 it was \$680,000, with an average recovery of \$11.75 a ton. Estimated net earnings for 1912 were \$407,500, as against \$308,800 in 1911.

There were three 20-stamp mills operated in Nelson mining division in 1912, but information is not available concerning two. The third is that of the Mother Lode Sheep Creek Mining Company. During four months, July-October, 7,755 tons of ore was crushed. The extraction was nearly 98 per cent; the value of the product was \$102,869—average \$13.265 a ton. Operating expenses were \$7,245 a ton. November and December figures were not received. The mill is modern in its equipment. Higher net returns are expected from next year's operations.

A new installation was that of a 10-stamp mill at the Inland Empire mine, near Paulson, in the western part of Trail Creek mining division. Only a light saving is made on the plates. The daily output of concentrate is 30 to 40 tons, which is shipped to Trail for smelting.

Lead and Zinc Concentration.

The Standard Silver-Lead Mining Company shipped 4,400 tons of sorted ore to the smelter, and concentrated 32,600 tons of mill ore. Products were 5,400 tons of silver-lead and 3,300 tons of silver-zinc concentrates. Metal contents of all shipments were: Silver, 832,000 oz.; lead, 12,440,000 lb.; zinc, 2,706,000 lb. Silver-lead concentrate was shipped to Trail, B. C., and silver-zinc to Bartlesville, Okla. The mill has been run only one-shift per diem, but it is to be operated double shift hereafter.

The Van-Roi Mining Company, also operating near Silverton, Slocan Lake, concentrated 54,000 tons of ore. Average assay of feed showed: Silver, 15.02 oz. per ton; lead, 3.66 per cent, and zinc, 6.26 per cent. Concentrates produced, and metal contents were as follows:

	Metal Contents.		
	Silver, oz.	Lead, per cent.	Zinc, per cent.
Lead concentrate, 2,392.5 tons.....	179.75	60.2	11.1
Zinc concentrate, 2,570.5 tons.....	60.8	3.7	45.1
Tailing	4.6	0.9	4.0

An auxiliary aerial two-bucket tramway was constructed from mine to mill.

At the New Canadian Metal Company's mill, at Riondel, Kootenay Lake, operated in conjunction with the company's Bluebell mine, milling was resumed on July 4, after a long suspension of operations. During six months 33,000 tons of ore was milled; production was 49,600 oz. of silver and 4,172,000 lb. of lead. Capacity of mill is being increased from 225 tons to 300 tons per diem.

Several other concentrating mills were operated, but on a smaller scale than those above-mentioned.

General.

Gold and gold-copper ores are either milled or smelted in the province, with the exception of the latter ores in the Coast district, much of which is shipped to the smelter at Tacoma, Washington. Silver-lead and lead ores are either shipped crude to Trail, or concentrated and the product sent there, the Consolidated Company's lead-smelting and refining works being the only one reducing lead ores in British Columbia. Zinc-lead ores are concentrated and the zinc product, as well as crude zinc ore, is shipped to the United States, chiefly to Bartlesville, Okla., for reduction. The works of Beer, Sondheimer & Co., at that place, receive the zinc concentrate output of the Standard, Van Roi, Hewitt-Lorna Doone, and Monarch mines. The Lucky Jim and Noble Five mines ship "crude zinc" ore, the latter in much less quantity at present than the former. Noble Five zinc ore contains up to 56 per cent

B. C. Of this total, about 70,000 oz. came from British Columbia mines, 2,500 oz. from Yukon Territory (Canada), and 7,500 oz. from Alaska. Much of this gold was from placer mines.

The greater part of the 250,000 to 260,000 oz. of lode—also copper-bearing.

Regarding the bounty paid by the Federal Government on lead mined and smelted in the province, when the London price of lead is £14 or under, the amount of bounty payable is at the rate of 75 cents per 100 lb., but by the sliding scale the amount payable diminishes as the price rises, until, at £18 per ton it is entirely extinguished. During 1912 the lead content of ore and concentrate delivered to the smelter at Trail totalled, approximately, 22,000 tons. Bounty was payable until June, in which month the price rose to £18 and higher, and it remained higher than £18 throughout the remaining months of the year. Consequently the total amount of bounty earned in 1912 was only about \$65,000. The original appropriation made in 1903 was \$2,500,000; of this sum there remains unearned about \$700,000. The bounty payable under the present act "shall cease and determine on June 30, 1913," but those chiefly concerned are petitioning the Federal Government to continue to aid the local lead-mining industry in a similar or some other suitable form.

It may be of interest to note that the Dominion Director of Mines has had one of the officials of the Mines Branch of the Canadian Department of Mines engaged in an investigation of the Canadian market for various mineral products in a crude or partly prepared condition. The official referred to spent part of 1912 in British Columbia, and afterward continued his investigations in other parts of Canada. The ultimate object of this investigation is the further encouragement of the use of Canadian minerals as material in Canadian manufactures. It also includes pointing out to producers the requirements of the domestic market, and indicating the form in which minerals should be prepared for use in the various industries in which they are to be employed. The results of this work will thus set forth what minerals are at present in use in the manufacturing industries of Canada, their sources, the uses to which they are put, the degree of purity required for the various processes in which they are used, and the physical condition in which they are purchased.

In conclusion, it may be added that a preliminary estimate of the mineral production of British Columbia in 1912 indicates that it has been, in both quantity and value of mineral produced, a record year for the province. The estimated total value of the mineral production is \$31,500,000, of which nearly \$17,000,000 is for metallic minerals, and of this less than half a million is for placer-gold. The highest previous annual production was that for 1910—\$26,377,000. Estimates for 1912 show the following quantities of lode metals recovered: Gold, 250,000 oz.; silver, 2,700,000 oz.; lead, 35,000,000 lb.; copper, 50,000,000 lb.; zinc, 6,000,000 lb. The gross output of coal was 3,040,000 long tons, part of which was made into 263,000 tons of coke. Miscellaneous non-metallic minerals, other than coal, are estimated at \$3,800,000. The outlook is favorable for further increase.

The Efficiency Society will hold its annual meeting at the headquarters of the Society, 29 West 30th Street, New York City, on Monday and Tuesday, January 27 and 28. There will be symposia and discussions on management; on democracy in industry and securing the consent of the governed; on efficiency in the government organization, national, state and municipal or a readjustment of the governmental departments; on industrial relations; on the relations between employer, employee and the community; on industrial hygiene and safeguards against occupational disease and accident, and on safety. Mr. James G. Cannon, president of the Fourth National Bank, is the first president of the society. Among the other directors are Messrs. Bion J. Arnold, W. W. Freeman, William Jay Schieffelin and Schuyler S. Wheeler.

Approximately 60,000 oz. of gold was received in 1912 for assaying at the Dominion of Canada assay office, Vancouver,

Protection of Intellectual Property in Relation to Chemical Industry.*

By Dr. Leo H. Baekeland.

Intellectual Property Rights.

The mass of unthinking people, as well as those whose views are predominantly guided by precedent, have little or no conception of the natural rights of intellectual property. It is difficult to teach such people that adequate protection of intellectual property is abundantly more beneficial to the community at large than to the temporary individual possessors of these rights.

Yet these same people consider as sacred and inviolable, any other property rights as soon as the latter relate to chattels or real estate, whether such rights were obtained by purchase, by inheritance, by gift, by privilege, by labor, or by any other way.

Furthermore, the laws of all nations are very strict in protecting such property rights, but do not concern themselves, beyond certain limits, whether the possessor of the property is morally entitled to it or not. Neither do our laws concern themselves whether the owner uses his property for good or for wrong, for the benefit of the community at large, or for the gratification of his own selfish purposes. From the standpoint of the law (with very few exceptions, such as for instance, Board of Health or police ordinances, or cases of so-called eminent domain), it matters little whether the private ownership of some property is a burden to the community or whether it is an impediment to the happiness or the free development of its citizens.

Neither is there any dispute as to the time the ownership of such property should last. Except for restrictions put on ownership by taxes, property rights are practically perpetual, and can only be transferred by accepted methods, as for instance, sale, barter, inheritance, or donation.

In some rare instances, there may be expropriation for public purposes (or eminent domain), but even then, some suitable compensation is usually made.

All this is readily accepted as an axiom, as an underlying article of faith by all laws relating to property. Only the socialist dares dispute these rights, while even the single taxer admits them to such a decided extent that he desires to abolish taxes on all property created by labor or enterprise, so as to shift the burden of all taxation on unearned land values.

When, however, it comes to recognize the claims of ownership to intellectual property, the result of the truly creative effort of the citizen, we butt right away against some stubborn conceptions, which have petrified into the code of our long established laws.

If Tom steals Dick's two-dollar scarf pin, Dick will have little trouble in putting Tom in jail, even if Dick himself has obtained his pin by questionable methods. But when it comes to protect even for the short period of seventeen years, the most logical, the most legitimate personal property, intellectual property as embodied in patent rights, with all that it involves, with enterprises depending thereon, based often on the work of a life-time, then our law courts are woefully deficient, on account of the uncertainties, delays and enormous expenses connected with the adjudication of patent rights. All this works overwhelmingly in favor of the litigant with the well filled purse, the large corporation.

Fundamental American Patent Law.

Yet, no country in the world has expressed in a fairer and broader spirit, the rights of intellectual property, than the United States, in Article I, Section 8, of the Constitution: "Congress shall have power to promote the progress of science and the useful arts by securing for limited times to authors and inventors, the exclusive right to their respective writings and discoveries."

This proclamation lifted the right of a patentee at once far beyond the mere privilege conferred by most other countries, which grant patents not only to the real inventors or originators, but also to those who are first to introduce unpublished inventions into their respective countries. With some legitimate pride, we can say that in this respect at least, American patent law stands head and shoulders above the laws of Germany, France and England.

The principles of the right of intellectual property so clearly defined in our Constitution, were repeated in the preamble of the French Law of Jan. 8, 1791, which declares:

"The National Assembly, considering that every new idea, whose manifestation or development may become useful to society, belongs to him who conceived it, and that not to regard an industrial invention as the property of its author would be to attack the essential rights of man; considering at the same time how much the lack of a positive and authentic declaration of this truth may have contributed till now to discourage French industry by occasioning the emigration of numerous distinguished artists and by causing to pass out of the country a great number of new inventions from which the Empire ought to have drawn the first advantages; considering finally that all the principles of justice, of public order, and of national interest imperatively command that it determine for the future the opinion of French citizens with regard to this class of property by a law which consecrates and protects it . . . etc."

The wisdom of these provisions has abundantly been proved by subsequent events. Only a man stubbornly blind to evident facts, will deny that just those countries which have the most liberal laws for patent protection, are also those which have taken the lead in the industrial and scientific development of the world. No man was more imbued of the benefits of the patent system than Abraham Lincoln, when in 1860, in his speech at Springfield, Ill., he said:

"In the world's history, certain inventions and discoveries occurred of peculiar value, on account of their great efficiency in facilitating all other inventions and discoveries. Of these were the art of writing and of printing, the discovery of America, and the introduction of patent laws. . . . The patent system . . . added the fuel of interest to the fire of genius, in the discovery and production of new and useful things."

Inadequacy of American Patent System.

Up to about thirty years ago, our patent system covered tolerably well the purpose for which it was intended. It stimulated individual inventions and promoted numerous private enterprises. Since then, with the extraordinary growth of our nation, with the tremendous increase of agglomerations of capital for industrial enterprises, and more specially with the astonishing increase in the ramifications of applied science, our patent system has become totally inadequate to the needs of the country: it suits our new conditions in about the same way as baby clothes fit an overgrown boy.

Our patent system, although based on an excellent fundamental law, has now degenerated into a set of exceedingly complicated technicalities of law practice, a system of legal acrobatics, whereby any contestation before the courts can be turned into "perpetual motion" to the advantage of wealthy litigants, and whereby the individual patentees of slender means and the small industrial concerns, find themselves under smothering disadvantages when opposing rich antagonists. In this way, our patent system, instead of accomplishing its intended purposes of stimulating individuality, simply reinforces the rich and big industrial enterprises, and discourages the individual inventor unprovided with a liberal bank account.

New Rules of Supreme Court.

It is true that on November 4, 1912, the Supreme Court of the United States, has promulgated revised Rules of Practice for the Courts of Equity, which intend to simplify our methods of litigation. Unfortunately, this is only a half-way measure, leaving still abundant opportunity for the tactics of delay.

* Presidential address delivered before the American Institute of Chemical Engineers at the Detroit meeting, December, 1912.

chicane, and expense which have too much disgraced American patent litigation.

These new rules might gain in efficiency, if they were supplemented by the creation of a final court of patent appeals. They might be made incomparably more efficient if they could be strengthened by a system whereby the adjudication of the validity of patents does no longer devolve upon judges who do not possess the technical or scientific preparation required nowadays for discerning the merits of complicated patent questions. Some of the far-reaching details of scientific technology absolutely baffle the comprehension of those who have no preliminary technical or scientific training. Certain problems of chemistry and physics involved in many patent suits, can no longer be understood by an intelligent judge, if he has not had long and systematic preliminary training in that branch of knowledge. I do not deny that an intelligent judge can be coached and instructed by long, tedious, time-robbing methods, even in intricate scientific problems; but his education has to be made over again for each special case. After you have made a chemist of him for one case, the next adjudication will require the knowledge of a physicist, an electrician, an engineer, and so forth.

What would any judge say of a chemist or a mathematician, or an engineer, totally ignorant of the practice of law, who tried to conduct a law case in court? Such an amateur lawyer might succeed in doing so, but to what hopeless loss of time, misunderstandings, and confusions would this lead before the subject had been mastered to some extent. Yet this is exactly what happens with a judge to whom we entrust to decide on the validity of a patent involving highly intricate scientific or technical subjects.

Judge Hand expressed himself very eloquently on this subject:

"I cannot stop without calling attention to the extraordinary condition of the law which makes it possible for a man without any knowledge of even the rudiments of chemistry to pass upon such questions as these. The inordinate expense of time is the least of the resulting evils, for only a trained chemist is really capable of passing upon such facts, *e.g.*, in this case the chemical character of Von Furth's so-called "zinc compound," or the presence of inactive organic substances. In Germany, where the national spirit eagerly seeks for all the assistance it can get from the whole range of human knowledge, they do quite differently. The court summons technical judges to whom technical questions are submitted and who can intelligently pass upon the issues without blindly groping among testimony upon matters wholly out of their ken. How long we shall continue to blunder along without the aid of unpartisan and authoritative scientific assistance in the administration of justice, no one knows; but all fair persons not conventionalized by provincial legal habits of mind ought, I should think, to unite to effect some such advance."

(See *Parke-Davis & Company vs. H. K. Mulford Company*, Circuit Court, Southern District of New York, April 28, 1911—189 Federal Reporter, 95.)

Even under the new rules, it will not be difficult to drag on a case by presenting an unrestricted amount of testimony taken before an incompetent examiner, and by calculating every step so as to tire out your opponent, and so as to induce the judge in doubt and error, by swamping him with endless contradictory expert testimony calculated to befog the issue instead of making it clear. Such tactics are relatively easy for the litigant who, for that purpose, can afford to pay accommodating experts and skilful lawyers. Even if at the end, the judge, after laborious and conscientious efforts, masters the technicalities of the case and reaches a good decision, much needless time has been wasted. All this might easily be avoided, and judges might be saved the trouble and responsibility of going in every single case through a different scientific or technical training, if their intervention could be limited to what they are more competent for, namely, to determine what claims of patents have been infringed and in how far this infringement entitles the patentee to damages.

German Practice of Settling Patent Suits.

That such a method of settling patent suits is quite practical, is shown by the example of Germany. In that country, patents are allowed after preliminary examination, just like here; but, after the patent is granted, it can be attacked for annulment or revocation before a competent court in the patent office. So that any party who is sued for infringement of a patent which he thinks is invalid, can avoid temporarily the adjudication of the infringement issue by starting an annulment or revocation suit. In the meantime, the courts in which infringement cases are examined have to take the patent as it stands, and it is only left to them to interpret the scope of the claims, and to what extent these claims have been infringed.

This relieves the equity court of all the complicated questions of validity or non-validity of a patent, and puts this whole matter in the hands of a properly constituted court of experts who can handle this subject with incomparably less hesitation, or delay.

Besides this, the whole system of practice in the German Patent Office, tends toward systematic elimination of invalid patents. After an examiner has decided upon preliminary allowance of a patent, the claims and specifications are open for public inspection, and for a period of two months, anybody who-soever can file arguments against the final grant of the patent. In this way, the nation does not confer too lightly patent privileges and has furthermore, the benefit of the free advice of any experts in the art, who may advance good reasons for non-allowance of the claims, of which the examiner was not aware, when he rendered his first decision.

These opposition proceedings give added thoroughness to the work of the examiners. They are relatively inexpensive and do not necessitate the intervention of law counsel. Sometimes they delay the issue of a patent, if there is any good reason for doing so. On the other hand, a patent that has successfully withstood vigorous opposition proceedings is very much strengthened thereby; this, in itself, is a very valuable compensation for any delays to which the patentee may have been subjected. In other words, by that system, a good patent becomes stronger, while a defective patent application is easily weeded out. A similar system of public opposition exists here in the United States in relation to the granting of trademark rights, and seems practical enough that it could be extended to our methods of allowing patents.

Such a sifting process, first by the examiner, then by opposition proceedings, sometimes by annulment or revocation proceedings, for wrongly issued patents, involves no serious difficulties nor great loss of time if carried out by courts of experts. Thanks to such a system, the work of a judge who acts on an infringement case, gains considerably in dignity and is, at the same time, enormously shortened and simplified. (See *Wertheimer, The German Patent System*, Electrical World, May 18, 1912.)

Proposed Changes of American Patent Office.

The German system throws the burden of technicalities and expert knowledge on the patent office, or the courts connected therewith. Nothing would be easier than to introduce a somewhat similar system in our country:

All officers of our patent office, high or low, should be made independent of any political favoritism; they should be better paid, with more opportunity for promotion, according to merit; their work should be made simpler by an improved office equipment and increased facilities for a thorough search; furthermore, our unnecessarily complicated and expensive methods of interference proceedings should be simplified.

With these reforms, there is no doubt that we can organize right in the patent office, a competent court, supplemented by the court of appeals of the District of Columbia, for deciding in a very expedient way, all questions of validity of patents.

This court of appeals, because it is situated right in Washington, would have easy and immediate access to all the records of the patent office; by this fact alone, it would have superior opportunities for prompt and efficient work.

During the late years, Germany has been trying to broaden its patent laws more and more toward the principles set forth in the American Constitution. For instance, it has practically eliminated the system of compulsory licenses except in some rare instances where public welfare is involved. If only we could borrow some of the more efficient methods with which the German patent law is administered and enforced, we might succeed in making an American patent *real* property for poor and rich alike, instead of a pretext for expensive and endless litigation with all the advantages it gives to the richer litigant, to the detriment of the consumer, who in the end, pays the bill.

Deficiencies of Proposed Oldfield Bill.

At least some of these facts seem to have been very well recognized in the masterly report of Hon. William A. Oldfield, chairman of the House Committee on Patents. (See report No. 1161, on H. R. 23,417, Aug. 8, 1912.)

Unfortunately, his proposed Oldfield Bill (H. R. No. 23,417) with a regrettable lack of consistency, neglects utterly the paramount issues, and busies itself with secondary regulations which if carried out will practically put a penalty on patented articles.

The new provisions of the Oldfield Bill aim at curtailing the power of patents in the hands of trusts or large corporations; but, in doing so, new provisions are introduced which will create endless new opportunities for protracted litigation.

The Oldfield Bill overlooks the axiom that whatever increases the expense or delays of litigation, is a very potent weapon in the hands of large corporations, which they can hurl against the poor litigant who stands in their way.

The saddest thing of all is that the new Oldfield Bill tries to abrogate the hitherto accepted principle established by our Constitution, that a patentee has the right to license or sell his patent on whatever terms he pleases. It has been feared that this principle, if carried too far, might become a dodge for avoiding anti-trust laws. Since the decision of the famous, but harmless, Dick case, the most hysterical exaggerations have been published on this subject. Fortunately, since then, the recent and unanimous decision of the United States Supreme Court in the "bath tub trust" case, Nov. 18, 1912, does away with all these redundant arguments and settles, beyond doubt, the principle that, patent or no patent, unlawful combinations in restraint of trade, can be stopped by the Sherman Law.

A Penalty on Patented Articles.

The Oldfield Bill, in its eagerness to avoid any hesitation on this subject, goes one step further, and unfortunately one step too far. It puts so many restrictions on the sale of a patent article, or on a patent license, that it may become a positive disadvantage to transact business by means of patents.

Examined in last analysis, it threatens a business based on patented processes or patented articles, with penalties which unpatented articles thus far are not subjected to. It takes the proposed patent law as a pretext for saddling a patented article with restrictions which have not heretofore been formulated for non-patented goods.

This unexpected paradox, promoted by the Oldfield Bill, is distinctly in opposition of the rights of intellectual property conveyed by the words and the spirit of the Constitution, and if the Oldfield Bill becomes an effective law, it will be the saddest blow ever given to our patent system. It will do comparatively little harm to large business interests, because for them, there are many ways of circumventing its provisions; on the other hand, it will cause great discouragement to smaller enterprises who, until now, have held the hope of matching inventive genius and initiative against the money power of big organizations. Make a large corporation respect the patents of a small concern, or of an individual, and you reduce at once any advantage of size or money power, and at the same time, you encourage the most beneficial form of competition, competition based on improvements. But to introduce curtailing restrictions for the licensing or selling of patented articles or patented processes to which non-patented articles are not subjected to, means simply obliterating the value of patents while needlessly

increasing still further the opportunities of endless and ruinous litigation and chicanery.

Suppression or Non-Use of Patents.

Another unfortunate miscarriage of purpose in the Oldfield Bill is its provision against so-called wilful "suppression" or "non use" of patents. It does not take into consideration that in numerous instances, a patentee or an assignee possesses a series of so-called alternative patents, which can be used to bring about identical or similar technical results by modified means. Among such alternate patents, the best or the most suitable is used, absolutely irrespective of any other reason or intention to suppress their use. Yet without the exclusive possession of every one of these patents, the invention would not sufficiently protect against competitors, and the field would be so much reduced as not to make it worth while to put one's best energies to the development of the invention.

In most cases, it would become a material impossibility for a small concern to maintain the exclusive ownership of its patents, if it had to go to the enormous expense of working simultaneously all its "alternate" patents; by omitting this expensive technicality, it would be exposed to the risk of being compelled by its competitors to grant a compulsory license; this would practically annihilate the advantage of exclusive ownership as expressed by the Constitution.

There again large concerns would be at an overwhelming advantage because they can at an expense relatively small for them, equip the necessary appliances for remaining within the technical provisions of the law. In the meantime, they could easily harass their financially weaker competitors in exacting from them compulsory licenses which would break up the only prospects of successful competition which the smaller concern might have possessed, until then, in its patents.

In other words, the Oldfield Bill is aiming at the petty side of the situation, and in doing so, has unwittingly picked out a vital spot of our patent system. It reminds one of the man who set his barn afire in order to drive out a hornet's nest.

I have no doubt that this bill has been framed with the best intentions for the interests of the country. Unfortunately, the framers of this bill do not foresee the far-reaching and dangerous effects of its provisions.

One-Side Popular Conception of Inventions.

The average man, even the average legislator, has a rather one-sided conception of patents or inventions. Most people's idea of a patent does not go far beyond some simple mechanical device, like a patented mole-trap, a safety razor, an alarm clock, or other similar invention, more or less easy to understand after the apparently simple mechanical principles have once been explained. Then everything seems so simple and easy to them, that their limited imagination cannot conceive how even these apparently simple devices have frequently cost incredible efforts and immense amounts of money before their advantages became available to the public.

This attitude of mind develops, naturally, the belief that a patentee has a "soft snap," the result of a lucky idea, in about the same way as a lucky prospector strikes a rich gold mine, or a lucky ticket draws the grand prize in a lottery.

Precisely on this account it becomes difficult to explain to such people the rights and purposes of intellectual property; it is still more difficult to convince them that the nation is greatly benefited by liberal patent laws.

When it comes to chemical patents, the ignorance of the average public is amusing if not pathetic. Since we have heard a New York alderman in an official address of welcome to members of the International Congress of Chemistry speak as if they were druggists or pharmacists, we must no longer be astonished if the average Congressman or Senator refers to a chemical patent as a synonym to "patent medicine."

The Far-Reaching Effect of Chemical Inventions.

But even to the better prepared legislators, it is difficult to understand how some chemical inventions have brought about the most far-reaching developments, not only in other industries and arts, but on civilization itself.

For instance, it is not so obvious to them how processes for fixing the nitrogen of the air or extracting soluble potassium salts from rocks, enable us to make food supplies independent from the restricted potash mines in Germany or the nitrate deposits in Chili. Such inventions are no more nor less than means for preventing possible starvation of our race.

Do they realize that the development of the automobile, with all what it directly and indirectly implies, was entirely dependent on Goodyear's vulcanizing process of rubber?

Shall we remind them of the fact that without the invention of explosives, like dynamite, gigantic engineering enterprises, the Panama Canal, blasting of rock for the excavation of our cities, mining for ores, tunneling and grading of railroads, would be impossible?

How could we expect even the most perfected modern printing presses to distribute to every citizen, rich or poor, young or old, that knowledge and culture, which means better citizenship, better opportunities for happiness and development of our race, if it were not for the inexpensive and abundant supply of paper furnished by the cellulose processes. The Greeks, the Romans, and even the Middle Ages had their sages, their poets: yet those were the times of slavery and oppression, because knowledge was only in the reach of such a limited number that it was possible for tyrants to throttle its diffusion by sending the few advanced thinkers to the gallows or burning them alive. For the same reason, scarcity of books, the destruction of the library of Alexandria, was a calamity for the intellectual development of mankind. Our abundant supply of cellulose makes a repetition of such conditions an utter impossibility.

Then, again, where would we find our supplies of steel, the main raw material for modern engineering, if the Bessemer, the Thomas-Gilchrist and others had not invented their processes? How about the marvelous synthesis of products derived from coal tar, which have literally created the most astounding series of new substances which have revolutionized therapeutics, surgery, hygiene, and are finding daily new applications in the most varied arts and in general technology?

The Increased Cost of Living in Its Relation to Invention and Patents.

At a time when all countries are confronted with that critical question of the increased cost of living, it may be interesting to point out that just those industries where invention and patents have played the smallest rôle are also those where the increase of price is most burdensome, while those commodities where patented inventions have had the fullest influence have, on the contrary, decreased in price, and in some instances, to an astonishing degree.

For instance, the price of sulphuric acid is about fifteen times less than it was in 1807 and about one-half of that of 1870. The price of soda ash is about one-sixth of what it was in 1823, and about one-half of the price in 1860. Nitric acid sells for less than one-half the price of 1861. Glycerine sells for about one-eighth of the price in 1855. Chloride of lime in 1800 sold for 30 cents a pound, in 1870 for about 2 cents per pound, to-day for about 1 cent a pound.

Any chemist knows that every one of these products is used directly and indirectly in the most ramified channels of our arts and industries, but the layman does not know that cheap soda means cheap soap, cheap paper, cheap glass, etc.; that cheap sulphuric acid means cheap fertilizers, better crops, cheaper corn, cheaper wheat, and so forth.

Let me point out that the decrease in price of these materials, even considerably greater than the bare comparison of figures indicates, if we take in consideration that the purchasing value of money has considerably decreased, while the cost of labor has enormously increased.

Nor are these examples nearly confined to chemical products. The reduction in price for articles where patents have played an important rôle is just as evident in steel products, tools, machinery, etc.

Compare these lower prices with the vastly increased cost of

rents, clothing, foodstuffs, and many agricultural products, where patents have played a less preponderant rôle. If you will carry your analysis still further you will find that in such branches of trade where patented inventions have had little or no importance, for instance, cattle raising, prices have soared highest. On the other hand, for such agricultural products where patented machinery could be used to best advantage, like wheat and corn, the increase of price has been relatively small. Then, again, garden vegetables, potatoes, etc., where the use of patented agricultural machinery is less available, show an enormous increase in price.

You may object that the price of shoes has gone up, but here again, the increase is entirely due to the greatly advanced price of hides, and were it not for the perfected shoe-making machinery, and for the better and cheaper chemical tanning methods, all due to patents, the cost of our shoes would be so high that they might again become an article of luxury, available only for the well-to-do.

The present price of clothing is high enough as it is; nevertheless, it would still be much higher but for the patented machinery for spinning and weaving, the patented chemical processes of bleaching, dyeing, mercerizing, etc.

I should not omit to mention our vastly improved and cheapened methods of transportation, of production of power and light, all developed and perfected on an interwoven system of patents: I could explain the far-reaching influence thereof on civilization, culture, on the happiness and security of life of the individual citizens; but even then I might not convince the pessimist or the scoffer who only sees the hole in a doughnut and stubbornly persists in ignoring the doughnut itself.

The Genesis of an Invention and the Incentive of a Patent Property.

The history of almost every invention which we are utilizing now, unconsciously, every day, is an epos by itself, the details of which are only known by the few pioneers who gave the best they had to give, who helped with their brains, with their money, and talent of organization; some with their very lives.

The oft repeated statement has been made: "An inventor cannot help inventing, whether you give him a reward or not." Then again, some others say: "Necessity is the mother of invention."

The most apparent fact is that the man who receives an ample income from his father, or some other privileged source, is less prompted to distinguish himself by arduous creative work on inventions than the poor but intelligent man who sees in invention a means of making himself financially free and independent, as well as giving an outlet to his inventive abilities.

Whoever has followed intimately the development of some chemical processes knows very well that whether "the inventor cannot help inventing," or whatever may be the incentive to invention, most of these important inventions could never have been carried out, or could never have been brought to the point where they became of public benefit but for the intelligent use of vast sums of money.

Too few people have a conception of the immense sacrifices, of the serious money risks, involved in the development of some patents. Many chemical inventions used now currently and open to the public at large have cost millions before they were brought into practical shape or before the public was educated to their advantages. Can any one expect that such expenses, such efforts, such risks would be undertaken unless there was the possibility of at least some chance of recouping by a temporary patent protection?

The Large German Chemical Corporations.

Let us take, for instance, those large German chemical companies, which employ hundreds of chemists and engineers, engaged exclusively in research work; to them we owe the development of many processes which have had an untold beneficial influence in many directions of the economies of our daily life, even on civilization itself.

They employ large aggregations of capital, reaching into many millions. The dividends of some of these companies may appear large to the superficial observer. Yet if you look more closely into it you will find that these very companies were founded long ago, some of them over half a century or more, that the large capital which they employ has never been "watered," that, although they have had the benefit of the devoted co-operation of an endless number of distinguished men, stars of first magnitude in their profession, the net returns on their invested capital, at the end of half a century of brilliant intellectual pioneer work, is relatively small, even if the dividends seem large. In fact, the net returns are decidedly lower than that of many American enterprises not over fifteen years old, and where progressive technical leadership was entirely lacking, but where tariff privileges and agglomeration of competing concerns into a trust insured a splendidly paying monopoly, notwithstanding the reckless financiering of their promoters.

If you will further investigate the history of those German chemical concerns which have become leaders of the industrial world by nothing but their intellectual pioneership, you will find that notwithstanding all the patents on which they have to rely the expenses involved in research work and pioneership, swallow up, to a large extent, the profits realized in some of the established branches. But with true scientific spirit their farsighted directors were willing to sacrifice a very considerable part of their earnings in their search for improvements and development of new ideas; they have set a magnificent example in the only competition beneficial to the public—*competition by improvement*.

One of our wealthiest retired multi-millionaire manufacturers, not so long ago, speaking about his money successes, gave the following advice: "Never be a pioneer; it does not pay. Let the other man do the pioneering, and then after he has shown what can be done, do it bigger and more quickly; but let the other man take the time and the risk to show you how to do it." To any one who advances the statement that an inventor "cannot help inventing," I desire to ask whether an inventor will do much inventing, if in order to carry on his research work, or to develop his intention, he has to spend hundreds of thousands, nay, sometimes millions of dollars, but does not possess them, and nobody is willing to take the risk to furnish the money unless there is a fair chance for his backers of obtaining some compensation by a temporary patent protection? Those who know the large sums of money which have been swallowed up by the research and development work connected with the artificial production of nitrates, with the Solvay soda process, the development of the steam turbine, electric light, electric traction, and numerous other inventions of far-reaching magnitude, will know what I mean.

Just on this account it is highly unreasonable of the Oldfield bill to try to make a distinction between the inventor in whose name the patent is drawn and the party who runs the risks in enabling the inventor to make the invention available to the public; any such legislation simply tends to discourage those who, at considerable risk, furnish the capital and the talent to develop an invention into a commercial possibility, and who thereby bring it into real public service.

The Gap between Invention and Commercial Success.

Now and then I have perceived that some of my fellow chemists, who, although highly trained, have never created anything of technical value and whose experience with matters of practical life frequently extends not beyond the confines of their lecture room or their laboratory, do not seem to grasp fully the immense distance that lies between the initial conception of an invention, or its study in the laboratory, and the overwhelming amount of careful work and money risks connected with its development on a commercial scale, until it has safely reached the point where the public can avail itself of the invention.

I wish to cite, for instance, the famous Solvay process, which gives us cheap, excellent and abundant soda, an article of

prominent importance in the wheels of our civilization. This process was known and described more than a dozen times, and had even been tried repeatedly at considerable loss on a commercial scale, many years before Solvay tied his genius to this difficult problem and developed from an unreliable laboratory reaction a process of great industrial importance, then, with a staff of able collaborators, and the employment of large amounts of cash, he overcame, by and by, the technical drawbacks which had caused the failure of all of his predecessors.

Hundreds of similar examples could be cited. Whoever has been intimately acquainted with the commercial development of some of the most successful inventions knows quite well the risks, dangers of failure, which have accompanied the herculean task of development and educational work. It is a well-established fact that the great majority of new enterprises fail, that few succeed.

The Educational Effect of Inventions.

The educational effect due to the introduction of patented inventions is of immense benefit to the public, although this fact is not very apparent to most people. In many instances the owner of a patent frequently has to go to extreme sacrifices before he succeeds in convincing the public of the merits of his invention; in fact, the public stubbornly refuses to benefit by an improvement to which it has not been fully educated.

The practical value of cash registers only became obvious after a most thorough and expensive educational campaign.

The metric system is just as useful as the cash register; it was invented long ago and systematized in all its details during the first French Republic. Nevertheless, to-day, there are still two large commercial countries, the United States and England, which have not yet been educated to its merits; if the metric system had been patented, like the "cash register," somebody, during the seventeen years of the patent monopoly, would have undertaken the money risk and arduous task of thoroughly explaining the advantages of the metric system to our conservative citizens, and we would have ceased long ago to submit to the burden of waste of time and money caused by our antiquated, cumbersome system of weights and measures.

It has been stated, with much reason, that the best way to postpone the benefits of an invention is to allow public use of a patent, because then nobody takes the risk of starting an educational campaign or of developing the invention, which, after all, means pulling the chestnuts out of the fire for the benefit of others.

Entirely new industrial enterprises are not easily started on inventions which are not patented, unless some other method is available for insuring some kind of a monopoly: for instance, by maintaining secrecy or by acquiring special skill, or by controlling the raw material, or by tying the market, or, in other instances, where the initial outlay for a plant requires a capital so large as to exclude others.

Moreover, if you scrutinize those industries where secrecy of methods, instead of published patents, is the prevailing tendency, you will find that the secret-process industries are precisely those which have least progress to record and where high prices rule.

Whoever desires to get posted on the modern literature pertaining to any industrial chemical processes will find that available text-books are many years behind in information as far as novelty and accuracy are concerned; for this reason alone, it is absolutely indispensable to get acquainted with all recent patent literature.

Were it not for the compensation expected from patent rights most of this information would be carefully kept secret, or, if it were divulged at all, this would mostly occur by accident. Every newly published patent sets to work the thinking cells of numerous inventors, who are not slow to suggest further possible improvements. Every patent of some importance is rapidly followed by a succession of other patents conceived by other inventors who were inspired by their predecessors and so the work of progress goes on unceasingly and at a quickened pace.

In the age of the alchemists there were no patents; inventions and discoveries were jealously guarded and buried with their originators, and the world and its inhabitants remained very much what they were, with most rights and comforts in the possession of those in power, and very little chance of improvement for the non-privileged classes.

What Can Be Done?

The public should be educated in these truisms. Unfortunately, the education of the public has been directed in the opposite way since patent infringers have utilized the daily press after the late decision of the Supreme Court in the Dick case to start a campaign for urging our well-meaning but ill-prepared legislators towards patent reform, which will give still broader scope to our modern buccaneers. This reminds of the man who, after stealing a stranger's pocketbook, kept on shouting "stop thief," so as to distract the attention from himself.

Two ways are open for our legislators:

One way is to try "to hit the trusts" by mutilating the best there is in our patent system, which has been such a potent factor in the development of our country; to chill the best incentive for private enterprise; to stunt that kind of competition most beneficial to the public, competition by improvement, incomparably better in this respect for stimulating industry, science and progress, than protective tariff privileges which, in many instances, have worked in the opposite direction.

The other way is not to put dangerous restrictions on the patent rights defined by our Constitution. If there has been any fear that such patent rights might be abused for evading the provisions of the anti-trust laws these apprehensions have vanished by the clear, unequivocal decision of the Supreme Court in the bath tub case.

But there is urgent need of reform in our patent system by simplifying procedure in the Patent Office as well as in the courts by insuring better, quicker, and less expensive means for adjudicating the title and validity of patents. Only such a reform will bring about that big or small, poor or rich alike may be stimulated by the advantages of our patent system, instead of making a patent an expensive but powerful instrument available only to the wealthy.

Whatever simplifies and lessens the cost of the administration of our otherwise excellent fundamental patent law gives the enterprising man with small means a better chance of competition by inventive progress and merit against ponderous aggregations of capital. By such reform, which insures such healthy competition, the nation is sure to be benefited.

In all above considerations my remarks were principally inspired from the standpoint of chemical patents, not alone because this very important class of patents is least understood by the average public and the legislator, but because chemical process patents are also those which are most difficult to protect them from infringers.

The University of Pittsburgh is undertaking an investigation of the smoke problem, and has issued *Bulletin No. 1*, containing an outline of the work.

The indexing of mining literature receives attention in the *Quarterly* of the Colorado School of Mines, Golden, Colo. Carl A. Allen, assistant professor of mining, has completed an extension of the Dewey Decimal System of Classification as applied to mining.

Semi-Automatic Furnaces.—The W. S. Rockwell Company, of 50 Church Street, New York, has issued illustrated catalog 16 on the Rockwell semi-automatic furnaces for annealing, hardening, tempering, heat treating and miscellaneous heating operation. This type of furnace has been developed to provide a means of uniformly heating large quantities of material of irregular shape and sizes and which cannot be well handled by an automatic furnace of the rotary or conveyor type nor is it economically heated by the ordinary stationary furnace. The "semi-automatic" furnace is of the stationary form, but the material to be heated is conveyed through it mechanically in a continuous or nearly continuous manner.

Measurement of Temperature by Electrical Means.*

By Chas. Burton Thwing, Ph.D.

The measurement of temperature, as compared with the measurement of other physical quantities, is beset with many difficulties, since temperature is a quantity which cannot be directly measured, as are length and weight, but must be inferred from the change in various physical properties, either of the body itself whose temperature is to be measured, or of other bodies in contact with it.

The changes, for example, in volume of the standard body, or thermometer, as it is called, for each temperature in its range, having been previously determined, the body and the thermometer are brought in contact and allowed to remain in contact until they have reached thermal equilibrium, when the temperature read from the thermometer is assumed to be the temperature of the body. It often happens, however, that both the body and the thermometer are immersed in a medium whose temperature, like the air of a gas furnace, is changing rapidly. In this case, if the body is large and the thermometer small, the thermometer will respond more quickly than the body and give false indications of the temperature of the body.

One of the earliest methods of comparing temperatures is still, for many purposes, one of the most accurate and convenient, i.e., the change in volume of a liquid, like mercury or alcohol, contained in a glass bulb, terminating in a slender tube. There are certain circumstances, however, under which the mercurial thermometer fails to meet all the requirements; first of all it is not suitable for measuring high temperatures; secondly, it is often inconvenient or impossible to so place the thermometer that it is in contact with the body whose temperature is to be measured, and at the same time accessible to be read; thirdly, it is not at all well adapted for making a graphic record.

The various types of electrical instruments for measuring temperature have proved very adaptable over a wide range of temperature, in the most inaccessible situations, and are admirably adapted to produce graphic records at any distance. The increasing use of such instruments, both in research work and the numerous lines of industrial operations, makes it desirable that engineers should be informed as to the general principles on which electrical temperature measurements are based, and that they should have some notions as to the availability of the different types of such instruments best suited to various conditions.

Leaving out of the question certain optical pyrometers, in which an electrically heated body is used for color comparison, there are three different types of electrical instruments used for measuring temperatures: (A) the simple thermocouple, connected to an indicating or recording galvanometer; (B) the radiation type, consisting of a device for collecting the rays at the junction of a sensitive thermocouple, which is connected to an indicating or recording galvanometer; (C) a resistance which varies with the temperature, used in conjunction with a suitable indicating or recording potentiometer or millivoltmeter, for measuring the variations in resistance, which may be either in the nature of a Wheatstone Bridge or a direct reading ohmmeter.

I shall proceed to describe the three types of instruments in the order named, and then discuss briefly the suitability of the various types for making measurements under some of the varying conditions which occur in practice.

Simple Thermocouple.

In the Type A instruments, the thermocouple is composed of two wires or rods of unlike metal, securely joined together, by twisting, riveting or welding, at one end, this junction being known as the "hot junction," and connected at the free ends to copper leads which convey the current to the millivoltmeter, the scale of which is ordinarily graduated in degrees of temperature, or, in other words, is "direct reading." This point of connection between the elements of the thermocouple and the cop-

*A discussion before the Philadelphia Section of the American Institute of Electrical Engineers.

per leads is commonly referred to as the "cold ends" of the thermocouple.

Only a limited number of metals is available for use as thermocouples, since the thermoelectric difference of potential is small for some of them. The choice narrows down in practice to copper, iron, nickel, and platinum, and the alloys of nickel and platinum. The first thermocouple to be employed was that known by the name of LeChatelier, consisting of platinum for the negative against platinum-10 per cent iridium for the positive element. The platinum-iridium alloy is at present almost wholly displaced by the use of platinum-10 per cent rhodium, which, although it gives a somewhat lower e.m.f., is more constant under continuous use for high temperatures.

The platinum thermocouple may be used for temperatures as high as 1600 deg. C. (3000 deg. Fahr.) for short intervals, but under long exposure, even when protected in the best porcelain tubes, the evaporation of the platinum at the "hot junction" and its absorption by the platinum-rhodium gradually reduces the e.m.f. of the thermocouple. Moreover, the failure of the porcelain tube to prevent the access of various contaminating gases to the platinum thermocouple permits in time such alterations in the couple as to necessitate frequent recalibration. The high cost of platinum makes it necessary to use very small wires, a circumstance which not only aggravates the faults above mentioned but introduces errors of considerable magnitude, due to the varying resistance of the thermocouple when immersed to varying depths in the heat.

These considerations led to the search for base metal thermocouples, some of which have proved to be, in many respects, superior to the platinum thermocouples for temperatures up to 1000 deg. C., at least. For the positive element of such thermocouples, which are to be used for temperatures as high as 1100 deg. C., an alloy of nickel with 6 to 10 per cent chromium is one of the most suitable. For the negative element, a nickel alloy with a small percentage of various other elements is used. This thermocouple has an e.m.f. about $3\frac{1}{2}$ times as great as the platinum thermocouple at 1100 deg. C. (2012 deg. Fahr.). These materials are, moreover, so much cheaper than platinum that the thermocouple can be made of larger size, thus eliminating very largely both the error due to the surface contamination of the elements and the error due to the varying resistance with varying depths of immersion. The latter error is still further reduced by the fact already mentioned that the e.m.f. of the couple is higher, thus permitting higher resistance in the galvanometer.

Where the maximum temperature does not exceed 900 deg. C. (1652 deg. F.), a nickel alloy consisting of about 41 per cent nickel with 59 per cent copper, used as the negative element against the above mentioned chrome-nickel alloy, or either copper or iron as the positive element, gives an exceedingly constant e.m.f. as well as a very high one; the e.m.f. being about six times that of platinum, at 900 deg. C., and exceedingly constant even after long use. This alloy, which is known as constantan, has the added advantage over platinum of having a very small co-efficient of resistance even for high temperatures, so that the error due to the varying depth of immersion of the thermocouple becomes practically negligible. The high e.m.f. of this couple makes it especially suitable for low temperature measurements; indeed, for all temperatures below 900 deg. C., it is preferable to platinum for thermocouples, even though here were not the difference in cost.

Limitations of Thermocouples.

The extreme simplicity of the thermocouple makes it one of the first means to be considered when electrical temperature measurements are to be made. It has, however, two faults; first, the quantity of energy available is small, so that the millivoltmeter employed must be much more sensitive than the electrical instruments employed for most commercial purposes. The e.m.f. of the platinum thermocouple is but 18 millivolts at 1600 deg. C. (2912 deg. F.) and that of the nickel thermocouples but 60 millivolts at their respective highest ranges. With the best modern designs of millivoltmeters there

is, however, no difficulty in making accurate instruments to measure the small amount of energy available, except where the range of temperature covered is small. Under these circumstances, it is sometimes best to use a number of thermocouples in series, though it is often preferable to adopt the resistance type of instrument, especially if the temperature is low.

The second objection is a more serious one, where measurements are required to be of extreme accuracy, and most serious when the range of temperature is small. This is known as the "cold-end error." It must be borne in mind that the e.m.f. which is measured by the galvanometer is a quantity which varies with the difference in temperature between the hot junction and the cold end of the thermocouple, or where the copper wires leading to the galvanometer are joined to the elements

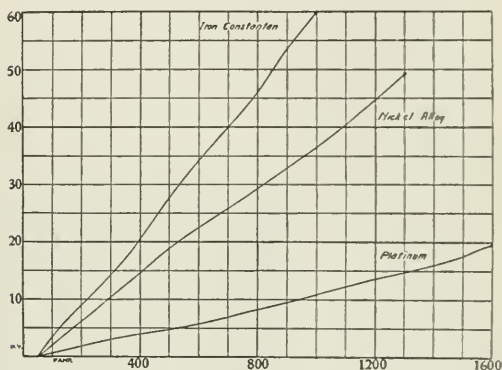


FIG. 1.—E. M. F.—TEMPERATURE CURVES FOR THREE TYPICAL THERMOCOUPLES.

of the thermocouple. If the instrument is calibrated for a "cold end" temperature of, say, 25 deg. C. (80 deg. Fahr.), it will read too low if the cold ends are at 30 deg. C., and too high if the cold ends are at 20 deg. C. It is quite commonly possible, where high temperatures are to be measured, to place the cold end at a point near the floor, or underground, or in contact with steam or water pipes, so that it may be maintained at a temperature which will not vary far from a standard fixed temperature. Of course, it is possible, when special refinements of temperature measurement are required, to observe this cold end temperature and make the necessary correction for it. In



FIG. 2.—RADIATION PYROMETER.

research work, or for calibration, the cold ends are immersed in ice water.

The curves shown in Fig. 1 illustrate the temperature e.m.f. relation of the three typical thermocouples mentioned above.

Radiation Pyrometer with Thermocouple.

Type B is available for temperatures which are too high for platinum thermocouples, or for situations not easily accessible, even where the temperatures are not too high, as, for example, in a large furnace, or at the surface of materials in motion, such as masses of steel passing through rolls or under the hammer. In this instrument, use is made of the law connecting the total energy radiated by a body with its temperature. This law is known as the Stefan-Boltzmann Law, and is expressed in the equation $E = K(T^4 - T_c^4)$ where E is the total energy radiated by a body of absolute temperature T , and K is a constant depending on the units used. This law has been found to hold not only for the highest temperatures measurable with gas thermometers and thermocouples, but has been found to give

results in perfect harmony with measurements made by employing optical pyrometers, in accordance with the law of Wien, as high as 2600 deg. C. (4712 deg. Fahr.).

The method consists, as stated, in collecting all the radiations, both luminous and non-luminous, from a limited area of the body whose temperature is to be measured and concentrating them upon the blackened junction of a sensitive thermocouple. In the form employed by Féry, the rays are collected by means of a concave mirror, and are focused upon the junction of the thermocouple by means of an ingenious optical device, contained in an eye-piece attached to the telescope.

In the simpler form devised by the writer, the radiations are concentrated by means of a conical aluminum mirror. Since the radiations are transmitted down the conical mirror by means of multiple reflections, as shown by Fig. 2, the mirror is equivalent to a fixed focus mirror, for the rays entering the cone through the diaphragm are transmitted to the outlet, and strike the thermocouple, regardless of whether or not they are parallel on entering the cone. This makes the instrument exceedingly convenient to use in foundries and rolling mills, and other places where the quick motion of the men would be interrupted by the use of an instrument which required mounting in one position for any length of time. The quickness of action of this instrument is further increased by the extremely small dimensions of the thermocouple, the heat capacity of which is so small that equilibrium is reached in five seconds after pointing the receiving tube of the instrument at the hot body.

This instrument has found application, both in the indicating and recording forms, for use in steel and rolling mills, steam boilers, smelting furnaces, and in the large rotary form of kiln used in cement burning and the nodulizing of iron ore. The portable form is used for taking the temperature of molten steel, both before tapping from the open-hearth furnace, as well as after tapping, and while pouring, and for taking the temperature of tools made of high-speed steel, in hardening furnaces, where it is desirable to indicate not so much the temperature of the furnace as the temperature to which the work itself has been heated. This type of instrument is not only capable of a high degree of accuracy in measurement but, since it is not exposed to any high temperatures, it deteriorates much less with use than any other form of electrical pyrometer.

Electric Resistance Pyrometer.

Type C, or the Resistance form of electrical pyrometer, has been employed for all ranges of temperature from that of liquid air to temperatures as high as 1100 deg. C. (2012 deg. Fahr.). For this higher temperature, the problem of protecting the resistance coil from mechanical damage, as well as from the contamination of the platinum by the presence of any foreign gases, becomes a very serious one.

The slightest admixture of any foreign material with platinum alters its resistance very appreciably, so that it becomes necessary to protect the coil both with a porcelain tube and an additional covering of nickel. The nickel protection, of course, is only durable at such temperatures as are the nickel thermocouples, and the whole bulb is much more expensive to install and maintain than the thermocouple.

For temperatures below 200 deg. C. (392 deg. Fahr.), resistance wire of pure nickel is employed. This has the advantage of low first cost and constancy in its indications over these low ranges of temperature. The coil is enclosed in a tube of varying shape and material, suited to the conditions under which it is to be used, and is capable of taking measurements comparable in refinement with those made by the best mechanical thermometers. The same bulb may be employed for measuring all temperatures from the transformation point of nickel, about 250 deg. C., to the lowest temperatures obtainable with liquefied gases.

When the bulb is used as one arm of a Wheatstone bridge, the galvanometer employed will only give accurate indications when used as a null instrument, unless provision is made for supplying a constant emf, in which case it may be used as a deflection instrument. The form of indicator most commonly used

with the null method has a dial graduated to degrees under a pointer which moves a contact over an adjustable resistance.

Selection of Type of Pyrometer.

In the selection of the suitable type and form of instrument for measuring temperature under any particular conditions, the temperature range will be one of the first considerations. For temperatures with a maximum range above 1500 deg. C. (2732 deg. Fahr.), the radiation type is the only one of the three which is available. The lower limit for this type of instrument is ordinarily about 500 deg. C. (932 deg. Fahr.).

While the upper limit is 1500 deg. C. choice may be had between the radiation and the platinum thermocouple.

Where the upper range is 1200 deg. C. (2192 deg. Fahr.), the chrome-nickel thermocouple, the platinum thermocouple, and the radiation type may all be used.

For temperatures up to 1000 deg. C. (1832 deg. Fahr.), the iron-constantan thermocouple would come first in consideration, with the platinum resistance type also available.

For temperatures below 200 deg. C. (392 deg. Fahr.), the iron-constantan thermocouple and the nickel resistance could be employed, with the choice probably in favor of the resistance type.

The form and size of the thermocouple or resistance bulb would also be determined by circumstances. Where large heat areas are under consideration, with slow changes of temperature, the thermocouple may be of large cross-section, and thoroughly protected with iron or refractory tubes.

Where the bodies to be measured are small, and the temperature fluctuations are very rapid, the thermocouple must be made smaller. For example, if we are determining the transformation point of a number of specimens of steel, it is desirable that the specimens be small, with a small opening into which we may insert a very small thermocouple.

The same considerations which make the thermocouple small, would determine the choice of the thermocouple as against the resistance bulb, since the former can be made almost infinitesimally small, and will correspond with corresponding quickness to the temperature of the enclosing body.

Where thermocouples are used, the greatest consideration must be given to the matter of cold-end variation, and in the installation the most complete precaution must be taken in the matter of wiring, to make sure that all joints are permanently made.

In the case of resistance thermometers, the failure to get good results is often due to the failure to protect the lead wires from moisture or from abrasion. In fine, we must recognize that in general when making temperature measurements, we are handling a complex problem, and are dealing with electrical temperature-measuring devices which are sensitive instruments, requiring the application of judgment to obtain the best results.*

Philadelphia, Pa.

The South African Mining Journal recently celebrated its twenty-first anniversary by issuing a souvenir number. The issue contains historical data on Rand mining and milling, as well as a general survey of present conditions, and is in keeping with the importance of the special field covered.

The Yarnall-Waring Company, formerly at 1100 Locust street, Philadelphia, announces their removal to Chestnut Hill, Philadelphia, having assumed management of the factory and sales of the Nelson Valve Company. The identities of both companies will be maintained as before.

Electrical Show at University of Illinois.—An electrical show is to be held at the University of Illinois at Urbana, Illinois, on February 6, 7, and 8, 1913, under the auspices of the Electrical Engineering Society. This show is a triennial event and a university affair, conducted wholly by the undergraduate students, of interest to the alumni as well as to electric power users and manufacturers who will be allowed to exhibit.

*In connection with this subject the article by Dr. Northrup on the thermal e.m.f. of tungsten and molybdenum on page 45 of this issue will be found interesting.—EDITOR.

A New Type of Blast Furnace Construction

A paper read at the Cleveland meeting of the American Institute of Mining Engineers by J. E. Johnson, Jr.

The general construction of blast furnaces has undergone no radical change in more than a generation. When the old style of masonry construction was replaced by the steel shell, the masonry piers were simultaneously replaced by columns never less than six and frequently twelve or sixteen in number, set under the mantle ring. These columns were at first unvaryingly made of cast iron, but in more recent years have frequently been made of structural steel.

The furnace itself has recently been the subject of radical

changes; in some cases the thin-lined construction has been adopted for the whole furnace, and in other cases has been adopted for a zone immediately above the bosh, raising the mantle several feet to make this possible, but still the style of construction with columns set immediately under the mantle has been universally followed. In spite of the apparent permanence of this type of construction it is open to grave disadvantages from the operating point of view, which may be briefly outlined as follows:

The bustle pipe is necessarily outside the columns as the slope of the bosh and size of the crucible are such as to leave but little room inside the columns for the necessary water piping, etc., consequently the penstocks have to pass between the columns. Moreover, the hearth jacket requires a diameter but little smaller than the circle of the inside of the columns, and as a result the cooling-water ditch for protection against breakouts is exceedingly limited in width. As a consequence of these conditions, when any work requires to be done around the furnace such as changing a tuyere or working on the cooling-water pipes, the amount of room left by the columns, penstocks and water pipes is more conspicuous by its absence than by its presence. Moreover, if a break-out occurs, the ditch is filled with iron, perhaps half way around the furnace, or more. This locks itself in between the columns and the hearth jacket in a way that not infrequently makes impossible its removal while in blast. I knew one furnaceman who declared that the proper thing to do with the ditch was to run

the first cast of iron into it and let it stay there so as to save future bother.

Another result that not infrequently has followed break-outs is the burning and cracking of the columns by the molten iron and cinder surrounding them.

Still another result of this construction is less immediately disastrous, but leads eventually to a very serious condition. The space between the base of the columns and the hearth jacket being so small the whole structure is necessarily set

on one foundation and the continual expansion, which all masonry structures undergo from continued heating, gradually pushes the column bases out. In some cases an effort has been made to resist this tendency by making a continuous sole plate, or ring, on which the bases of the columns rested, but I have never known a case in which these were not broken and some plants have abandoned their use altogether, using simply an individual base under each column on the ground that as they will break apart anyway it makes a neater job to make them separate in the first place.

I think all furnacemen have experienced these

conditions and will agree to the general correctness of these objections to this construction.

It had seemed to me for several years that it would be possible to avoid these evils by building a frame-work of structural material strong enough to carry the weight of the whole furnace and supporting this on columns set at the corners of this frame-work, which would throw them so far back from the furnace proper that they would be safe from all danger from break-outs or other accidents, and at the same time would allow room enough around the furnace for necessary access to all parts with great safety, ease and speed in all necessary work; and that if the ditch became filled with iron, this would be unable to lock itself around the columns, and, being free on one side could readily be removed.

When our investigation concerning the quality of charcoal iron had indicated the desirability of reconstructing the fur-

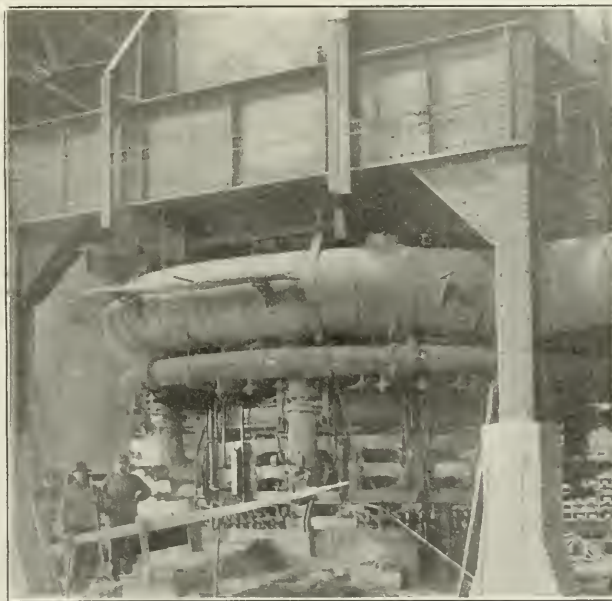


FIG. 1.—NEW TYPE OF BLAST-FURNACE CONSTRUCTION.

nace to obtain the lines needed, we laid out various plans for doing the work and this construction appeared to be so practicable from an engineering standpoint, and so desirable from an operating standpoint, that it was finally adopted.

We first laid it out with the idea of using a triangular frame of girders with a column at each angle, and in order to get the girders above the bustle pipe we considered using the thin-lined construction for several feet above the bosh angle and letting the mantle at the top of this thin zone rest directly on the girders. This is shown in Fig. 2.

But on further consideration there were two features of this design which we did not like. We desired to use the steel bosh jacket construction, cooled by external sprays which

the crucible upward from the hearth, and consequently space must be allowed in which this expansion can take place without injury to the structure, without shearing off any water pipes, and in such a position that the crack formed by the expansion will be sealed by cinder as fast as formed. This thin skel on the bosh jacket is obviously unsuited to support any brick-work above it, and, while the lining is self-renewing on the sloping surface of the bosh jacket, we were fearful that this would not be the case on the vertical or overhanging section above the bosh. We, therefore, thought it best not to adopt the thin-lined zone above the bosh.

In regard to the three-column construction, while it sets the columns back further from the furnace than does the four-column construction, and while three columns are theoretically enough to support any structure, still in the event of a serious accident around the bottom of the furnace, it was conceivable, even though, I think, highly improbable, that one column might be sufficiently affected by heat to yield, and with the three-column construction this would throw the whole furnace to the ground.

With the four-column construction on the other hand, if anchor bolts to hold the structure down to the foundation were used, one column could fail completely and the two adjacent to it would carry the structure with the assistance of the counter-weighting action of the column opposite to the one that failed. This, of course, would only be true if the columns were made so strong that any two would carry the structure.

Accordingly, the four-column construction was adopted and the columns designed to have an ample factor of safety with any two of them supporting the entire weight.

In order to raise the girders above the bustle pipe and at the same time avoid the high mantle possible only with the thin-lined zone above the top of the bosh, it was necessary to support the shell on the girders by brackets. In order to reduce the local stresses on the bottom of the shell, due to these brackets, as far as possible, eight of them were used. This also had the great advantage that by spacing the brackets so that the centers of the girders came half way between them the load on the girders was applied relatively close to the columns and the bending movement enormously reduced. At the same time the distance from the shell to the center of bearing on the base of the brackets was increased but little over what it would have been had they come on the center of the columns.

It is obvious that the mantle plate would be entirely unsupported except at its outer edge and that it might deflect at the inner edge, owing to the weight of the brick-work, if some provision were not made to prevent this.

The extent to which this action could occur in view of the brick being so firmly supported at the outside by the heavy angle riveted to the shell, is problematical, but it was not desired to take any chances and a cantilever bracket was therefore designed to support the mantle plate from below to within a short distance of its inner edge.

The angles which are employed to rivet the bracket to the shell are continued down below the mantle and support a heavy vertical plate, the inner end of which projects under the mantle plate, while the outer end is prevented from rising under the weight so applied by heavy angles fastened to it, which pass up outside the girder and take hold of the outer end of the shell bracket. It is obvious that the angles next the shell act in tension and those outside the girder in compression, making a very rigid support for the overhanging end of the cantilever under the mantle plate. In order to prevent any possibility of deflection of the mantle plate between these brackets, heavy channels were riveted to the inner ends of the cantilevers in such a position that the center of the channel comes as close as desirable to the edge of the mantle plate. This forms an octagonal ring of heavy channels which also reinforce powerfully the inner ends of the cantilevers and make it impossible for them to fail by buckling.

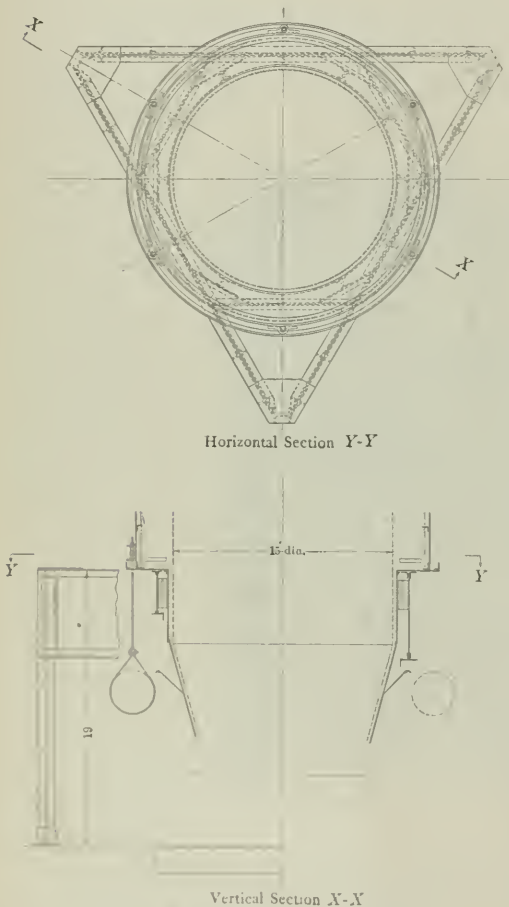


FIG. 2—HORIZONTAL AND VERTICAL SECTION.

has proven so durable and satisfactory at many plants in the South and at some in the North, and which avoids entirely the stepped bosh wall, almost as angular and rough as a flight of steps, into which the cooling plate construction soon wears.

When the bosh jacket construction is used it is necessary to realize that no matter how much brick may be laid on the jacket at the beginning of the blast, it will all soon be gone, and its place taken by a combination of carbon, slag, etc., held on the slope of the jacket by the water cooling. When this construction is adopted provision must be made for the fact demonstrated by experience that the bosh will expand downward from the mantle on which it must be supported and

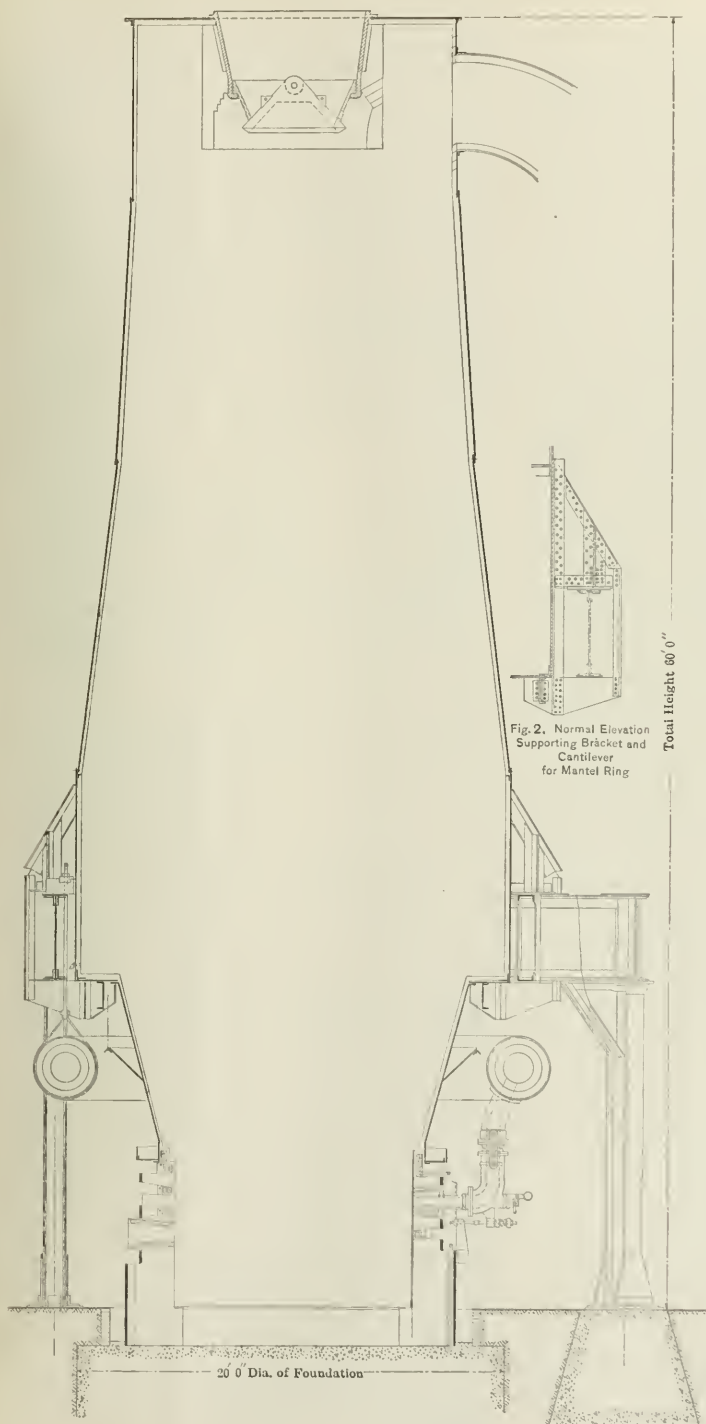


FIG. 3—BLAST FURNACE DESIGN.

The general construction is shown by Fig. 3. The section on the right hand side is taken through the center line of column and a tuyere and penstock are shown as if they lay in the same plane. As a matter of fact, they do not; the center lines of those tuyeres which come nearest to the columns pass a foot or two to the side of the columns and allow ample room for rodding the tuyeres. This section was made in this way to show the minimum of room around the furnace at any point. The left-hand section is on a plane at right angles to the girder, and while the column in that view looks comparatively close to the furnace, it is, of course, in reality at the distance shown on the right-hand side.

It will be seen that four independent piers of concrete were provided and that these set back at a great distance from the foundation of the hearth of the furnace proper. These piers are at such a distance from the foundation of the hearth and so entirely separated therefrom that it seems impossible there should ever be any transmission of the expansion of the hearth through the soft dirt between the two, which could force the piers out of position. Therefore, the columns should remain as built indefinitely.

It will be noticed that brackets were introduced between the top of the column and the bottom of the girder on each corner to prevent any possibility of the columns failing by leaning all in the same direction circumferentially. Across the corners of the main girders channels were riveted at top and bottom to form diagonal braces. These were latticed together vertically for greater strength, as shown in Fig. 4, which shows the structural work without the shell. The girders were so located that the inner edge of their flanges was 4 inches from the shell at the closest point, and the corner braces were put 7 inches from the shell at the closest point, thus leaving plenty of access to the shell for anything that may be necessary, even behind the girders.

The brackets are not riveted to the girders either at top or bottom. This is for the reason that if the bottom ring of the shell should give way in any manner and carry the brackets out, then if the latter were riveted to the girder this would put a bend in the top or compression flanges of the girders and might induce their failure. The girders and brackets not being riveted together, opportunity is provided for the bracket to slide on the girder and leave the latter undeformed under any possible condition. In addition the bottom

ring of the shell was made very strong to provide against this contingency.

The inward thrust at the top of the brackets comes on the seam uniting the bottom and second shell plates and throws this into compression. The proportions of the bracket are such that the inward reaction from each one is relatively slight and the probabilities are that the sheets alone would be able to resist it without deformation, but in order to leave no doubt whatever on this score, a heavy 6-inch angle was run entirely around inside the shell just at the bottom edge of the second sheet in such a position as to receive the full thrust from the brackets.

This being the first time this construction had ever been used, it was not considered desirable to leave undone anything that could contribute to additional safety or provide against possibility of failure in unlikely ways. The probabilities are that some of these extra provisions for strength could be left off without detriment, but it was not considered desirable to do so in this initial installation.

In regard to the strength of the girders, they were designed to be safe with a low unit-stress, assuming that the weight of the shell, top rigging and top house; lining, back lining, bustle pipe and the entire charge from the hearth to to the stock line were supported on them. This last condition is most unlikely, but it is conceivable that the furnace might become so badly scaffolded just above the tuyeres that the weight of the whole charge would come onto the mantle and so onto the structural work. We, therefore, provided against this contingency in the design.

The general appearance of the furnace in operation is well shown by the photograph, Fig. 1. Two men stood shoulder to shoulder as nearly in line as they could be placed between the wicket on the penstock and the nearest column, in order to show the amount of room available. In addition to this, as above stated, this tuyere is not in line radially with this column so that the access to the tuyeres is even more unrestricted than would be indicated by the photograph. This could have been made even better than it is except that the hot-blast main was so located as to prevent placing the columns four-square with the line of the tapping hole, which would have been better in several respects, but we did not desire to disturb the hot-blast main and shifted the columns a foot or two to the left to miss it. It will be seen that the columns are bricked up to a height of 6 feet to prevent any possibility of damage to them by fire. We considered also making provision to keep them filled with waste water for several feet up, but this was not considered necessary. The brick-work surrounding them is hard fire-brick laid in cement and almost indestructible.

The accessibility of the furnace for work of any kind is remarkable and I do not think that anyone who had worked around a furnace of this design would ever desire to return to the old type of construction.

It will be noticed in Fig. 3 that the outside wall of the ditch is merely $4\frac{1}{2}$ inches of brick. We had some trouble with the cinder cooler on one occasion since the furnace was blown in and got quite a little iron into the ditch beneath it. It ran around the furnace a distance of 7 or 8 feet. This would have been sufficient to tie itself in behind one or more columns with the old construction and would have been very difficult to get out. As it was, we simply tore down the $4\frac{1}{2}$ -inch wall from the outside of the ditch, pulled the "chunk" away from the hearth-jacket, out of the ditch, in one piece and replaced the $4\frac{1}{2}$ inch wall. This accessibility is also a great advantage in working at water pipes, and, especially in following these from the cocks on the circle pipe to the cooling member that they supply.

The very short and direct connection from the bustle pipe to the tuyeres is shown both by Fig. 2 and by the photograph. In fact the bustle pipe is almost too close to the bosh wall for greatest convenience. It is obvious that there is ample room to increase its circle and provide more room if it were ever desired to do so.

Objection will probably be made to this construction on the ground of cost, but the great saving in real cost over the standard construction is one of the features which pleased us most. The first cost of the four-column construction was less than \$2,000 in excess of that of the standard construction, but of this saving about \$1,000 would have been wiped out by the cost of work on the bustle pipe had we adopted the old construction for the following reason.

In order to have gotten the furnace lines desired with the standard construction we would have had either to provide a new bustle pipe complete or cut the present one down, cut it in two in six places and insert rings to increase its diameter and reline it complete, which could scarcely have been done for a thousand dollars. The actual money cost was, therefore, a few hundred dollars greater for the four-column construction than for the standard, but we have to consider also the element of time.

Owing to the location of the foundations for the new columns, one of these was put in before the old furnace was blown out. A second one could have been had we known

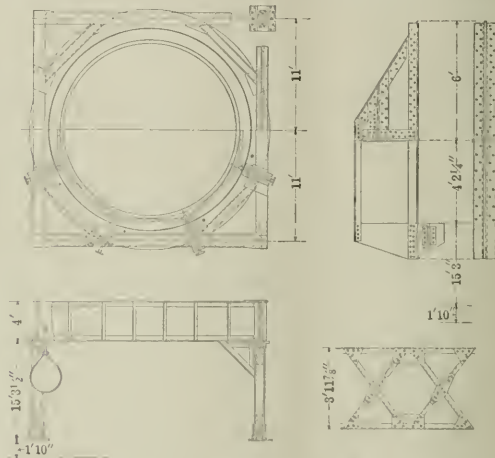


FIG. 4—SUPPORT FOR BLAST FURNACE

more accurately the location of the waste water sewer, but this and the two front ones were actually put in immediately after the furnace blew out and they were done before the work of tearing down the old furnace had progressed very far. Of course, if the standard construction had been employed the old columns would have had to be removed and the whole foundation exposed before the new foundation could be added. As it was, not only the foundations were built but the structural work for the new furnace was erected by the time the bosh and the upper part of the lining of the old furnace had been torn out. We estimate a saving of from two to three weeks in time on this account alone.

When the steel bosh jacket is used with the ordinary type of furnace it has necessarily to be fitted up and riveted together in several sections inside the columns, and, in fact, it is commonly attached to the edge of the mantle ring as it is put together with the necessity of a great deal of fitting of the vertical seams. In the present instance the bosh jacket was built entirely independent of the furnace and when the construction had advanced far enough to receive it, it was simply shoved in through the columns, hoisted up to place and riveted fast, the entire job taking only about two days as against two or three weeks that would have been required had it been built up in place in the customary way.*

We consider that we made easily a saving of five weeks on these two accounts alone. The loss of profits, overhead

*A paper read at the Cleveland meeting of the American Institute of Mining Engineers.

charges, etc., at such a plant can not be estimated at less than \$500 a day, and this saving of thirty-five days, therefore, represents a saving of \$17,500, or deducting \$1,000 for the excess cost over the standard construction, a net saving by the use of this construction of \$16,500.

It is customary to regard charcoal furnaces as insignificant in size, but this furnace is 60 feet high and of a size that would make from 200 to 300 tons per day with moderate driving on coke iron. Furnaces have such an enormous capacity for exerting destructive energy that the tendency in the minds familiar with them is to overestimate their actual weight. It is doubtful if the weight of a large coke furnace figured on the basis above described would exceed 2,000 tons, and there is no difficulty whatever in carrying a load of this amount with a construction of this kind.

It may be well to point out that this construction lends itself particularly well to the thin-lined type of furnace because the weight of the lining in such a furnace is an insignificant fraction of that of a standard thick lining, and because some outside structure is generally necessary for supporting the platforms, spray apparatus, etc., even if not the furnace top proper. This can all be done with great advantage from the girder frame if the type of construction here described be used.

It is well to point out also that if the cooling-plate construction of the hearth be used so as to form an adequate support for a thin lining in the zone immediately above the bosh, then the high mantle, which results from that construction, may be directly supported on the girders about as shown in Fig. 1. Methods can also be devised without difficulty for supporting the lining of the thin zone with the steel bosh jacket construction by providing a narrow water-cooled false mantle at the bosh angle, but we did not consider this construction necessary for our conditions.

CONCLUSIONS.

1. A furnace designed on the plan described is not only feasible but a safe and substantial construction.

2. This construction is especially well adapted to the thin-lined, or partly thin-lined furnace.

3. Its convenience in operation is so much greater than that of the older type of furnace that there is no comparison from an operating point of view.

4. The time to be saved by the use of this construction, at least in rebuilding, is so great as to constitute a money saving of far greater extent than the slight excess cost of this construction over the standard.

Ashland, W. Va.

Annual Tables of Constants.—The price of the second volume of the Annual Tables of Constants, Chemical, Physical and Technological, published by the International Commission, will be \$6, unbound, to subscribers who subscribe before January 31st, 1913.

Roasting and Drying Furnaces.—The Ridge Roasting Furnace and Engineering Company, 62 London Wall, London, E. C., has been formed to take over the patents of Mr. H. M. Ridge for roasting, calcining and drying furnaces.

Identification of Tellurides.—In a study of the tellurides of Kalgoorlie, published in the September *Journal of the Chamber of Mines of Western Australia*, J. Allan Thomson gives the following tests of identification: The presence of tellurium may easily be detected by blow-pipe tests. When heated on charcoal in the oxidizing flame, all the gold, silver, lead and mercury tellurides quickly fuse and emit dense white fumes of characteristic odor. They coat the charcoal with white TeO_2 . When the bead is touched with the reducing flame it imparts to it a bright bluish green color. When heated in a closed tube the tellurides again form white TeO_2 . This fuses on further heating to a liquid which is yellow when hot and white when cold. Sometimes a black sublimate of metallic tellurium is obtained. A simple wet test is to heat a fragment of the mineral on a porcelain plate and allow a few drops of sulphuric acid to flow over it.

Recent Progress in Electrolytic Process for Zinc.

Dr. Victor Engelhardt, the well-known chief engineer of the electrochemical department of the Siemens & Halske Company, delivered a lecture on electrolytic zinc processes at the recent first general meeting of the Gesellschaft Deutscher Metallhütten und Bergleute. Although Dr. Engelhardt is lecturing on electric furnaces at the new Institute of Technology in Breslau, he is more particularly familiar with electrolytic practice from large-scale work. The following is an abstract from his interesting paper:

The world's production of zinc, as taken from *Mineral Industry*, amounted to 815,097 metric tons in 1910, of which probably not more than 10,000 to 15,000 or 1.2 to 2 per cent were used as a high-grade metal of special purity. Zinc of 99.98 per cent purity is not much in demand, except for laboratory use and for making special pure brasses for use in cold drawing and pressing work such as are needed for cartridges, etc. This fact has held back the development of electrolytic zinc processes, especially as three considerable technical difficulties had to be overcome, viz., the difficulty of the design of suitable apparatus, the high-power consumption per unit of metal, and the fact that the zinc is liable to be deposited in spongy form.

Since there is little commercial use for high-grade electrolytic zinc at a higher price than ordinary zinc commands, the electrolytic process must be able to compete with the present marketable product. This, the author points out, is now possible under certain conditions.

He first reviews the older work in this line by briefly characterizing the main types of different processes which have been tried or suggested, without going into details of the enormous mass of patent literature.

I. Refining.

(a) *Anodes without precious metal.*—The mere refining of commercial zinc, analogous to copper refining, is economically absurd.

(b) *Anodes with precious metals.*—The refining of the ZnAg alloy, obtained from the desilverization of lead, with up to 10 per cent Ag, did not prove cheap enough and was apparently abandoned before the technical details could have been worked out.

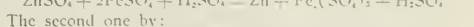
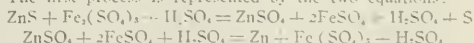
II. Direct Processes.

(a) *With soluble anodes containing zinc.*—In order to utilize the energy due to the solution of the anode a great many attempts have been made since 1880 to prepare anodes containing various zinc compounds, mixed with coke or coal powder with and without pressure, with various binding materials, etc. They all must be recorded as failures as was the case with the analogous Marchese process in which it was tried to use copper ores or intermediate copper smelter products as anodes for direct production of copper.

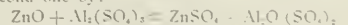
(b) *With soluble anodes free from zinc.*—The patents of Cassel and Kjellin of 1892 started a series of equally unsuccessful experiments with diaphragm cells and anodes of iron in ferrous-sulphate solution or of lead in zinc chloride. Iron will soon contaminate the cathode electrolyte, and other trials gave no better results.

(c) *With non-soluble anodes and utilization of the anode energy.*—In two processes of Siemens & Halske, in 1880 and 1890, an attempt was made to use an electrolyte which would leach the sulfide or oxide ore and which would then be passed through an electrolytic cell where the metal would be deposited on the cathode, while the solution was regenerated at the anode. Both processes are merely of historical interest to-day.

The first process is represented by the two equations:



The second one by:



The basic aluminum sulfate was, however, partly decom-

posed during the leaching process and an integral part of the electrolyte was thereby lost.

(d) *With insoluble anodes and without utilization of the anodic energy.*—Processes of this group are the only ones which have reached the commercial stage. A typical example of $ZnCl_2$ electrolysis is the Hoepfner process, a typical example of $ZnSO_4$ electrolysis the Siemens-Halske process. The plants installed by Hoepfner in Germany, England and Austria are no longer in operation.

Hoepfner started from a cheap raw material, pyrite residues with about 10 per cent Zn, mostly as ZnS , which had to be subjected to a chloridizing roasting with subsequent leaching, freezing out of the Na_2SO_4 , oxidation with hypochlorite of lime and precipitation, first by carbonate of lime, then by zinc dust. The purified solution was electrolyzed in a diaphragm cell with carbon anodes, and rotating cathodes of zinc or iron. The zinc produced by this process was of good quality, but the complexity of operation and apparatus and the necessity of collecting several by-products which were hitherto foreign to metallurgical plants and for which a market had to be found were serious objections.

The sulfate electrolyte affords simpler conditions in many respects. One point requiring the most careful attention in both cases is to avoid the formation of spongy metal. The cathodic zinc deposit cathode is not smooth enough for immediate rolling. The trade demands for alloying purposes small and easily detachable units. The electrolytic deposit is therefore molten and cast into forms of the customary shape. But when this is done, spongy metal oxidizes and is completely lost.

This problem is the most vital point of the whole process and decides on success or failure. The deposition of the purest possible zinc is therefore absolutely necessary. This is better attained in the Siemens-Halske sulfate process since the latter dispenses with diaphragms and rotating electrodes. The solution is purified in the same way as by Hoepfner, without requiring the removal of sodium sulfate. For economic reasons the accumulation of free sulfuric acid by reaction at the anode is carried on as far as the quality of the metal will allow. If it becomes too high, the danger of impurities in the cathode and of formation of spongy metal becomes imminent.

Lead Peroxide and Manganese Peroxide Anodes

The latest progress made by the Siemens-Halske Company relates to the manufacture of suitable anodes. Acheson graphite, the best anode for electrolysis of chlorides of alkalis and of zinc chloride, is corroded by sulfuric acid, platinum is too expensive and partly soluble, and the coating of PbO_2 formed on lead electrodes is not sufficiently coherent to prevent formation and dissolving of lead sulfate.

The invention of Dr. Ferchland, offered to the Siemens & Halske Company seven years ago, concerning the electrolytic formation of PbO_2 electrodes has been improved and perfected and is now in commercial use. PbO_2 is precipitated on anodes of carbon tubes (an addition of colloids permitting to produce a smooth deposit) and the carbon bored out afterward. Several of these lead peroxide tubes are held and connected together in parallel by a lead casting.

These anodes have been used in a small plant in Lipine, Silesia, Germany, where roasted upper Silesian blende is treated and smooth and thick plates of the following purity are produced

	From			From
	Silesian Blende			Refractory Ores.
Pb	0.019	0.028	0.013	0.014
Fe	0.015	0.017	0.003	0.018
Mn, As, etc.				trace
Zn (difference)	0.006	0.005	0.004	0.006

The ampere-hour efficiency was 80 per cent at 3.8 volts per cell. The specific power consumption was, therefore, not quite 4 kw-hours per kilogram of zinc. The loss in melting was 3

per cent. The ductility of the zinc effected a 10 per cent production in the cost of rolling.

Whereas in this case the cost of the anodes ran up to 70 marks per ton of metal (\$16 per 2000 lb.), the company succeeded quite recently, through the efforts of Dr. Max Huth, to reduce the anode cost to one-seventh. He tried to prepare anodes in the shape of plates and is now able to produce them from manganese peroxide, held in a carbon contact piece. The life of these anodes in commercial practice is estimated as about one year. Details of their manufacture are not revealed.

The power consumption has been reduced to 3.4 kw-hours per kilogram Zn, while chloride electrolysis requires 4.2 kw-hours for continuous operation.

Large experimental plants have been erected on this basis in England and in the United States, while the erection of a big commercial plant is projected.

The leaching of the ores which was done in Lipine in open stirring tanks is carried out in England and the United States by a special process of Dr. Isherwood, who uses an insufficient amount of sulfuric acid under pressure and thus obtains purer solutions from the start.

We omit Dr. Engelhardt's calculation of the operating costs as differing too widely under varying conditions. It will be sufficient to say that cheap ore, preferably such ores as will be less suitable for reduction in the muffle, and cheap water power are the main requirements for the new process. At marks 222 per ton of zinc in the ore, the power would have to cost 2.5 pfennigs or 0.6 cent per kw-hour in order to just cover the manufacturing cost with the market price.

The interesting discussion of the paper by Dr. Hans Goldschmidt revealed the fact, that the low loss in melting the cathodes results from dipping the cathodes into molten zinc and excluding the air to the greatest possible extent.

The complete paper is published in *Metall und Erz*, Vol. 10, p. 60-72, 1912.

British Salt Trade.

An increase in the number of salt works on the Cheshire side of the Ship Canal is expected as a consequence of the establishment of the right to pump brine; and at Lymn the existence of very large salt deposits has been ascertained. The Castner-Kellner Alkali Works, at Runcorn, draw brine from the Marbury pipe, and extensive operations are expected to be undertaken shortly on the Heatley estate, which has a long frontage on the canal. The result of borings made near Heatley Station is asserted to show the existence of a thick bed of rock salt on the estate.

British Engineering Imports and Exports.

The returns issued by the Board of Trade for the nine months ended on Sept. 30 show considerable increases in the value of both imports and exports of engineering material—imports of ships excepted—as compared with the corresponding period of last year. Imports of iron and steel, including manufactures, totaled £9,232,320, an increase of £1,031,046, and exports were £34,549,271, an increase of £2,697,070. In other metals, including manufactures, imports reached £22,230,943, an increase of £1,500,064, while exports touched £8,684,345, an increase of £659,384. Imports of electrical goods went up to £1,036,376, increasing £23,733, and exports figured at £3,198,770, a rise of £1,207,918. Imports of machinery amounted to £4,995,839, an improvement of £584,181, and exports moved to £24,056,385, a rise of £1,520,443. For new ships the imports were £25,308, a decline of £35,608, but exports rose to £4,433,042, an improvement of £393,560.

Taking the month of September by itself, imports and exports of iron and steel showed, respectively, increases of £338,493 and £1,405,816; in other metals imports fell £233,210 and exports rose £232,717; imports and exports of machinery rose £62,033 and £756,186, respectively; in electrical goods imports dropped £10,859, but exports were £277,024 better; imports of new ships were £3,719 lower, and exports improved £384,792.

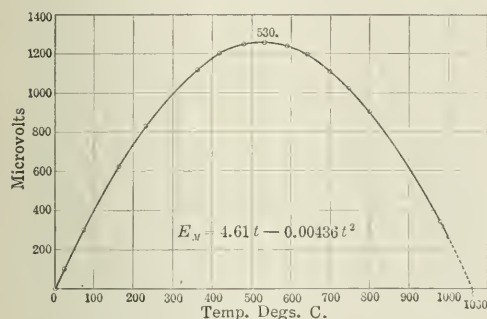
Tungsten and Molybdenum—Their Thermal Emf.

By Edwin F. Northrup, Ph.D.

The writer suggested at a recent meeting of the Philadelphia local branch of the American Institute of Electrical Engineers, that the possibilities for further developments are greater in thermocouple pyrometry than they are in electrical resistance pyrometry. He remarked that possibly tungsten and molybdenum might be used at high temperatures for a thermocouple and that although these metals oxidize at high temperatures, it might be possible to avoid oxidation by the use of some kind of protecting covering.

In reply to these statements it was remarked by a manufacturer of pyrometers that the writer's suggestion seemed to be an excellent one, with one exception, namely, that between tungsten and molybdenum there exists *no* thermal emf. The speaker said he knew because he had experimentally tested the truth of his statement.

The writer was silenced because he had not at the time tested the property in question of these two metals. He was not,



E. M. F.—TEMPERATURE CURVE OF TUNGSTEN-MOLYBDENUM THERMOCOUPLE.

however, convinced and later through the kindness of Dr. W. R. Whitney, of the Research Laboratory, of the General Electric Company, he was able to obtain a wire of tungsten and a wire of molybdenum, each about No. 20 B. & S. gauge. With the furnace at his disposal he was able to determine the curve relating emf to temperature in the range 0 deg. C. to 1000 deg. C. The curve obtained is given in the accompanying diagram.

The emf's were measured with a Leeds and Northrup potentiometer and the temperature was measured by measuring the emf developed by a platinum, platinum plus 10 per cent rhodium thermocouple. This thermocouple was accompanied by a certificate from the Bureau of Standards.

The following points may be mentioned:

The equation,

$E = 4.61t - 0.00436t^2$ microvolts fits the curve in the range from 0 to 1000 deg. C. with an error of less than 4 per cent.

When the cold junction is at 0 deg. C. the emf is a maximum at 530 deg. C. and is 0 at 1060 deg. C., which, within the limits of error of the experiment, is the melting point of gold (1062 deg. C.). The current in this range flows from tungsten to molybdenum through the hot junction. The thermal emf. is sufficiently high at 300 deg. C. or at 750 deg. C. to be measured on the potentiometer used within 0.2 of 1 per cent.

If the law of the thermal emf as given by the equation continues to hold to a temperature of 2300 deg. C. the thermal emf would be at this temperature, when the cold junction is at 0 deg. C., 10,960 microvolts or about 11 millivolts—an easily measured emf. The molybdenum would then, of course, be positive to tungsten at the hot junction.

It is very desirable that many samples of tungsten and molybdenum be tested in the above manner. If they all have a neutral point at the same temperature, namely, at 1060 deg. C., then

a thermocouple of these metals would serve in a very convenient way to fix this temperature. A couple or a resistance thermometer to be tested would be put in the same electric oven with the tungsten molybdenum couple and the temperature raised until the latter showed zero emf. The temperature would then be 1060 deg. C.

The writer does not think that there is no hope of finding a protecting case for the couple at 2000 deg. C. or more. At 100 deg. C. the oxidation is negligible and for laboratory work it could be prevented in most cases for higher temperatures by employing a non-oxidizing gas.

*Polymer Physical Laboratory,
Princeton, N. J.*

Some Present Pickling Methods.

By Oliver W. Storey.

In the hot working of iron, steel and cast iron, the unavoidable formation of a scale makes the subsequent operation of pickling necessary, especially in the various galvanizing, plating and enameling processes. This scale consists essentially of the magnetic oxide of iron, which is resistant to corrosive action.

Ordinary mill scale, as found on hot-worked materials, is present in various thicknesses. On black sheets it is present as a thin coating, while on heavy rolled shapes it is much thicker, often being over 1/32" in thickness. The coating especially when heavy is usually porous and brittle, while more flexible and continuous when thin.

The resistance of a certain quality of iron oxide to corrosive action is the basis of the protective coatings of the Bauer-Barff and Wells processes. By these methods a thin continuous coating of the magnetic oxide is produced on the surface of the iron and the article is rendered immune to corrosion as long as the continuity of the coating is not broken. Since the magnetic oxide is electronegative to iron by nearly two volts, the damaging of the surface will start a rapid electrochemical corrosion of the iron base.

For ease in removing scale by pickling a continuous coating of the magnetic oxide is undesirable since it is insoluble in the pickling acid and prevents the action on the underlying iron. It is well known that the principal function of the pickling acid is not dissolving the scale but the dissolving of the underlying iron sufficiently to loosen the scale. Scale formed under ordinary hot working conditions is porous and is usually easily removed.

If all the scale found on iron and steel were pure magnetic oxide, the problem of pickling would not be as troublesome as it is at present. Not only does the thickness of the scale vary, but its composition varies owing to impurities. Ordinary black sheets do not always present a smooth surface but often are very rough. It will be found that cinder, slag and scale contained in these rough places cause trouble in pickling. Since a much longer time is required to remove the scale from these rough spots the rest of the surface will be overpickled and the cleaned iron will present an uneven surface. Similar troubles are found in the pickling of iron pipe. Forged material is especially difficult to pickle owing to the cinder and scale that is driven into the iron.

Acids Used for Pickling.

The acids usually used for pickling are sulphuric, hydrochloric and hydrofluoric. The last is frequently used on castings to dissolve any adhering sand that may be in a position where it cannot be easily removed by tumbling or sand blasting. Sulphuric acid is the one almost universally used for pickling purposes. This is owing to its cheapness and ease of transportation in large quantities by tank cars. Sixty deg. and 66 deg. Be. acid is worth from \$6c. to 8c. a hundred f.o.b. plant in most industrial districts, this price holding for a brimstone acid containing a minimum of arsenic. The sulphuric acid must have high purity since certain impurities will cut down the pickling power. Arsenic especially is most undesirable as an exceedingly small quantity is harmful.

Prof. C. F. Burgess was among the first to show the quantitative relation between the arsenic content of sulphuric acid to its pickling power and to give an explanation of this phenomenon.¹ It had been known in general for over sixty years that the presence of arsenic in sulphuric acid was detrimental when used for pickling purposes, but no definite work had been done or had any explanation been offered.

Dr. O. P. Watts has elaborated on this work and the influence of various materials on the rate of corrosion of iron has been investigated in the Chemical Engineering Laboratories of the University of Wisconsin and some interesting results obtained.² These results show that arsenic is an effective retarder of corrosion and the same is true of tin. Lead, which is soluble to a slight extent in sulphuric acid, is also a retarder. While certain materials are more or less effective in retarding corrosion, others may cause accelerated action.

It would appear that a mixture of acids, such as sulphuric and hydrochloric, ought to cause an accelerated rate of corrosion, but the results show a high rate of retardation.

These experiments indicate that the acids used for pickling purposes must be free from those materials that retard corrosion to be able to secure a maximum pickling efficiency. A small amount of these retarding materials present in the acid will often cut the capacity of the pickling plant in two and perhaps double the acid cost. A study of the action of sulphites and of selenium on the corrosion rate would be of importance, since these are also usually present in varying quantities in commercial acid. In fact, the whole question of the influence of impurities as to their influence on efficiency of pickling is worthy of further investigation.

Where large quantities of the acid are used it is bought in car-load lots and stored in large iron containers. If the acid has to be transported and stored in glass carboys it adds materially to the cost, owing to the large amount of handling during transportation, higher freight charges owing to freight classification, breakage of carboys and the danger in handling glass containers of acid.

The layout of a pickling plant depends on several factors, these being principally the character of the material to be pickled, capacity required, and the relation of the pickling process to the rest of the plant.

The character of material is of utmost importance as the various grades of iron, steel, and cast iron corrode at widely varying rates, the rate depending upon the kind of acid and the purity of the iron.³ In general, the lower the carbon content of the iron the lower the corrosion rate in materials subjected to the same heat treatment.

Pickling of Black Steel Sheets.

Probably the two materials that are most frequently pickled are black steel sheets and wrought iron and steel pipe, including both water pipe and conduit. Steel sheets are usually covered with a thin scale that can be easily removed, while the scale usually found on pipe is more difficult to remove owing to its weight. The methods used for pickling these materials will be taken up in detail.

To operate a pickling plant successfully requires a large supply of hot and cold water. A large amount of water is needed for washing after pickling to remove all traces of acids and metallic salts.

The pickling of black sheets is usually done with the so-called pickling machines. Such machines are now on the market capable of handling the material rapidly and economically, a typical one in general use being the Mesta pickling machine.* It consists essentially of a vertical plunger working in a cylinder, with three arms 120 deg. apart at the top of the plunger which is worked by steam or compressed air. Two vats are required with this type of machine; one acid and the other water. While one crate of sheets on the first arm is pickling in the first vat, the second crate on the second arm is being

washed in water, while the third is being unloaded or re-loaded. If a third vat is necessary the machine can be built with four arms.

The vats are constructed to the dimensions required by the material to be pickled. The crates are often made of wood but more often of some acid-resisting alloy. Where wood is employed nails or bolts of copper are necessary. The sheets are rested on edge in this crate, which is usually large enough to hold from 1500 to 2000 lb. of the material.

When sulphuric acid is used the strength of the solution usually varies from 3 to 8 per cent, and it is usually kept as near boiling as possible by injecting live steam into the vat. The high temperature has a remarkable effect on the rapidity and efficiency of the action. The sheets are forced apart to secure complete pickling by violently agitating the acid solution. With suitable agitation and separation of the sheets, this process may be completed in six to 10 minutes with a 6 per cent pickle. When a weak pickle, such as a 3 per cent solution and a temperature of about 180 deg. Fahr. is used, the sheets usually require about 30 minutes for complete removal of scale.

After being pickled they are transferred to the washing tank where the acid and ferrous salts are removed as completely as possible by copious supplies of fresh water. If the wash water is hot the sheets will dry quickly when brought into the air with the formation of but a negligible quantity of iron oxide. If the sheets are not for immediate use and are to be stored or shipped they are dipped in a lime or soda solution to kill any acid that may be present and prevent the formation of rust.

This method of pickling will remove all of the scale, but owing to the short period in the acid will not remove any cinder that may have been rolled into the surface of the sheet. If the sheet remains in the acid a greater length of time, the acid may attack enough of the iron to precipitate out a black deposit of carbon or some form of combined carbon that may cause trouble in later processes. Therefore, the sheets stay in the acid the minimum length of time required to remove the ordinary mill scale. About 140 lb. of 66-deg. acid is required per ton of sheets, though this of course varies with the thickness and other characteristics of the material.

Pickling of Pipe.

In the pickling of steel and wrought-iron pipe a different method is in use. For ordinary galvanizing purposes only the exterior of a pipe needs to be entirely free of scale in contrast with pipe for conduit purposes, where both interior and exterior have to be cleaned.

Vats are made the requisite length, 4 ft. to 6 ft. wide, and from 2 ft. to 3 ft. deep. The acid bath, the strength of which varies from 3 to 10 per cent of sulphuric acid, is heated by means of injected steam. The pipes are held in two acid-proof chains and at short intervals of time they are tumbled over each other by working these chains.⁵ This operation is repeated several times during the pickling period. The tumbling of the material is necessary for several reasons. It circulates the bath, especially in the interior of the pipe and it helps loosen some of the tougher scale of the exterior by abrasion and it also washes away the detached scale. When the exterior is sufficiently clear of scale the pipes are washed in water, as with sheets, and if they are to be hot galvanized, they are dipped into a hydrochloric acid solution. Upon removal the iron chloride formed is allowed to dry and acts as a flux in the galvanizing bath.

The pickling of pipe for conduit purposes presents a different problem. There are two methods in use for giving conduit a protective coating, enameling and galvanizing. Galvanizing may be hot, wet or electro, and dry or sherardizing. The same general method of pickling with sulphuric acid is used. Conduit pipe is 10 ft. long and must be thoroughly cleaned on the interior and exterior. Any scale remaining on the inside of conduit pipe will be a sufficient cause for its rejection by the Board of Underwriters, whose requirements are rigid.

*Buehert, Stahl und Eisen, 32, 1487-9.

¹Trans. A. I. E. S., 8, 165, 1902.

²O. P. Watts, Trans. A. I. E. S., 21, 337, 1912.

³Burgess and Frazer, Trans. A. I. E. S., 19, 1906.

⁴Metallurgical and Chemical Engineering, Vol. 8, No. 10, p. 598.

Before entering on the description of methods used for pickling conduit pipe it will be necessary to describe the scale found on the interior and exterior of the pipe. This will necessarily require a brief description of the methods of manufacture.

Conduit pipe is butt welded. The skelp is heated in a furnace whose bottom is usually lined with silica. At the temperature of the furnace the acid silica and basic iron oxide unite to form one of the easily fusible silicates of iron. When the skelp is withdrawn from the furnace it is drawn into the forming die so that the bottom of the skelp will come in the inside of the pipe. The silicate slag will therefore be found on the interior of the conduit. This silicate then runs to the bottom of the conduit and solidifies as a strip of hard glassy slag that adheres quite firmly to the iron. It has been suggested that the upper part of the skelp be turned in, but owing to the rapidity with which the silicate slag ruins the dies this is never practiced. This silicate slag is the cause of the trouble that is encountered in pickling pipe.

Since the interior of conduit pipe must be free from scale it presents difficulties that are not present when ordinary pipe is treated. The acid in the interior must be kept in circulation so as to have the rate of pickling as fast on the interior as on the exterior. Various methods have been tried, of which the more successful may be described.

Pickling on end is a natural method to suggest, the material being hung on racks and lowered into the acid vats. The hydrogen liberated on the interior of the pipe and rising to the surface creates a circulation of the acid bath. This method is effectual but it is open to several objections. The hanging of the material on the racks requires an extra operation and the same is true in taking the material down. The racks occupy a large space for a given number of pipe and the space required for the pickling plant is large. Large hoists are required for the lifting of material over a distance of at least 11 ft. Should the drainage sewers be down at the ordinary distance of about 6 ft. and it is required that the waste liquor run into them, it will be seen that the tanks will have to be elevated some distance to procure proper drainage. If any break should occur in the hoisting apparatus, a fall of 11 ft. to 12 ft. of the heavily loaded racks would probably be fatal to the pickling tank. The tanks would not be adapted for the pickling of other classes of materials.

The other methods of pickling consist of treating the conduit in a horizontal position instead of vertical. In this way the layout of the plant will be simpler since the tanks will be from 2 ft. to 3 ft. deep, 4 ft. wide and 11 ft. long. The material is laid in a cradle arrangement, the couplings on the one end of the pipe causing them to be completely separated and allowing the entire surface to be subject to the acid attack. This method allows of large quantities of pipe to be pickled in one vat. The methods used for getting a good circulation of acid in horizontal tanks differ.

The intermittent method is one in which the circulation is secured in the following manner: At certain intervals of about every fifteen or twenty minutes the racks or cradles of pipe are lifted from the tanks so that the pipes are at an angle with the horizontal, allowing them to drain easily. By plunging the rack up and down in the acid several times the loose scale and dirt are removed from the interior and the pickling is again allowed to proceed. This method is especially satisfactory where a weak pickle is used.

Another method of securing circulation is to have the pipe lay in a rocking cradle working in the acid. This secures excellent circulation, but it is open to the objection that it requires machinery to be present in a pickle room and in the bath where deterioration is rapid. By introducing paddle wheels, propellers and plungers, into the vats, circulation has been secured, but none of the methods given seem to give entire satisfaction.

In the treatment of conduit, the time of pickling is judged by the interior appearance. The exterior may become free from scale in a short time, but the interior may still contain a large amount. The conduit must remain in the pickle until all of this

scale is removed or until the iron surface or screw threads become visibly corroded. When this happens the pipe must be removed from the pickle to avoid being scrapped and the remaining scale must be sand blasted or otherwise removed from the interior.

Where the pipe contains the hard ridge of iron silicate, ordinary pickling and sand blasting will not entirely remove it, and the remainder is generally left in the interior of the pipe. Such a rigid scale is detrimental since it makes fishing difficult and is liable to cut the insulation of the fished wire.

If the pipe is to be used for cold galvanizing purposes it is washed with cold water after removing from the pickle. It may be dipped in cold dilute hydrochloric acid and then taken directly to the plating tank to prevent any oxide from being formed. If the pipe is to be sherardized, it is dipped into hot water to remove the acid and iron salts, then into a lime water and finally into a hot-water bath. Upon removing from the hot water the pipe dries quickly and a minimum of iron oxide is formed. This small amount of oxide is not detrimental, but, in fact, appears to give a brighter coating. If the pipe is to be enameled it is treated in the same manner as is the sherardized material.

Strength and Temperature of Pickle and Other Details.

The strength of the pickle varies a great deal, running from 3 to 8 per cent sulphuric acid. These various strengths have their advantages and disadvantages. When a 3 per cent solution is used a larger plant is required. This is offset by the various other advantages gained. When a 3 per cent solution, at a temperature of about 170 deg. Fahr., is used the rate of pickling is slow, requiring from three to twenty-four hours to pickle ordinary pipe. With this slow rate pipe will not be easily over-pickled if it should be accidentally left in the pickle too long. The slow rate of hydrogen liberation lessens the amount of finely divided acid thrown into the air and makes it easier upon the workmen and also upon the buildings and machinery. It is also possible to "kill" a low percentage bath until almost free from acid and this prevents any acid being run into the drains. With high-percentage acid baths it is customary to "kill" a bath to a certain percentage and then run it off as waste. The high-percentage bath is also harder upon the workmen. With a low-percentage pickle the amount of acid used per ton of pipe is much less than with a high-percentage pickle, usually running below 80 lb. per ton.

The temperature of the pickle is also very important, since the higher the temperature of the bath the faster the pickling. For most purposes it is best to have a boiling bath because it not alone produces a bigger tonnage, but it also removes all grease from the article to be pickled. With a boiling pickle, however, large volumes of acidulated steam are given off, with attendant corrosion of machinery, buildings, and the bad effect upon workmen.

The pickle is run until the action becomes too slow, when more acid is added. When a solution becomes concentrated with ferrous sulphate, the pickle is "killed" and then is generally run off as waste. The concentration in ferrous sulphate when "killed" varies a great deal, standing from 12 to 25 per cent, depending upon the practice at the plant.

Few plants attempt to utilize the ferrous salts and sulphuric acid in the waste pickle, since no satisfactory method has been offered. Its utilization has been made subject of a large number of patents. One of the most recent is that of F. F. Farnham, who electrolyzes the solution and obtains iron and sulphuric acid.⁶ Where utilization is practiced a common method is to neutralize the free acid by means of iron scrap, evaporating and crystallizing the iron salts, producing the copperas of commerce. This copperas has various uses, such as the making of various shades of red pigments, Indian red and Venetian red. Recent advocacy by agricultural experiment stations of this material as a weed killer has increased the outlet for it, but with all of the uses proposed, a general recovery from all spent liquors would undoubtedly flood the market.

⁶F. F. Farnham, Patent No. 1,006,836, Metallurgical and Chemical Engineering, 9/12/1911, p. 67 O.

In galvanizing and sherardizing plants, where the zinc sometimes has to be stripped from the iron, owing to defects and other causes, an old pickle is usually used for stripping the zinc and is found to work best. The potential of iron is below that of zinc and naturally the iron in the pickle as ferrous sulphate has a tendency to plate out and remove the zinc, though little acid is present.

Air hoists are generally used in pickling rooms owing to their simplicity, ease of operation and low cost of upkeep. Though compressed air as a source of power is costly, the other advantages gained by the use of compressed air counterbalance this factor.

Large and non-corrosive drains should be provided to take care of large amounts of water and killed acid. The best floors for pickling rooms are made of the iron scale that is deposited in the bottom of pickling tanks. This material is porous, is not attacked by acids, cheap and easily laid. Cement and concrete will quickly disintegrate and are therefore not available.

Another use found for the scale and sludge deposited in the bottom of the pickling vats is for the stopping of leaks in these vats.

By merely plastering this sludge into the leaky part the leak may be easily and effectually stopped.

This paper does not attempt to give all methods in use, but rather to describe what may be considered as general practice in pipe and sheet pickling at the present date. Pickling is a branch in the metallurgy of iron and steel that is open to a large amount of improvement. Mechanical treatment and cleaning of the materials has often been tried, but sulphuric acid treatment seems most satisfactory. Pickling plants are run more or less on hit-and-miss methods and the only improvements that have been made are in the mechanical end. The fault does not lie entirely with the chemical end of the industry for improvements may be made in the hot working of the metal so as to give a product of a kind that can be more easily pickled.

The ideal condition will be where the scale would be entirely soluble in some solvent while the iron would be insoluble. Since this is probably impossible, it would be well to obtain a condition where a minimum of scale is formed and in this way requiring but a short acid attack, resulting in minimum corrosion of the iron base.

Viewing the operation from a scientific standpoint the pickling process seems to be the application of "main force and awkwardness." The object is to present a clean iron surface, and this is done not by dissolving or otherwise eliminating the coating, but by dissolving away the iron surface itself. There is here an economic loss, not only in the mill scale and of the acid, but of the good iron which is taken away. It is difficult to predict in what direction the possibility of improvement lies; it may be in the discovery of some cheap solvent for the scale itself, in some electrolytic or other method of reducing the iron oxide, or perhaps in some mechanical or physical method.

Even though the remedy is not in sight the problem is of sufficient magnitude to warrant extended investigation.

Chemical Engineering Laboratory,
University of Wisconsin.

The Pittsburgh Testing Laboratory of Pittsburgh, Pa., announces that Mr. James Otis Handy, for 23 years chief chemist, has been appointed director of research and chemical engineering and that Mr. Howard H. Craver, for 14 years assistant chief chemist, has been appointed chief chemist of their laboratories.

The Old Freibergers in America held a pleasant dinner meeting on December 20th in the Hofbrau Haus, Broadway and Thirtieth Street, New York City. All old students of the Freiberg Bergakademie, who have not yet received notices concerning such meetings are requested to send their address to Mr. C. L. Bryden, 1015 Myrtle Street, Scranton, Pa.

A Curious Manifestation of Electric Constriction.

By Edwin F. Northrup, Ph. D.

If a trough of mercury having the dimensions shown in Fig. 1 be provided, and if a piece of copper sheet of the shape shown in Fig. 2 be floated upon the surface of the mercury, then, when twelve volts from a large storage battery are applied to the ends of the mercury in the trough, the following phenomena are manifested: The copper pieces when placed at either end of the trough with its rounded end pointing toward the far end will move with a speed of about 80 centimeters

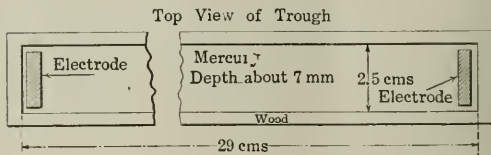


FIG. 1.—DIMENSIONS OF TROUGH.

per second the length of the trough. If the copper piece is held stationary the mercury will be greatly agitated at its pointed end.

These effects are to be thus explained: Copper has about sixty times the conductivity of mercury. The current stream lines are thus very much more dense at the pointed end of the copper piece than they are at the rounded end. The elementary mercury filaments by their mutual attraction produce a greater pressure at the pointed end than at the rounded end, hence the copper "boat" moves rapidly forward.

The writer is not unaware that practical uses may be made of this manifestation of the pressure forces in a liquid carrying a current (direct or alternating). He has designed, on paper, what appears to be an excellent ammeter, and other uses are thinkable. If in a tube containing molten metal there is fastened a piece of solid metal of high conductivity and having the shape of a plumb bob, then when current passes the fluid metal would move rapidly if the solid piece cannot move.

Very much has been written about the "pinch" phenomenon, but the writer wishes to call attention here to the fact that it was he who first thoroughly investigated this phenomenon experimentally and deduced the equations which express all the laws of the phenomenon for a conductor of circular cross-section. These experiments and laws are fully set forth in an article by the writer entitled "Some Newly Observed Man-

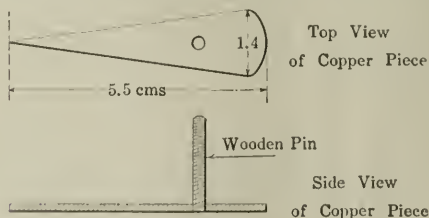


FIG. 2.—DETAILS.

ifestations of the Forces in the Interior of an Electric Conductor," *Physical Review*, Vol. XXIV, No. 6, June, 1907, pages 474-497.

In a paper by the writer entitled "A New Type of Ammeter for the Accurate Measurement of Alternating Currents Above 1000 Amperes," presented at the meeting of the American Electrochemical Society, May 17, 1909, the writer showed how the forces in question had been applied by him to the construction of an accurate type of ammeter for measuring large direct or alternating currents.

Palmer Physical Laboratory, Princeton, N. J.

The Western Metallurgical Field.

Eastern and Western Spelter Conditions.

The situation with regard to the mining of zinc ores and production of spelter has not greatly changed during the past year. The high price of spelter naturally has spurred the miner to greater activity. There has been an increase in ore tonnage both East and West, but not along the same lines. The Joplin increase is shown in the marketing of 7000 to 8000 tons of concentrates weekly where the 1911 output was but 5000 tons. The ore is being extracted as rapidly as possible and rushed to market.

The Western increase is not so much in larger weekly shipments but in very substantial additions to ore reserves awaiting extraction. This reflects quite clearly the difference in operating conditions between East and West. Joplin mine and mill equipment is comparatively cheap, and the machinery is readily salable at second hand. Under these conditions ore reserves are not essential. Furthermore, the product is high-grade and meets a sure and immediate market. The Western operator requires a more expensive equipment, usually unsalable when worn, and finds a much more irregular and restricted market for his lower-grade output. Therefore, ore must be in reserve before he may safely provide proper equipment. It is quite likely, however, that actual shipments from the West during the latter part of 1912 would have been larger but for shortage of cars.

It has become more certain during the year that a considerable portion of the Western zinc tonnage will come from scattered ore bodies of moderate size rather than from large deposits, and these smaller ore bodies are quite uniformly complex and difficult to treat. No large bodies of high-grade ore exist in America to compare with Broken Hill in Australia, say, where 5,000,000 tons of tailings still await treatment and where there are still millions of tons of unbroken ore in the mines.

The only deposit approaching this is that of the Butte and Superior Mine in Montana, where about 3,000,000 tons of 20 per cent zinc is said to be fairly in sight. There is a tendency among some mining men to assume that methods adopted by the owners of this property may be blindly followed by most others having complex ores awaiting treatment. This may easily prove dangerous ground to take.

The Butte and Superior ore is notably free from iron and copper, as complex ores go, so that concentration and flotation yield high-grade zinc concentrates not requiring separation. Even the adjoining mines in Butte so far as now opened do not show the same clean ore.

Cost of Milling at the Liberty Bell.

At the mill of the Liberty Bell Gold Mining Company, Telluride, Colo., recovery of gold and silver is made by amalgamation, concentration and cyanidation. Situated in a district noted for high freight rates, wages and cost of living, the cost of treating a low-grade ore by this process is remarkably low, and specially interesting to engineers having similar obstacles to overcome. The figures given in the right-hand column represent operating conditions in October, November and December, 1911, and are based on the treatment of 104,460 tons of ore.

Kick's Law and Rock-Crushing Efficiency.

The use of Kick's law in computing the crushing efficiency of different machines has evoked considerable discussion, particularly on the Rand, where Mr. H. Stadler has been an advocate of this method. Recently some interesting work has been done at McGill University, in which Rittinger's law has been disproved and Kick's law adopted as more nearly in accord with the truth. The following extract is from a report issued by the University:

ROCK CRUSHING.—"A series of investigations have been carried on by a succession of fellows during several years on rock crushing, and the consumption of power has been very accurately measured for several standard types of crusher, work-

ing on different kinds of rock and crushing to different degrees of fineness. The result of these experiments is to finally disprove the 'law' of crushing originally enunciated by Rittinger

LABOR.		Cents per ton.	SUPPLIES		Cents per ton.
Superintendence	2.68		Pipe lines ¹		0.61
Heating	1.29		Bins ¹		0.85
Electric Plant	1.06		Buildings ¹		6.28
Lubricating	0.34		Electric Plant		1.11
Pumping Plant	2.12		Pumping Plant		1.98
Watchman	0.83		Fuel and Heating Plant		4.71
Examination and Tests	0.68		Tools ¹		0.43
			Cyanide ¹		26.14
			Lime ¹		4.26
			Lead Salts		3.30
			Light & Power ²		3.78
			Oil & Waste		0.91
			Assaying & Melting ¹		3.47
			Examination & Tests		0.27
			Miscellaneous		0.01
Total general mill labor		9.01	Total general mill supplies		\$8.11

¹Indicates combined labor and supply.

²Power in this charge represents that used in pumping between departments.

Direct Charges to Several Departments.

LABOR.		Cents per ton.	SUPPLIES		Cents per ton.
Crushing	5.31		Crushing		1.99
Stamping	9.35		Stamping		9.73
Regrinding	1.04		Regrinding		6.65
Settling & Agitation	1.50		Settling & Agitation		3.90
Filtering	3.72		Filtering		5.00
Concentrating	4.76		Concentrating		1.99
Amalgamation	4.26		Amalgamation		2.24
Precipitation	1.37		Precipitation		7.55
All Labor	40.32		All Supplies		97.22
All Supplies	97.22				

All Local Cost137.54 or \$1,375.4 per ton ore treated.

and accepted by nearly all authorities since his day, and to show that the more recent theory advanced by Kick is either true or so nearly so that it affords a close working approximation to the truth.

"Kick's 'law' was originally stated in 1885 as a general proposition in mechanics, and it is only within the last few years that it has been applied to ore dressing first by Stadler in South Africa, and more recently by the McGill staff. But in terms of rock crushing it may be stated as follows: The power necessary to crush a given quantity of any given rock from any known size to any other known size, will be directly proportional to the reduction in volume of the particles—in other words have a ton of rock all in pieces, say, 1 in. in diameter, it will take almost three times as much power to crush them to pieces $\frac{1}{2}$ in. in diameter as to pieces $\frac{3}{4}$ in. in diameter; or it will take twice as much power to crush from 1 in. to $\frac{1}{4}$ in. as from 1 in. to $\frac{1}{2}$ in.

"The law as above stated is of course impractical, first, because it makes no allowance for the imperfections of the crushing machinery, or for the difference between one kind of crusher and another, and second, because it is not a practical possibility to feed a crusher with rock all broken to exactly the same size; but as Stadler has pointed out, if the law is theoretically true, it is possible to take any quantity of mixed rock, and by sizing it before crushing and weighing the different sizes, then crushing, say, half the lot in one machine and the remainder in another, and finally by again sizing and weighing

the products, it is possible to determine the relative useful work done by each machine, or in other words to determine the comparative efficiencies of different crushers.

"The McGill experiments were first directed to settling the question of the truth of the theory and, as stated above, the results of a very large series of experiments agree with the law to within the reasonable limits of error of experiment on so variable a material as broken rock.

"The next thing to be done was to compare the efficiency of a single crusher, say, a Gates breaker, over different parts of its range, i.e., to determine whether it gave as high an efficiency when working from say, 4 in. to 1 in. as from 4 in. to 2 in. or from 2 in. to 1 in.; and similarly to study other crushers such as stamps, rolls, tube mills, etc., with a view to finding the most efficient range of each machine and the comparative efficiencies of the different machines.

"Obviously the amount of ground to be covered is enormous, especially as the experiments have to be carried out on a large scale yet with the utmost care to eliminate errors. Therefore, only a small part of the work has as yet been completed, but enough has been done to justify the preparation of a preliminary report which is now being prepared for publication, and it is confidently believed that it will soon be possible to state with a very considerable degree of certainty just what types of crusher are most suitable for particular degrees of crushing, and to state what output may be expected for any rock for which the confines have been determined."

Utah.

At a recent meeting of the Utah Society of Engineers, at Salt Lake City, Mr. H. D. Randall, of the General Electric Company, read a paper on the Hydro-electric Power Resources Tributary to Salt Lake City. He gave a list of the developed water-power plants, excluding a few of the smaller municipal systems, showing a total rated development of 66,850 kw. Of this quantity, Mr. Randall estimated that about 45,000 kw. is available for delivery in the tributary markets. As to the future possibilities, he estimated that 105,750 additional kilowatts could be developed, of which about 90,000 kw. would be always available.

The year 1912 closed without the discovery of any important new mining districts in Utah. Increased activity was noted, however, in all the old districts, with some progress in metallurgical work on ores from old or abandoned mines. The Mines Operating Company, which acquired the old Ontario at Park City last year, is reported to have completed its experimental work on the old low-grade ore, and has adopted a chlorination process. Cyanidation was first tried, and it was thought that the process would be successful, but further work indicated that it would not be best adapted to the general run of ore, and further experiments resulted in the decision to use a chloridizing roast, followed by leaching. Details of the process have not been made public, and probably will not be until the new mill is working satisfactorily.

The Utah Copper Company is again operating at full capacity, following the labor strike of last fall. On December 1, general manager Jackling issued a statement to the effect that the company was then mining 20,000 tons per day, and that substantially a full crew of men was at work. He also announced an increase of pay for pit men, track men and laborers of all classes, as well as for employees in the mechanical department. This applies to men working at Bingham, and is consistent with the increase announced a month earlier for men at the Garfield concentrating mills. The increase ranges from 20 to 25 cents per day, and will be effective during the period of relatively favorable prices in the copper market.

Dry Concentration in Colorado.

Events of recent occurrence indicate that dry concentration is to receive an impetus in Colorado, due to the activity of the Sutton, Steele and Steele Manufacturing, Milling & Mining Company. The company succeeds Sutton, Steele & Steele, Inc., of Dallas, Texas. A 50-ton custom and demonstrating mill is being built in Denver, where a tract of three acres has been

bought in the south part of the city. It will be ready for operation early this year, and will be the principal testing station of the company. In addition to this mill, the company will operate also the old Yak mill of the American Zinc Extraction Company, at the mouth of the Yak tunnel in Leadville. This mill has been closed for over a year, but has been purchased by the Sutton company, and will be in operation early this year, treating low-grade zinc sulphide ores of the Leadville district. Dry table concentration will be the system used at both these mills.

Company Reports.

The twelfth annual report of the *Consolidated Mercur Gold Mines Company*, Utah, has just been issued, covering the year ended June 30, 1912. In the eleventh report it was shown that the company's operations were on a precarious basis, with a prospect of a complete cessation of work. The present report does not hold out much hope for better conditions, although during the past year some new ore was found which has enabled the company to continue. The total quantity of ore milled was 201,652 tons, consisting of base ore (34 per cent), oxidized ore (59 per cent) and tailing (7 per cent) and yielding a recovery of \$2.45 per ton. The mill tailing had an average value of \$0.82. The total cost of milling is given as \$1.10 per ton. Cyanide consumption was 0.73 lb. per ton of ore; lime consumption was 15.27 lb.; zinc consumption, 0.5 lb.

Synopsis of Chemical and Metallurgical Literature.

Gold and Silver.

Metallurgical Costs at the Homestake.—From an exhaustive treatise on the metallurgy of the Homestake ore, published by ALLAN J. CLARK and W. J. SHARWOOD, in *Bulletin No. 98*, Inst. Min. & Met., we take the following summary of costs and metallurgical practice:

BREAKING ORE AT SHAFTS.		COST PER TON.		Approximate Per-centage of Total Cost of Break-ing.
		Range.	Average.	
		Cents	\$	
Power:	Steam power, including labor of engi-neers, firemen, etc.	3-4	0.0351	58.5
Attendance:	Labor, including the handling of cars from shaft to grizzly	1-1.5	0.0125	21.0
	Lubricants	0.08-0.14	0.0009	1.5
Maintenance:	Materials and supplies for repairs and renewals	0.65-1.2	0.0091	15.0
	Labor on repairs and renewals	0.14-0.38	0.0024	4.0
Total		5.7-6.6	0.0600	100.0

The cost of distribution of the broken ore to the mills by means of the tramway is approximately 2 cents per ton.

The following costs do not include amortization charges nor water supply; they do, however, cover superintendence, ordinary repairs and renewals, and a proportion of extraordinary renewal charges when the life of an installation may be estimated, as, for instance, the tube-mill lining and the pipe-lines conveying sand and slime.

The cost of Stamp-milling, at the lead mills of the company, for a period of twelve months, is as follows:

	Per ton crushed.
Power (including engineer's labor)	\$0.104
Machinery (renewals)	0.094
Operating labor	0.091
Silver plating	0.005
Mercury	0.005
Repairs (reconstruction of batteries, bins, etc.)	0.041
Miscellaneous	0.006
	\$0.346

At the North Side mills the cost is about \$0.27, the difference being almost entirely due to the electric drive at those mills.

Regrinding.—Costs for three-month period, 1910:

Per ton ground.	
Power	\$0.0634
Labor	0.0808
Renewals	0.1064
Miscellaneous	0.0064
	\$0.2570

METALLURGICAL SUMMARY.

A summary of the metallurgical operations, showing the amounts recovered at the various stages of the treatment, concludes our description. This covers the entire metallurgical system for a period of four months, when the ore was rather below the average in value. The North Side mills, where 36 per cent of the stamps are located, are often supplied with surface ores which are cheaply mined through glory-holes and open-cuts, so that ore of a total gold content of less than \$1.50 per ton may be treated at a profit. During the period covered by our summary the ore crushed at these mills aver-

COSTS AT CYANIDE PLANTS, JANUARY-FEBRUARY, 1911, PER TON TREATED.

	No. 1. SAND PLANT: 93,262 TONS.					No. 2 SAND PLANT: 43,609 TONS.					SLIME PLANT: 99,300 TONS.				
	Labor.	Power.	Chemicals.	Other Supplies.	All.	Labor.	Power.	Chemicals.	Other Supplies.	All.	Labor.	Power.	Chemicals.	Other Supplies.	All.
	\$	\$	\$	\$	\$	\$	\$	\$	\$	\$	\$	\$	\$	\$	\$
Transportation.....	0.0013			0.0015	0.0028				0.0001	0.0028					
Classification.....	0.0092	0.0020		0.0001	0.0113	0.0142				0.0142					
Thickening.....											0.0035				0.0035
Neutralization.....	0.0021	0.0005	0.0188		0.0214	0.0038	0.0003	0.0191		0.0232	0.0057	0.0004		0.0001	0.0062
*Treatment.....	0.0341	0.0177	0.0939	0.0034	0.1491	0.0332	0.0092	0.0502	0.0031	0.0957	0.0383	0.0181	0.0653	0.0138	0.1355
†Precipitating.....	0.0059	0.0015	0.0041	0.0008	0.0123	0.0057	0.0009	0.0038	0.0005	0.0109	0.0070	0.0017	0.0067	0.0011	0.0165
‡Miscellaneous.....	0.0014	0.0016		0.0109	0.0139	0.0028	0.0005		0.0107	0.0140	0.0066	0.0013		0.0054	0.0133
Assaying.....	0.0039		0.0010	0.0014	0.0063	0.0045		0.0002	0.0005	0.0052	0.0023		0.0007	0.0011	0.0041
Refining.....	0.0032		0.0011	0.0009	0.0052	0.0028		0.0010	0.0006	0.0044	0.0038		0.0029	0.0018	0.0085
Superintendence.....	0.0074				0.0074	0.0115				0.0115	0.0099				0.0099
All.....	0.0685	0.0233	0.1189	0.0190	0.2297	0.0812	0.0109	0.0743	0.0155	0.1819	0.0771	0.0215	0.0756	0.0233	0.1975

COMBINED COSTS: ALL CYANIDE PLANTS: 236,171 TONS.

Labor.....	0.0744 per ton treated
Power.....	0.0203 "
Chemicals.....	0.0927 "
Other Supplies.....	0.0202 "
	0.2076

CHEMICALS CONSUMED DURING ABOVE PERIOD: LBS. PER TON TREATED

	No. 1 Plant.	No. 2 Plant.	Slime Plant.	Combined Consump., All Plants.
	lb.	lb.	lb.	lb.
Sodium Cyanide.....	0.393	0.209	0.225	0.280
Lime.....	5.01	5.24	4.55	4.87
Zinc Dust.....	0.063	0.075	0.114	0.188
Hydrochloric Acid.....			0.499	

*Includes filling and discharging. †Includes pumping of all sludges. ‡Includes heating and lighting.

DETAILED METALLURGICAL SUMMARY FOR THE FOUR MONTHS ENDING AUGUST 31ST, 1910.

	VALUE RECOVERED.		WASTED OR LOST IN RESIDUES.	
	Percentage of Total Ore Value.	Value per Ton of Ore Crushed.	Percentage of Total Ore Value.	Value per Ton of Ore Crushed.
STAMP MILLS —Batteries.....	35.596	\$1.280		
First row plates.....	27.007	0.970		
Skimmings.....	2.901	0.104		
Trap sands.....	1.059	0.038		
Second row plates.....	3.146	0.113		
Third row plates.....	1.641	0.058		
Fourth row plates.....	0.648	0.023		
	71.998	\$2.586		
	0.735	0.026		
RE-GRINDING MILL				
CYANIDE PLANTS—Sand Treatment.....	13.737	0.493	4.923	\$0.17
Slime Treatment.....	7.257	0.261	0.993	0.03
	20.994	0.754	5.916	\$0.212
SLIME NOT TREATED			0.357	0.013
Totals.....	93.727	\$3.366	6.273	\$0.225

Total Value of Ore Crushed, per Ton, \$3.591.

This Table is very approximately correct. Slight errors exist in the stamp-mill distribution, as the variations in the silver contents of the various amalgams have not been considered, the distribution of the total amalgam value being based on the gold content only. The actual value of the amalgam from the lower rows of plates is therefore a trifle higher than here stated, and the battery correspondingly lower.

Lime-Burning.—Cost of a ton of lime (CaO), delivered on board cars at kiln, twelve-month period:

Stripping and Quarrying—Labor.....	\$1.03
Supplies.....	0.04
	\$1.07
Burning and Loading—Labor.....	\$1.13
Fuel.....	2.65
Supplies, Repairs.....	0.11
	3.89
	\$4.90

aged \$2 gold per ton, while the average for the entire system was \$3.50. About 2 per cent of the ore crushed was wasted in the form of slime.

Pan renewals, per ton, ground in pans..... \$0.125

Tube renewals, per ton ground in tube-mill..... 0.036

Action of Calcium Sulphate on Cyanide Solutions.—In the *Journal of the Chem. Met. & Min. Soc. of S. Africa* for August, 1912, Joux Watson contributes some data on the effect of calcium sulphate on cyanide solutions, being a discussion of an original paper in the April, 1912, issue of the same journal, by ANDREW F. CROSSE. Mr Watson made some

experiments to ascertain the extent to which calcium sulphate decomposes a solution of potassium cyanide. Mill solutions used for sand and slime treatment respectively, were divided into separate portions and placed in bottles containing one litre each. To one portion of each kind of solution he added 2.5 g. of pure dried calcium sulphate and agitated well. The bottles were allowed to stand unstopped in the laboratory, and were shaken each day. The following table gives the results of tests made on treated and untreated samples of solution.

Time of Analysis.	UNTREATED SAND SOLUTION		TREATED SAND SOLUTION.	
	Free KCN	Alkalinity CaO.	Free KCN.	Alkalinity CaO.
As taken from sump	0.223%	0.059%	0.223%	0.059%
Three and one-half hours after addition of CaSO ₄ ...			0.214	0.059
After 2 days	0.214		0.202	0.059
After 4 days	0.192		0.184	
After 12 days	0.183	0.053	0.164	0.048
Percentage loss of KCN and CaO.....	17.93	10.2	26.45	18.6

A similar table shows results obtained on the solution used for slime treatment, in which the percentage loss of KCN and CaO were 46.8 and 52.8 for the untreated, and 59.3 and 87.6 for the treated solution. It must be remembered that the untreated solutions contained some calcium sulphate.

In a solution of sodium cyanide prepared with distilled water, and having the approximate strength of the sand solution used before, Mr. Watson added 1 g. lime to 2700 c.c. and divided the solution in three parts and allowed to stand for one day. Then to one part he added 2.07 g. CaSO₄, to another 29.7 g. Epsom salt, leaving the third portion untreated except with the previously added lime. Tests were then made on the consumption of KCN and CaO as before, with the result that after seven days the following percentage losses of KCN and CaO were noted: In the untreated solution, 0.4 and nil; in the CaSO₄ solution, 4.1 and 5.1; and in the Epsom salt solution, 22.3 and 26.4.

Cyanide Practice at Komata Reefs Mill, New Zealand.—In the *Mexican Mining Journal* for November, 1912, S. D. McMIKEN describes the changes made in the treatment of the Komata Reefs ore. Originally the ore was dry-crushed in stamps and cyanided. This failed to recover much coarse gold, so wet-crushing and amalgamation were adopted preceding cyanide treatment. Still later the process has been altered to include fine-grinding and slime treatment.

The ore, which is a hard, cherty quartz mixed with calcite and containing little sulphide, is crushed to 1½-in. size and fed to twenty stamps. The mortars are fitted with screens having 4½ holes per linear inch. Stamp capacity is 5 tons per stamp per 24 hours. The stamp product is roughly classified in pointed boxes, the underflow being fed to tube-mills at a consistency of 1:1. The tubes are 16 ft. by 4 ft., lined with Brown patent liners which last twenty-two months, grinding 38,500 tons of ore. The consumption of iron is 0.48 lb. per ton of ore, and of pebbles 2.2 lb. per ton of ore. Power required for two mills is 50 hp, the tubes running at 28 r.p.m. The following table shows the work done by the mills.

	On 20 Mesh %	On 25 Mesh %	On 30 Mesh %	On 200 Mesh %	Through 200 Mesh %
Tube-mill (16 ft. x 4 ft.)	25.5	25.2	16.5	14.8	18.0
Tube-mill charge		1.0	22.0	25.9	51.1

The finely-ground pulp is amalgamated on Muntz metal plates. This metal has been found superior to copper, being more easily kept clean, and absorbing less gold. Forty per cent of the gross value of the ore is saved by amalgamation.

After amalgamation the pulp is separated into sand and slime, which receive separate treatment.

The sand contains 40 per cent of material which will pass 200 mesh. Percolation is carried on in steel vats with filter bottoms of jute sacking. The sand is drained to 15 per cent moisture and treated with a wash of 0.05 per cent KCN solution. After percolation this wash is precipitated and run to waste. It serves the purpose of reducing the consumption of cyanide when the strong solution is put on, as it contains considerable lime. A strong solution of 0.8 per cent KCN is then run on and circulated through the sand for eight days before precipitation. It is aerated and treated also with ozone which is found to increase its activity. The strong solution is followed by one of 0.08 per cent KCN and then by water. The usual time of washing is fourteen days. Extraction is 93 per cent of the value of the sand.

The slime is thickened in Dorr thickeners, being first treated with lime at the rate of 20 lb. per ton. The Dorr discharge contains 40 per cent moisture. It is pumped to the Brown (Pachuca) agitators which are 38 ft. high and 10 ft. diameter. Power required for ten agitators is 8 hp, the air being compressed to 22 lb. per square inch. These tanks have been tried with short air-lift pipes, but they did not prove satisfactory on account of the tendency of the coarse slime to separate from the fine. The slime in the agitators is diluted with 0.1 per cent KCN solution until it contains about 65 per cent moisture. A common charge is 40 tons dry slime. Air agitation continues for five or six days, after which time the pulp is again thickened to about 45 per cent moisture. The overflow from this thickening is filtered through sand and precipitated. The thickened slime is filtered in a Moore filter of the "turnover" type invented by Mr. McMIKEN. A vacuum pump giving a vacuum of 24 in. mercury builds a cake 1¼ in. thick in 30 minutes. A wash of barren solution is given for about the same length of time required to build the cake. No water wash is given. The cake is discharged with 25 per cent moisture, with a loss of soluble gold amounting to 12 cents per ton, and of silver 4 cents per ton. The total extraction is 95.6 per cent of the value of the slime.

Precipitation is by zinc dust. The precipitate is given a sulphuric acid treatment (10 parts water to 1 of acid), after which it is dried and mixed with powdered borax glass in the proportion of five parts precipitate to one of borax. Melting is done in plumbago crucibles. The bullion is 900 fine, the ratio of gold to silver being 1:8. Amalgam from the plates yields bullion 990 fine, containing gold 600 and silver 300.

The total extraction from this ore is 96.4 per cent. This high extraction is assigned mainly to having large cyanide capacity per stamp.

Gitsham Cyanide Process.—The *Kalgoorlie Miner* gives some details of this modified cyanide process, in which ordinary potassium cyanide solution is acidified with sulphuric acid, giving rise to hydrocyanic acid. The solution is allowed to percolate through the ore in the usual way, but before precipitation it is neutralized with lime water. After precipitation it is again acidified and used as an extractor. The inventor claims that antimonial and copper ores can be worked without trouble, and that refractory ores generally yield their precious metals to this solution.

Photolyse.—Under this caption GASTON JACQUIER describes in the *South African Mining Journal* for Sept. 14, 1912, a process of precipitating cyanide solutions, in which photochemical effects play a part. Wooden tanks are provided with removable aluminium linings, and aluminium sheets are adapted to be suspended in the tank. The latter are arranged at right angles to the direction of the magnetic needle, and must be exposed to the sunlight. After twenty-four hours exposure all the gold and silver contained in the solution is precipitated on the aluminium sheets which are then removed and brushed to collect the precipitate. The aluminium does not in any way change or lose weight. The reader will recall in this connection the "luminator" purification of water, mentioned in our issue for November, 1912, page 762, in which

aluminium is used in direct sunlight for the treatment of water for boilers. Here also a photochemical effect is suggested, as the aluminium does not change nor lose weight.

Estimation of Lead in Cyanide Solution.—In the *Mexican Mining Journal* for December, 1912, J. E. CLENNELL gives methods for determining lead in cyanide solutions. For rapid approximate determinations the author proceeds as follows:

The solution is filtered if necessary, and 100 c.c. is agitated with an excess of sodium carbonate or bicarbonate, usually 0.3 to 0.4 g. is sufficient. The precipitate which consists almost exclusively of lead and calcium carbonates, without copper, gold or silver, is allowed to settle, and is then washed on a filter until free from cyanide. The residue is then dissolved in hot 20 per cent acetic acid, and the liquid transferred to a measuring flask of from 100 c.c. to 500 c.c. capacity. A measured volume of standard potassium bichromate solution is then added, more than sufficient to precipitate the lead, and the flask filled to the mark with distilled water. After thorough agitation the precipitate is allowed to settle, and an aliquot part of the solution drawn off in a pipette. Filter if necessary. The excess of bichromate is titrated with a ferrous sulphate solution of corresponding strength, first adding a few drops of hydrochloric acid. Potassium ferricyanide is used as an indicator as in the bichromate titration for iron. The bichromate and ferrous sulphate solutions should be standardized against a lead solution of known strength. A convenient standard is one containing 1 g. lead per litre. One gram of pure lead foil is dissolved in the least possible quantity of dilute nitric acid, sodium carbonate is added until a permanent precipitate is formed, and then acetic acid until everything is in solution. The solution is made up to one litre with distilled water.

An alternative method is as follows: 100 c.c. cyanide solution is heated to boiling and a slight excess of sodium sulphide added. One or two cubic centimeters of a 10 per cent solution is sufficient. The precipitate contains sulphides of lead, silver, zinc and possibly mercury. Wash by decantation and finally on the filter until the residue is free from soluble sulphides. Wash the precipitate off the filter and cleanse the latter with 10 c.c. of boiling 50 per cent nitric acid, allowing the liquid to run into the vessel containing the precipitate. Concentrate to small bulk, cool, remove globules of sulphur, add to c.c. sulphuric acid and evaporate to white fumes. Cool, add 50 c.c. water and boil; cool and allow to settle, using 10 or 20 c.c. alcohol if the precipitate is small. Filter and wash with 5 per cent sulphuric acid. The filtrate contains the zinc and silver. Dissolve the lead sulphate in sodium or ammonium acetate, transfer to a measuring flask and add bichromate solution as in the preceding method and finish as there indicated.

Flotation of Minerals.

"Agitation Froth" Process.—In our last issue we published an abstract of an article by J. W. ASHCROFT, taken from *Bulletin* No. 97, Inst. Min. & Met., describing the operation of the flotation process on a copper ore in New South Wales. In *Bulletin* No. 98 of the same society, Mr. H. L. SULMAN gives a discussion of Mr. Ashcroft's paper, in which some essential details of the minerals separation process are given. Mr. Sulman suggested that the term "froth flotation" or "agitation froth" process be applied to the method under discussion, since an essential factor in successful treatment was the obtaining of a thick, substantial or "matted" froth, necessary to support the coarser mineral particles. If the floating mass becomes thin or skin-like, it does not afford the same opportunity to recover coarse mineral. If necessary, the area of the flotation chamber has to be constricted to produce a thick froth.

By mere vigorous agitation of ore and water and the addition of a minute proportion of oil, say 2/3 lb. crude eucalyptus oil per ton of dry ore (as in the case under discussion), froths were produced which carried over 80 per cent of the total mineral as a concentrate assaying over 20 per cent copper.

This was obtained from an ore assaying only 3½ per cent copper. From poorer ores the flotation process was obtaining elsewhere 90 per cent of the mineral as a concentrate containing 22 per cent or 23 per cent of the metal.

While the process is by no means a delicate one and requires no intricate adjustments or highly skilled supervision, certain conditions are demanded, as follows:

The ore sent to the flotation department should be of uniform consistency, say four or five parts water to one of ore.

The feed should be uniform as to rate of flow and mineral content. If either factor is subject to wide variation, plant adjustment would be required accordingly.

The mesh to which the ore is crushed should be that at which the best results can be obtained, and should be determined for each ore.

The power applied to the agitation spindles should be uniform; the agitation should not be alternately rapid and slow.

Failure to apply these conditions will give poor results which cannot properly be charged to the process.

The "froth flotation" process is now being applied to a wide variety of ores; zinc-lead, copper, silver, molybdenum and others. At the Braden mine, South America, a plant is installed to treat 3600 tons per day of low-grade chalcocypite.

The power required for agitation is about ¼ hp to ¼ hp per box; thus an eight-box unit would require about 6 hp. The agitation shafts revolve at about 300 to 350 r.p.m., the peripheral speed of the impellers being some 1500 ft. per minute. The cost of a flotation plant of this type is far less than for a wet concentration unit of similar capacity.

Mr. WALTER BROADBRIDGE contributed the following data regarding the flotation methods at the Braden Copper Co., in Chile. A change was made at this property, substituting the minerals separation process for wet concentration of low-grade copper ore. At present a 3000-ton flotation plant is under construction. The copper minerals are chalcocypite, bornite, chalcocite, cuprite, metallic copper and the carbonate and silicates of copper. The recovery by wet concentration is given as 50 per cent to 60 per cent. In a test run by flotation the averages were as follows: Ore, 2.61 per cent copper; tailing, 0.56 per cent; concentrate, 21.19 per cent; recovery, 80 per cent. Consumption of chemicals was 3.57 lb. H₂SO₄ and 2.33 lb. Texas and wood tar oils, per ton of ore treated. The estimated cost was 10d. (80.20) per ton, based on a treatment of 1,000,000 tons per annum.

Combustion.

Surface Combustion.—In *Feuerungstechnik* of November 15, 1912, Dr. C. KINZBRUNNER reviews the different theories which have been proposed by Schnabel, Bone, Ellis, Leather and Lucke concerning the mechanism of "surface" or "flameless combustion" and reaches the conclusion that its essential feature is the combustion of large quantities of gas and air in approximately theoretical proportion concentrated into a small space, while the use of porous refractory materials permits an automatic regulation of the combustion zone within the pores of the mass and the incandescent surfaces exert a strong catalytic effect. The combustion is not flameless, but as the place of the flame is within the incandescent porous mass, it is not visible to the outside. "With respect to its practical applications the invention may become of about the same importance for the technics of firing as the invention of the incandescent gas mantle has become for lighting."

Recent Chemical and Metallurgical Patents.

Iron and Steel.

A method of treating metal ingots to prevent piping and segregation has been patented by GROVE H. BENJAMIN, of New York City. As a preliminary step in the production of the ingot, the molten metal is poured into a mold which is so constructed that one of its sides shall have a higher temperature than the other three sides. This may be accomplished by heating one side of the mold or by providing cooling means for the other sides. By casting the ingot in such a mold one of its

sides will retain a higher temperature than the others and the soft metal on the hotter side will accommodate the shrinkage on the cooler sides and prevent the formation of pipes. When the ingot is cool enough to handle it is removed from the mold and placed in a machine between pressure plates mounted on shafts and carried in suitable bearings. One of the plates has an adjustable part that can be used to compress a part of the soft side of the ingot while cooling takes place. Also, during the exertion of this pressure, the ingot can be rotated to overcome the tendency toward segregation of the materials forming the ingot. (1,038,271, Sept. 10, 1912.)

Gun Barrel and Similar Steel.—**Franz Hatlanek**, of Kladno, Austria, endeavors to improve the wearing qualities of steels which are subject to repeated alternating heating and cooling, such as the barrels of guns and cannons. The continual expansion and contraction of such steel causes a destruction or a change in the texture of the material which weakens it. The metal is then less capable of enduring any other additional strains. An addition of from 4 per cent to 5 per cent copper is said to perform this improvement. It has the further advantage of making the steel more indifferent against the action of residues of the combustion of explosives. The pits and spots of rust which form in an insufficiently cleaned weapon impair its shooting accuracy considerably. It has been found that a steel with 7 per cent Ni and 4 per cent Cu did practically not rust in a moist atmosphere of nitric acid, nitric anhydride and carbonic acid. The resistance to rusting becomes specially evident when the percentage of nickel in a gun steel is made slightly higher than the usual 3 per cent. (1,005,115, Oct. 3, 1911.)

Flue Dust.

Dust Separators.—Devices for separating dust and solid particles from blast-furnace gases by subjecting them to centrifugal action have been patented by several inventors. Recently **JULIUS A. DYBLIE**, of Joliet, Ill., has proposed the device shown in vertical and horizontal sections in Fig. 1. The vertical section is on line 1-1 of the horizontal, and the latter on line 2-2 of the former.

The catcher consists of a casing 10 mounted on a base 11. The dust-laden gas enters at 18 and passes through the spiral passage 21.

The dust is forced to the outer edge of the spiral and falls through its open lower edges into the dust-collecting chamber 12, and thence to the outlet 14, where it is held by the cap 15 until removed. A certain portion of the dust is caught also by the hook 25, and drops downward into the collecting chamber.

The clean gas passes through the aperture 24 into the central chamber 22 and thence upward into the portion 30 of the dust-catcher to the outlet pipe 31. Any dust which passes into the central chamber 22 can pass downward by gravity to the chamber 12. (1,045,532, Nov. 26, 1912.)

In Fig. 2 is shown the device of **LAWRENCE ANDERSON**, of Chicago, Ill., in its relation to a blast furnace. The separator consists of a metal pipe C^1 having the form of a helical coil C^2 . The force of the blast from the furnace drives the dust-laden gas through the coil, throwing the solid particles to the outer surface thereof. Here they are arrested in the pockets d^1 from which lead pipes d^2 ending in extensions d^3 , which dip into water, thus sealing the pipes against the exit of gas. The coil shown is divided into three parts of four convolutions each. In the first a dry separation takes place; in the second a water spray is introduced by the water pipe C^3 and its extensions. In the third section the cleaned and washed gas is dried by centrifugal action, and finally delivered to a tank connected with the outlet C^4 . (1,039,008, Sept. 17, 1912.)

Treatment of Flue-Dust.—**JOHN I. SOUTHER**, of Westmont, Pa., has patented a method of treating flue-dust by agglomerating or sintering it into a rough lumpy condition by means of high heat applied to the dust in a revolving cylindrical furnace, using blast-furnace gas as a fuel. (1,041,363, Oct. 15, 1912.)

Zinc, Copper and Nickel.

Zinc Digester and Filter.—One of the patented features of the zinc reduction process of **PERCY C. C. ISHERWOOD**, of Leytonstone, England, is a digester and filter illustrated in Fig. 3. The patent is assigned to the Refractory Zinc Ore Treatment Company, a corporation of New York. The object

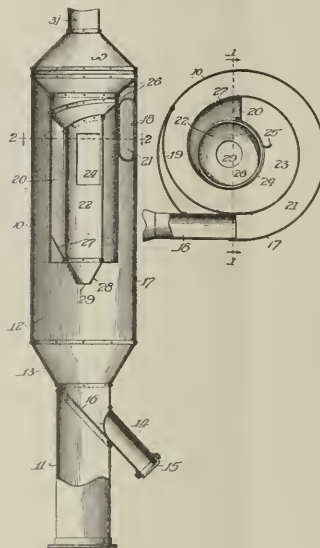


FIG. 1.—DUST COLLECTOR.

of this invention is to provide means for filtering a solution of zinc while hot, so that the zinc salt cannot crystallize out of the solution. The digester consists of a vessel a mounted on trunnions and adapted to be inverted. The cover b has a valve d , and when in place holds a filter c at the point shown. A mixer e is used to stir the charge of zinc-bearing material during the process of solution. Heat may be applied in any manner, as by a gas burner f . When solution is complete the digester is in-

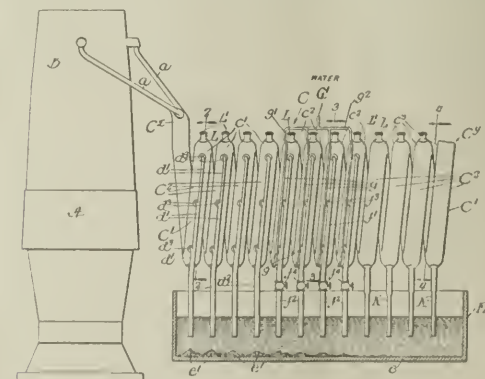


FIG. 2.—DUST SEPARATOR.

verted, whereupon the mass within falls on the filtering medium and the zinc liquor passes through due to the pressure inside the tank. The operation is similar in some respects to that of a chlorination barrel, but is only one step in a process which involves solution of zinc in sulphuric acid and ultimate electrolysis. (1,043,939, Nov. 12, 1912.)

Repairing Copper Converters.—An invention designed to facilitate the repair of converters has been patented by E. A. C. SMITH, of Baltimore, Md. Usually it has been necessary, in repairing a large converter, to remove it from service and allow it to cool sufficiently to permit a workman to enter it. Mr. Smith has patented a feature which will make the region of the tuyeres accessible without the usual trouble and expense. The arrangement is such that a portion of the casing may be removed so that the lining in the tuyere zone can be reached from without by the workman. The use of the invention permits the repair of tuyeres and lining within a few hours, whereas formerly several days were required for the same work. When it is desired to repair the lining at any point along the tuyere zone, or to remove a corroded or defective tuyere, the casing section nearest the point of repair is removed, thus leaving an aperture through which the repair can be made. On completing the work, the removable section is replaced and wedged in position. (1,044,587, Nov. 19, 1912.)

Hardening Copper.—The use of potash, ferrosilicon and limestone in different proportions and modified by the addition of other ingredients is the basis of a patent granted to JULIAN KICH, of Lyndora, Pa. As an example of a mixture which will give the desired result, the inventor uses the following: $\frac{1}{2}$ oz. each of ferrous sulphate, copper sulphate, ferrosilicon, potash, limestone, manganese, phosphorus, and welding compound. The last is composed of boric anhydride, 50 per cent; iron oxide, 15 per cent; soda, 15 per cent, and potash 20 per cent. The mixture first mentioned is added to molten copper in the proportion of 1:3. Modifications of the formula are claimed to give different results. (1,041,297, Oct. 15, 1912.)

Purifying Metals.—For the purification of copper and other metals, particularly for the removal of arsenic, AXEL G. SUNDBERG, of Helsingborg, Sweden, has patented the use of an alkali carbonate. In practice the inventor prefers to add the alkali carbonate, say sodium carbonate, to the bath of

per cent silicon. Such alloys have a limit of elasticity of 12.7 to 19 tons per square inch, tensile strength of 30.5 to 34.9 tons per square inch, extension of 25 to 35 per cent, and a high resistance to the notch bending test. They can be drawn, rolled and hammered at normal temperature, and can be wrought, rolled and pressed when hot. (1,040,027, Oct. 1, 1912.)

The economical manufacture of brass on a large scale is the basis of a patent granted to LAWRENCE ADICKS, of Chrome, N. J. He proposes to melt copper in a reverberatory furnace, such as is used for copper casting, having a capacity of from 20 to 250 tons per twenty-four hours. By means of a suitable ladle, several tons of copper is withdrawn from the furnace and the necessary quantity of zinc added and quickly incorporated. The 2 or 3 tons of brass is then promptly cast from the ladle into ingots. (1,041,940, Oct. 22, 1912.)

Alloy of Titanium and Copper.—ISADOR LATOFF, of Cleveland, Ohio, has patented a method of alloying or compounding titanium and copper without melting the same at the high temperature which would be required for the combination of the primary metals. He uses the oxides of the two metals, TiO_2 and CuO , and incorporates with them a suitable binder, such as oil. The mixture is then molded into any desired shape and introduced into a muffle furnace where it is subject to the action of a reducing atmosphere, such as carbon monoxide, at a temperature of from 600 deg. C. to 800 deg. C. To further insure the reducing atmosphere the molds may be packed in powdered coke. At the temperature mentioned the reduction proceeds, starting with the copper oxide and continuing catalytically with the titanium oxide. The reduction and combination is thus said to be effected at a temperature below the melting point of copper. (1,042,694, Oct. 29, 1912.)

Nickel.

Reduction of Copper-Nickel Ores.—According to patent specifications recently issued to WILHELM BORCHERS, of Aachen, Germany, and HARALD PEDERSEN, of Trondhjem, Norway, the following process may be successfully used in the treatment of copper-nickel ores:

First, the ore is smelted, preferably in the electric furnace, producing a crude matte containing practically all the nickel and copper originally contained in the ore.

Second, the crude matte is roasted in such a manner that both copper and nickel are for the most part converted into sulphates, while the iron forms oxide. This can be done at a temperature of 600 deg. C.

Third, the copper and nickel sulphates are dissolved with acidulated water obtained by passing the roaster gases in a counter stream of water in a percolation tower. This water will contain sulphurous and sulphuric acids.

Fourth, the residue from this leaching operation, consisting principally of iron oxide, but containing also undissolved copper and nickel, is resmelted to form new matte. As the iron oxide favors the oxidation of iron sulphide to oxide, and therefore into slag-forming material, it aids in the formation of a richer matte.

Fifth, by adding calcium or sodium sulphide to the solution obtained from the leaching process, copper is precipitated as a sulphide. After separating this by filtration more calcium or sodium sulphide is added for the precipitation of the nickel as sulphide.

Sixth, the sulphides thus separately obtained are reduced to their respective metals by smelting with suitable fluxes, say limestone and carbon. Calcium sulphide will be obtained as a slag.

Seventh, the calcium sulphide obtained from the sixth operation is converted into sodium sulphide, if desired, by treating with suitable sodium compounds, as sodium sulphate in aqueous solution. The sodium sulphide thus obtained is used as a precipitating agent, in the fifth step, thus giving sodium sulphate solution which may be reduced to sodium sulphide by more calcium sulphide.

None of the steps in the process is novel in itself, but in combination they form a useful and economical method, possessing several advantages over previously proposed methods. In

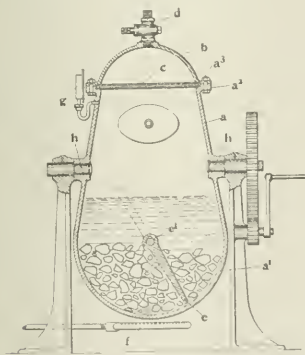


FIG. 3.—ZINC DIGESTER AND FILTER.

molten copper by introducing it in a current of air or gas beneath the surface of the metal. This procedure effectually removes arsenic which may be combined as arseniate of copper, forming sodium arsenate, which rises to the surface as a slag. Copper containing 0.02 per cent arsenic has thus been purified to an arsenic content of 0.002 per cent by blowing in sodium carbonate in quantity less than $\frac{1}{2}$ per cent of the copper. (1,043,371, Nov. 5, 1912.)

Alloys.—A brass possessing valuable qualities has been patented by ALFRED SCHMID, of Zürich, Switzerland. To an alloy of copper and zinc he adds, as an exclusive adjunct, silicon and tin. This produces a metal of high limit of elasticity, toughness, soundness, resistance to the attack of salt water, dilute acids and alkalis, and no undue resistance to cutting tools. The following example is given: 58 lb. copper is melted and superheated; to this is added, while stirring, 40 lb. zinc, 1 lb. tin, and 1 lb. silicon copper, the last containing 30

the smelting tests, slags were made containing less than 0.1 per cent copper and nickel. (1,043,291, Nov. 5, 1912.)

Another process for the same purpose has been patented by HORACE L. WELLS, of New Haven, Conn. He takes a matte of copper and nickel, produced by smelting their ores, and grinds it to a fineness of 60-mesh. The matte is then treated with dilute hydrochloric acid (18 to 25 per cent HCl) at a temperature between 110 deg. Fahr. and 212 deg. Fahr. Most of the nickel will be extracted at the first application of acid, but a second is necessary to complete the extraction. The second acid solution thus obtained may be used on fresh matte. (1,044,316, Nov. 12, 1912.)

Aluminous Abrasive Material.

The production of a new abrasive is patented by AUGUST F. BLOVIN, of Springfield Township, Delaware County, Pa. An example of his material is given as follows: 100 lb. of an aluminium ore containing 78 per cent Al_2O_3 , 12 lb. anhydrous borax, 0.3 lb. chromite, and 4.75 lb. carbon. The mixture is fused in an electric arc furnace and the product crushed and cleaned for abrasive purposes.

Furnaces.

Improved Grondal Furnace.—An improvement in furnaces of the channel type has been patented by NILS V. HANSELL, of Bloomfield, N. J., and assigned to the American Grondal Company, a corporation of New York. In furnaces conforming to the Grondal patent it is customary to load the briquet trucks at some distance from the furnace and feed them to a preheating chamber which is aligned with and forms a continuation of the furnace proper. Where land is valuable this form of furnace requires too much ground, and it is the object of the present invention to provide a furnace requiring less space as to length. Furthermore, in the old type of furnace it is customary to mingle the air blown beneath the trucks to cool them, with the combustion gases after they have left the combustion chamber, and discharge them through a common stack. The combustion gases, however, contain valuable by-products which may be recovered and it is another object of the improved furnace to keep the cooling air and combustion gases separate and thus avoid diluting the latter.

In the new form of furnace the preheating chamber is located adjacent and parallel with the combustion chamber. The latter is graded upward from its entrance to discharge point, so that the newly loaded trucks may return by gravity through the preheating chamber. In the combustion chamber the gases of combustion and the cooling air are kept separate by the usual means of a sand seal on the sides of the furnace. In the entrance vestibule of the combustion chamber, however, it is necessary to provide some means of preventing the air and gas commingling as the truck is fed into the chamber. This is accomplished by means of a shield which follows each car into the chamber and maintains the gas above the trucks separate from the cooling blast beneath it. (1,043,695, Nov. 5, 1912.)

Acid-Proofing the Floors of Battery Rooms.

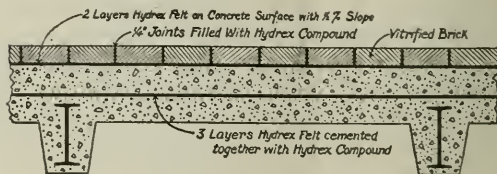
A great deal of attention is being given to the proper method of acid-proofing the floors of battery rooms. It is recognized that nothing can be done to make a concrete floor in itself acid proof. They have been treated with linseed oil thinned down with gasoline, but this method has retarded the action of the acid only temporarily. Acid-proofing preparations have been tried out by one of the leading concrete construction companies of New England but have not proved a uniform success.

The general construction of the floor of a battery room consists of vitrified brick or tile placed on a concrete base. Leading authorities state that the best way to acid-proof such a floor is to place several layers of acid-resisting felt cemented together with an acid-resisting compound between the brick and the concrete. The correct way to do this is first to give the concrete floor a coating of hot compound, then lay in this several thicknesses of felt lapped so that the joints of one

layer break joints with the under layer. Where this procedure is followed it is necessary to use sand for embedding the brick. To protect the walls and any columns in the battery room it is advisable to turn up the felt for a foot or so at the walls or columns, putting three courses of brick against the felt and pouring the compound behind the brick so as to make a tight joint.

The New York Edison Co., which is noted for using large factors of safety in all of its construction, takes extra precautions in acid-proofing floors. They usually specify two distinct courses of several layers of acid-resisting felt cemented together with acid-resisting compound.

In one of the recent specifications of this company for the floors of a transformer and battery station, it is stated that the contractor should spread over the rough concrete floor a layer of 1½-in. cement mortar finished with a smooth surface. As far as acid-proofing is concerned the specifications were as follows:



ACID-PROOF FLOOR CONSTRUCTION.

"Over the cement mortar surface shall be laid three layers of Hydrex felt bonded with Hydrex compound. The cement surface shall first be coated with a continuous and uniform layer of compound, to which it must adhere at every point. Where the felt is turned up on the walls and drain boxes an additional layer of felt and compound running parallel with the intersection of the two plane surfaces shall be applied.

"Over the completed acid-proof course shall be applied a layer of compound.

"Immediately after the acid-proofing is cool or soon enough after to insure against damage, the contractor shall spread a layer of cement mortar about 1½ in. thick over the entire horizontal area of the acid-proof course. This will form, after setting, a working surface on which the rest of the work shall be built."

After the protecting layer of cement mortar has set the specifications call for a course of concrete with a cement mortar finish and on top of this a second layer of acid-proofing consisting of two layers of the acid-resisting felt with three courses of the hot compound. Upon the top of this last acid-proofing course was laid bricks, the joints of which were filled with the hot compound as specified above.

It is interesting to note that the tests of the New York Edison Company have shown that the felt and compound used in this connection are not at all affected by sulphuric acid solutions.

An Improved Metallurgical Microscope.

By Wirt Tassin.

The value of metallographic methods in mill control has been recognized but the usefulness has been limited by the lack of portability in the appliances necessary for their use. This has made it difficult to study the forging, the casting or the bar as a whole, with the result that the metallographic field has to a large extent been limited to the examination of more or less small specimens cut from the piece and which may or may not be representative of the mass.

In June, 1910, I described in the *Iron Age* an illuminator which, attached to the metallographic microscope materially increased its compactness. I now propose to describe a com-

plete metallographic outfit which is light, compact, and portable and may be used either in the mill or in the laboratory and is equally serviceable for the study of the ingot, the forging, the bar, the casting or the small specimen, and which, it is believed, will meet the needs of the metallurgist, the engineer of tests, and the inspector.

The apparatus consists of a camera, microscope and illuminating device, all self-contained (Fig. 1).

The illuminating device consists of a tube which carries the lenses and is mounted in a sleeve provided with a set-screw to lock it in any position. Attached to this sleeve by means of a trunnion is a hanger. This is fastened by a set-screw to an arm which locks into the barrel of the microscope. The rear of the tube carries a shield which is provided with clips to hold the source of light when electricity is used and is slotted to hold a rod and lamp carrier when gas is used.

The source of light may be an acetylene jet or an electric lamp.

When acetylene is used the gas may be obtained from a generator or from a "Prestolite" tank. The lamp is a "Twentieth Century" jet, the support for which is held in a carrier made of two pieces of tubing brazed together at right angles. Each piece is provided with a set-screw so that the jet may be moved up or down, front or back, turned on its axis and set at any desired distance with reference to the plane of the lenses of the condensing train. The apparatus as arranged for illuminating with gas is shown in Fig. 2.

When electricity is used the lamp is carried in a socket fixed in an insulated metal hood. The whole is held in position with reference to the lenses of the condensing train by means of clips fastened to the shield. This arrangement provides for an up and down movement of the light. The apparatus as arranged for illuminating with electricity is shown in Fig. 1.

The current may be obtained from an accumulator or by cutting down the street supply. A convenient lamp bank to reduce street current for use with Mazda miniature lamps is wired as in Fig. 3. The lamp resistance needed with a 110-volt current is:

Two 32-cp and one 16-cp carbon lamps for a 6-volt, 16-cp Mazda miniature lamp.

One 32-cp and one 8-cp carbon lamp for an 8-cp Mazda miniature.

One 32-cp and one 4-cp carbon lamp for a 6-cp, 6-volt tungsten.

One 4-cp and one 16-cp carbon lamp for a 4-cp, 6-volt tungsten miniature lamp.

The lamp recommended is a 6-volt, 16-cp, headlight Mazda, No. 12 G., candelabra base.

The microscope consists of a barrel and draw tube mounted on a handle arm and provided with a coarse and fine adjustment, the latter of the lever type. Attached to the barrel just below the draw tube is a collar which holds a rod controlled by a set-screw. The base of this rod rests on the handle arm. The arrangement prevents the possibility of the coarse adjustment overhauling when the camera is in use. The base of the microscope is the stage. Through the center of this is a 1-in. circular opening which affords free space for the objective when examining large masses below the stage. Leveling screws are provided which permit of the adjustment of the apparatus perpendicular to nearly any surface. The advantages of this are to be seen in Fig. 1 where the microscope is shown mounted on a 6-in. shaft. Provision is made for the use of a detachable mechanical stage when needed.

The camera is attached to the barrel of the microscope by a tube which slips in and out like the draw tube and may be removed with the same ease. The whole moves with and becomes a part of the barrel so that any degree of focusing is possible. The distance between the eye piece and the ground glass is a constant, so that the amplification is a standard for all magnifications. The image is $3\frac{1}{4}$ in. in diameter and is formed on a $3\frac{1}{2} \times 3\frac{1}{2}$ -in. plate. The time required for the exposure is rarely over 10 seconds.

The apparatus packs in a space $12 \times 12 \times 5\frac{1}{2}$ in. A suit case

fitted with compartments to hold the various parts makes a very convenient container. One that measures 12 in. in width by 24 in. long by 6 in. deep, will hold the camera, the microscope and all its accessories, the lamp bank or an acetylene



FIG. 1.—MICROSCOPE CAMERA, ILLUMINATING DEVICE AND LAMP BANK, AS USED WITH ELECTRIC LIGHT AND MOUNTED ON A CINCH SHAFT

generator of the "Neverout" type, four plate holders, four boxes of plates, two tanks, one for developing and one for fixing, together with the chemicals necessary for etching and for developing. The suit case when fitted with a curtain also serves as a dark room.

In examining a large piece such as an engine frame the surface is "spot polished" in as many places as desired. A dam of modeling wax is then built around the spot to hold the etching medium and when the etching has been completed the dam is broken, the surface washed with water and gasoline and the

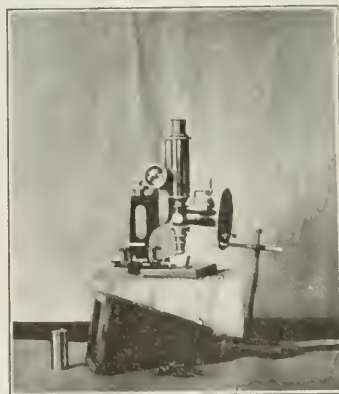


FIG. 2.—MICROSCOPE, CAMERA DETACHED WITH ACETYLENE LIGHT AND MECHANICAL STAGE

microscope adjusted and all is ready for the visual examination and the taking of a photographic record.

To polish, use is made of a portable 14-hp motor with a flexible shaft. As it is almost impossible to hold a wheel steady in the hand to give a surface flat enough for a microscopic examination use is made of a steady rest somewhat like the "Old man" of a drill. This insures a uniform bearing for the grinding wheel and with the proper gradation of the abrasive a mirror polish is obtained without much difficulty. A 130 F. car-

borundum wheel is used for surface grinding and is followed by a set of buffs charged with flour emery paste, washed flour emery paste and gold rouge in the order given.

Arrangements are under way for the manufacture of the apparatus and when completed it may be secured from the Arthur

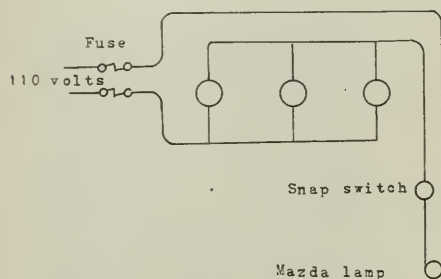


FIG. 3.—WIRING OF LAMP BANK.

H. Thomas Company, of Philadelphia, who are already prepared to furnish the illuminating device which may be attached to any microscope.

Washington, D. C.

Application of the Moore Filter to Rapid Filtration and Purification of Sewage.

In our December issue we mentioned a very successful exhibition given on November 20 of a new sewage purification process at the laboratory of the Moore Filter Company and Chemical Process Company in New York City, at which rep-

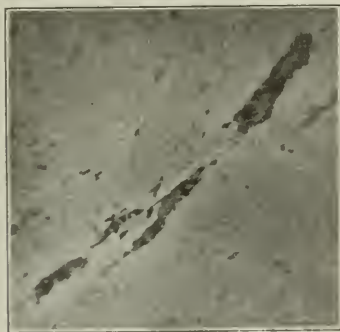


FIG. 4.—MANGANESE SULFIDE AND SLAG X 150 IN A MAN-HOLE COVER—TAKEN WITH THE APPARATUS.

representatives of the water departments of various Eastern cities were present.

It is essentially an application of the well-known Moore vacuum filter to rapid sewage filtration. The chemical features of the process are the invention of Mr. James M. Neil. The rights to the patents are owned by The Chemical Process Company, 170 Broadway, New York City.

From a description of the process, given out by the company, we extract the following information. The system is called the M. S. D. & W. P. or Municipal Sewage Disposal and Water Purification System.

"The operation of the process requires the use of suction pumps, and vacuum filters whose surfaces are permeable to liquid and impermeable to solids. Movable vacuum filters are preferably used so that the separated solids can be removed from the sewage tanks at will.

"The flow of sewage to be treated is directed into a tank or other suitable receptacle, first passing it through a screen or screens in order to remove the floating and other large bodies

therefrom. In this tank suitable precipitants are added to the sewage (greatly in excess of the amount actually required for precipitation of the organic matters) and the whole intimately mixed by agitation, which may be effected by air, steam or water pressure, or by mechanical means, or by a combination of these forces. Such agitation and aeration are maintained continuously during the process, in order to break up and granulate the solids and to commingle the precipitable and fatty matters with the precipitant so as to render them in a condition amenable to collection on the filtering medium and to effect complete precipitation and final separation of the solids from the fluids in the manner hereinafter described.

"By employing an excess of precipitant in the treatment tank its effect is practically instantaneous while owing to the fact that no solids consisting of precipitant or solids in the sewage can escape through the filtering medium the consumption of precipitant is not greater than the amount actually required to effect complete precipitation of the organic and foreign matters contained in the mixture.

"The vacuum filter is placed in the treatment tank at a point preferably remote from the sewage inlet, and when it is submerged and suction begun a continuous flow of the agitated mixture is drawn toward the medium, on the surfaces of which the excess precipitant, precipitates and other solids in suspension in the mixture are deposited, while the liquid is drawn through the film or cake of precipitant and the filtering medium to the pump discharge.

"Complete precipitation and instantaneous separation of the purified fluid from the solids are thus effected.

"The operation may be continued until the deposit of solids becomes so thick on the filtering medium as to unduly interfere with the volume of liquid being drawn through it. This will be indicated by the vacuum gauge. The medium may then be raised from the sewage tank and moved to the dumping point, suction being maintained to hold the solids thereon during its removal, and in order to dry the solids, the medium may be held suspended in the air under suction as long as desired before dumping.

"When suction is stopped the cake of solids falls from the medium, or its removal from it may be assisted by a reverse current of air. After dumping the cake, the medium may be returned to the sewage tank, and the operation repeated indefinitely.

"Instead of raising and moving the medium every time it is loaded, the suction may be momentarily stopped and the solid matters thrown back into the mixture and again subjected to the forces of agitation in the tank—a reverse current of fluid or air may be used to assist in such throwing off.

"Suction being resumed again, the operation of loading is repeated, and when the surfaces of the medium are again sufficiently coated the solids may be again thrown back into the tank and these operations of loading and unloading may be repeated until it becomes desirable to remove some of the accumulated solids from the tank, which may be done in the manner already described.

"There being a continuous flow of sewage into the tank carrying valuable ingredients, this method of operation results in displacing the excess of precipitant from the cake with these ingredients. The cake may thus be finally recovered in a concentrated and highly desirable condition for subsequent treatment whereby the oils, greases, etc., may be liberated by any of the well-known methods and the residue used as a fertilizer.

"Several mediums may be used loading, drying and discharging alternately, so that the process may be carried on continuously. Extra tanks and mediums may be used when necessary to cope with a large or varying supply of sewage, the pump or pumps being always run at a capacity equivalent to the flow of sewage, and suitable precipitants added during the operation in sufficient quantities to deal with the amount and character of the sewage being treated.

"Heretofore in the art of treating sewage it has been found

necessary to use large quantities of hypochlorite or other germicides in order to destroy harmful bacteria. By this process harmful bacteria are almost entirely destroyed, so that in order to insure their complete destruction only a comparatively small amount of germicide need be used. This may be added to the mixture in agitation in the sewage tank, or it may be added to the effluent after it has been separated from the organic and solid matters.

"The results obtained by the above process and which have been effected without the use of a germicide are remarkable. Numerous tests of effluents obtained from samples of sewage originally rich in organic matter and harmful bacteria treated by the usual methods of precipitation and separation were found to be still high in organic matters, and dangerously high in harmful bacteria contents, while comparative tests of effluents obtained from the same sewage treated by this process showed them in every case to be entirely free from organic matters and harmful germs.

"Important features of the process are that it can be operated in open tanks without creating objectionable odors and at a minimum expense for labor."

At the exhibition of the process it was stated that the precipitant used in the process was simply waste material from paper mills, but that lime may be used instead.

PERSONAL.

Mr. William F. Carroll, formerly with the Compania Metalurgica Mexicana, at San Luis Potosi, has gone to Pachuca, Hidalgo, Mexico, for the Blaisdell Coscoticlan Syndicate.

Mr. A. F. Flynt, metallurgist, of Tepic, Mexico, has returned to the southern republic after an extended visit in the United States.

Mr. R. W. French, superintendent for the United States Refining & Reduction Company, at Florence, Colo., will spend the winter in Arizona.

Mr. Robert W. Hunt, consulting engineer, of Chicago, was awarded the John Fritz medal for 1912. This award is given "to commemorate notable scientific and industrial progress," and is made annually.

Dr. Richard L. Moore, physical chemist of the Bureau of Mines, is now stationed at the branch office, 502 Foster Bldg., Denver, Colo. Other members of the staff attached to the same office are Karl L. Kithil, mining technologist; J. C. Roberts, mining engineer in charge of the rescue car, and John A. Davis, mining engineer.

Mr. J. Wilson Nevius, representing the Los Angeles Chamber of Mines and Oil, has been making a study of the bag-house system of fume control at the smelter of the Mammoth Copper Mining Company, Kennett, Cal.

Dr. Jos. W. Richards, of Lehigh University, sailed with Mrs. Richards on December 21 for Jamaica, for a few weeks' vacation.

Mr. B. Viola, formerly managing and chief engineer of Charles Pfizer & Co., has opened a consulting engineer's office at 140 Liberty Street, New York City, for the installation and equipment of chemical and allied industrial works and power plants.

Mr. David White has been appointed chief geologist of the U. S. Geological Survey, vice Waldemar Lindgren, resigned. Mr. White has been in the Survey since 1886, and has had charge of several departments of the work. Dr. F. L. Ransome will succeed to the duties of Mr. Lindgren, in the economic geology of metalliferous deposits. Mr. Lindgren will be retained by the Survey for special work in a study of the Hone-stake mine, South Dakota, and at other places in Arizona.

Mr. C. E. Whittlesey, treasurer of the McGraw Publishing Company, celebrated his seventieth birthday on December 14, 1912. On this occasion he was entertained at luncheon at the Engineers' Club by his fellow directors of the McGraw Publishing Company and was presented with a silver salver as "a tribute of esteem and affection." Mr. Whittlesey, be-

sides being the treasurer of the McGraw Publishing Company, has been the treasurer of the Electrochemical Publishing Company, the former publisher of METALLURGICAL AND CHEMICAL ENGINEERING, from its early days. All the members of the staff of this journal are indebted to Mr. Whittlesey for innumerable kind words and acts of advice and encouragement, and join his many other friends in extending to him their most cordial wishes.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

629,685, July 25, 1899, Carl Hoepfner, of Berlin, Germany.

Relates to a process for electrodepositing zinc or other metals in solution by using an insoluble anode or an anode of a metal electronegative to that to be deposited, preferably lead, copper or mercury or the compounds of such metals, such as the sulfid or an argentiferous sulfid of lead, the silver, etc., remaining undissolved as a sludge. These compounds are either cast into bars, etc., or may be placed in a wire basket holder, etc. The anodes should preferably be kept in motion. The cathodes, which are also kept in motion, consist of carbon or of the metal to be deposited. The apparatus consists of a series of U-shaped frames bolted together and separated by diaphragms, forming separate cells, alternate cells constituting anode cells, and the intermediate ones the cathode cells. Communication between like cells is established by passages in the walls of the frames. The anode electrolyte is an acid in which the anode metal is soluble or a salt containing such acid as its radicle. The cathode electrolyte is any solution containing the metal to be deposited, such as zinc, iron, nickel, copper, etc. Both electrolytes are preferably heated and kept in constant circulation through the cell. The lead or other metal dissolved from the anode is precipitated outside of the cell as an insoluble compound, from which the metal may be reduced by smelting. The solution of the metal as anode will generate an electromotive force and to that extent reduce the electromotive force ordinarily applied to the cell.

629,686, July 25, 1899, Carl Hoepfner, of Frankfurt-on-the-Main, Germany.

Relates to the extraction of zinc or zinc and other metals from ores of the Brokenhill type and also of the Leadville Colo., type. Refers to his patent No. 664,269. The process consists in reacting upon a crude zinciferous sulfid mineral containing also other metals, as lead and silver, with a suitable chlorid solution, as cupric or perchlorid of iron, extracting a portion of the lead, removing the solution and roasting the undissolved residue; reacting upon this residue with hydrochloric acid as in the presence of water or a chlorid solution from a prior treatment, dissolving first the lead and silver and finally obtaining a solution of zinc chlorid, which is electrolyzed first with a low electromotive force to remove the electronegative metals present, gradually raising the electromotive force to successively precipitate the intermediate metals and finally precipitating the zinc; the electrolyte may now be used in the hydrochloric acid gas extraction.

631,040, August 15, 1899, John E. Greenawalt, of Denver, Colo., assignor of one-half to William Robinson, of same place.

Relates to the extraction of gold or silver from their ores. The ore is crushed to 20 to 30 mesh and subjected to a chloridizing roasting, after which it is placed in a vat having a bottom made of slats covered with a perforated board; above the board is placed a gravel bed, the coarser stones at the bottom and smaller ones at the top. Upon the small stones is placed the ore. A leaching solution consisting of a nearly saturated solution of sodium chlorid with some bromine is

electrolyzed in the anode compartment of a cell, producing free chlorine and hypochlorous acid and free bromine. This solution is used to leach the ore and is then passed through an electrodepositing cell using lead cathodes with a current density of about 0.07 amp per square foot of cathode surface.

633,544, Sept. 19, 1899, Harry S. Badger, of Denver, Colo., assignor of one-half to Charles W. Caryl, of same place.

Relates to apparatus for extracting and amalgamating precious metals. A suitable tank is provided with a central hollow vertical shaft having hollow horizontal cross-arms at the bottom thereof, the cross-arms carrying hollow stirrers with openings at the bottom. The vertical shaft is arranged to be rotated, is connected to a supply of compressed air and is connected so that the horizontal cross-arm is the anode. On the bottom of the tank is a copper plate flooded with mercury, connected as cathode. The pulverized ore is added to a solution of cyanide of potassium and such other reagents as facilitate the dissolving of the precious metals. The rotating stirrers and compressed air keep the ore in a state of suspension and leave the mercury free for the amalgamation and electrodeposition of precious metal.

639,579, Dec. 19, 1899, John Jones, of East Melbourne, Victoria.

Relates to the recovery of zinc from its ores. The ore is first roasted to sulfate, sulfite or oxid, leached by water or acid and the solution obtained freed from iron and other impurities, which is effected by flowing the solution down through absorbing towers through which is passed upward a stream of ammonia-containing producer gas, which operation is regulated so that the zinc remains in solution while the iron, etc., is precipitated. The purified solution is then acidified with sulfuric acid in a separate vessel and again passed through an absorbing tower through which is passing additional ammonia-containing producer gas. The purified producer gas may be used for fuel, etc. The zinc-sulfate ammonia-sulfate solution is passed into the anode compartment of an electrolytic cell, while an acidified zinc-sulfate ammonium-sulfate solution passes into the cathode compartment. In order to prevent the catholyte from becoming neutral or alkaline it is constantly acidified either by free acid or by circulating the anolyte through the cathode compartment. Any manganese salts in the solution are precipitated in the anode compartment by a suitable reducing agent.

639,766, Dec. 26, 1899, Lewis E. Porter, of Los Angeles, Cal.

Relates to rotary amalgamating barrels for extracting precious metal from base ores. The rotary barrel has an amalgamating cathode lining, either of removable plate, or by first copper-plating the inside, and then amalgamating. Adjacent the cathode is a diaphragm lining of burlap, etc., secured between a double layer of wooden slats. The anode consists of a series of carbon plates supported upon spiders, and are concentric within and rotate with the barrel. An adjustable valve, adapted to be automatically opened as the barrel rotates, affords an exit for any excess of gas pressure generated during the operation. The pulp mixed with the desired chemicals is fed into and removed from the space within the diaphragm through a removable closure in the end of the barrel; during the operation, compressed air is forced into the barrel through a hollow shaft, the air assisting in forcing the liquid through the diaphragm, and of acting as an oxidizing agent. Refers to a co-pending application filed Aug. 18, 1896.

640,717, Jan. 2, 1900, Charles P. Tatrow and George Delius, of Seattle, Wash., assignors to Harry S. Sharpe, of same place.

Relates to an apparatus for extracting precious metals from ores. The apparatus comprises a tank having a rotary drum supported on a shaft within the tank; a series of fixed carbon bars serving as an anode around the drum, a rotary stirrer within the tank below the anode, and a conical bottom containing mercury to amalgamate any large pieces of gold, etc., that may be in the ore. The pulp, mixed with a suitable solvent, is fed into the tank, the drum rotated and current turned on. The rotating stirrer keeps the ore in suspension, while the dissolved metals are electrodeposited upon the drum, from which

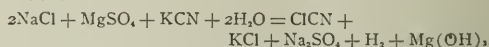
they are removed by a scraper pressing against the upper surface. After the operation is completed, the pulp and mercury are removed, the mercury is then retorted or otherwise treated to remove any precious metal. Refers to a co-pending application (patent 640,718) for the process.

640,718, Jan. 2, 1900, Charles P. Tatrow and George Delius, of Seattle, Wash., assignors to Harry S. Sharpe, of same place.

Relates to a process for extracting precious metals from ores. Refers to a co-pending application (patent 640,717) for the apparatus used. The ores are pulverized to pass an 80-mesh sieve, the pulp mixed with a bath comprising an aqueous solution of common salt, and such reagents, as lime, to precipitate the baser metals, such as iron, while leaving the precious metals in the solution. With some ores, an acid bath is first used to dissolve a particular base metal, which is then precipitated by an alkali. A small per cent of bromine may be added to facilitate the solution of the gold. The precious metal is electrolyzed and recovered as described under patent 640,717.

641,571, Jan. 16, 1900, Wilhelm Witter, of Hamburg, Germany.

Relates to the production of a solution of cyanogen halide, such as chloride or bromide, for the extraction of precious metals from their ores. A solution containing an alkali halide, such as chloride or bromide, an alkali cyanide, and a magnesium salt, is electrolyzed in a cell without a diaphragm and with inert electrodes, such as carbon. The solution produced contains cyanogen halide. The following equation expresses the reaction:



The magnesium salt and alkali cyanide are used in excess of the molecular proportions. The reaction may be carried out in the leaching vat direct in the presence of pulverized ore.

643,096, Feb. 6, 1900, Samuel B. Christy, Berkeley, Cal.

Relates to the electrolytic precipitation of gold and silver from the large volumes of dilute solutions of alkaline cyanides resulting from the cyanide extraction of gold and silver ores, etc. Uses a tank containing a large number of removable electrodes; the tops of the anodes remain about an inch below the solution level, and divide the tank into a number of separate compartments; between the anodes are the cathodes which project above the solution level, but leave a space of about an inch between them and the bottom of the tank. The liquid is electrolyzed while flowing through the tank, between the electrodes, above the anodes and below the cathodes; the deposit upon a number of cathodes being subsequently redissolved as an anode in a smaller cell, and deposited upon a single smaller cathode, thereby reducing the quantity of base metal to be separated from the gold.

649,151, May 8, 1900, William Wright, New York, N. Y., assignor by mesne assignments, of eleven-twentieths to William Bouldin, trustee, of Orange, N. J.

Relates to an apparatus for extracting metals from refractory ores, and comprises a leaching tank having a perforated false bottom, a filtering medium over the perforated bottom, a cathode within the space below the false bottom, a plurality of stationary pins arranged in concentric circles projecting upwardly from the false bottom, a rotatable shaft supporting radial cross-arms from which carbon anodes are suspended and arranged in concentric circles intermeshing the upwardly projecting pins. In operation, about 2000 lb. of ore are mixed with 1200 lb. of water containing from 20 to 40 lb. of common salt. The rotation of the anodes keeps the mass in constant agitation, the current liberating chlorine and decomposing the ore, the metals dissolved being deposited upon the cathode beneath the false bottom.

650,646, May 29, 1900, Frederic H. Long, Chicago, Ill., assignor to Ross J. Beatty, trustee, Muncie, Ind.

Relates to an apparatus for electrolytically treating ores, and comprises mixing, reducing, and filtering tanks; within the former the pulp is thoroughly mixed to a sludge with the electrolyte; the mixture is then transferred to the reducing tank, which consists of a circular vertical tank containing an upper

anode compartment and a lower cathode compartment, separated by a cone-shaped filter; centrally within the tank and communicating with the anode compartment is an upright penstock containing a reciprocating piston, which causes a circulation of the sludge through the anode compartment and penstock. During this circulation the ore passes the anode and is subjected to the action of the chlorine there liberated, whereby the metals are dissolved and then electrolytically deposited upon the cathode, which may be mercury. The base metals in the anode liquors may be precipitated as hydroxides by lime. Any gases liberated are fed into the mixer, where they assist in dissolving the metallic content of the ores. After the treatment, the sludge is transferred to the filter in which the liquid is separated for re-use or recovery of any remaining metallic salts.

BOOK REVIEWS.

The Principles of Applied Electro-Chemistry. By A. J. Allman, D.Sc. $5\frac{1}{2} \times 8\frac{1}{2}$ inches ($14 \times 21\frac{1}{2}$ cm.), 547 pages, 136 illustrations; price \$5.00. New York: Longmans, Green & Co. London: Edward Arnold.

A volume designed for students and technical men. Assuming an elementary knowledge of chemistry and electricity, Part I, 192 pages, treats of the fundamental phenomena and theory of electrochemistry, particularly of aqueous solutions. Part II treats of the various technical processes, from the standpoint of showing the close connection between theory and practice, treating the material critically, and excluding mere "paper" processes.

The author has succeeded in producing a valuable book; comprehensive, clear, fair and making full use of the copious electrochemical literature to date. To give a catalog of the subjects treated is unnecessary; it covers practically the entire field of inorganic electrochemistry. Organic electrochemistry is not included.

We commend the book as valuable to students and technical men. It is the equivalent of Haber's "*Grundriss der technischen Elektrochemie*," in English, revised to date, and improved upon.

* * *

Cyanide Practice. By H. W. MacFarren. 8vo ($15\frac{1}{2} \times 23\frac{1}{2}$ cm.), 291 pages, 32 illustrations; price \$3.00 net. New York: McGraw-Hill Book Company.

The basic idea of this book is not to treat exhaustively every detail of the cyanide process, but to give a bird's-eye view of the whole field, explaining clearly the principles involved in each step and giving sufficient detailed information to illustrate the principles. Description of variations of details at different plants is not attempted. The object thus in view is to bring the student or practitioner to the point of understanding thoroughly the process as a whole, and of being prepared to study more in *extenso* any branch of the process. To assist in the latter a very complete bibliography (of English literature only), covering 55 pages and divided into 33 heads, gives abundant information. It is a carefully prepared book and accomplishes its avowed object.

* * *

Per-Acids and Their Salts. By T. Slater Price, D.Sc. Six by 9 in. ($15 \times 22\frac{1}{2}$ cm.), 123 pages; price \$1.00 net. New York and London: Longmans, Green & Company.

This is the second of the Monographs on Inorganic and Physical Chemistry, which Dr. A. Findlay is doing the great service of editing and publishing. Modern scientific literature really needs only two classes of books: brief syllabi of an entire science, setting forth clearly principles and describing their application in the fewest possible words and comprehensive monographs going into great detail on the various branches of the science. Such must be the program of future scientific literature; the day of the vast encyclopedia of science which covers in one work all science or all of a science in all its details is rapidly passing.

The present book is a good illustration; it gives more precise, detailed, satisfactory information about per-acids and their salts than can be found anywhere else. Just as we go to an expert oculist or dentist for advice about eyes or teeth so must we turn to just such books as this for the best information about a special subject in science.

Dr. Price has done his work well; one has only to read a little in it, anywhere, to get that feeling of satisfaction which comes from the realization that the author knows all about all he is discussing. "May his tribe increase."

* * *

Theoretical and Physical Chemistry. By S. Lawrence Bigelow, Professor of General and Physical Chemistry, University of Michigan. Octavo (14×21.5 cm.), 544 pages, 80 illustrations; price \$3.00. New York: The Century Company.

Intended to lead third-year students in college or university to "a critical consideration of these half-learned, three-quarters forgotten, fundamental principles"; and recommended as following first courses in general chemistry and qualitative analysis. The author very concisely divides the book into four parts: "The first section (three chapters) aims to show the value of philosophy in science. In the second section (seven chapters) the ever-present question is: What are the ultimate constituents? In the third section (nine chapters) we are, primarily studying the properties of substances as such. In the last section (eleven chapters) attention is centered upon the processes by which substances become what they are."

The book is written with great care, clearness and insight. It is just the kind of text-book that would engage and hold the student's attention. To one not a student it appears in places to lack candor; that is, not to get down close enough to simple reasoning from observed facts, and at others to state conclusions on insufficient evidence. In all teaching conclusions should be given their exact values, and never overdrawn; thus, only are students taught exact thinking and reasoning. However, this book is above the average in these respects.

* * *

Church's Laboratory Guide. Revised and largely re-written by Edward Kinch, formerly Professor of Chemistry, Imperial College of Agriculture, Tokio. Ninth edition, 12mo (12×19 cm.), 368 pages, 44 illustrations; price \$2.50 net. New York: D. Van Nostrand Company.

This is a complete course in chemical laboratory practice for agricultural students. Part I comprises 39 lessons in elementary chemical manipulation and experimenting, chosen with discrimination and excellently presented. Part II is a short course in qualitative analysis, also well compiled and written, but necessarily very brief. Part III introduces the student to quantitative analysis, and after a short introduction takes up at some length the analysis of manures, soils, waters and foods. For the student in an agricultural college who does not intend to become an agricultural chemist, the book appears a most excellent laboratory manual; the agricultural chemist, however, should lay broader foundations in both qualitative and quantitative analysis than are to be gotten from this book.

* * *

General and Industrial Inorganic Chemistry. By Dr. Ettore Molinari. Translated from third Italian edition by Dr. E. Feilmann. Large octavo ($15 \times 25\frac{1}{2}$ cm.), 704 pages, 283 illustrations; price \$6.00 net. Philadelphia: P. Blakiston's Son & Company.

The author is professor of industrial chemistry in Milan, and has written this treatise for the use of students in order to bring before them the applications of chemical principles in industry simultaneously with the comprehensive study of the principles themselves. He assumes some knowledge of the elementary facts of chemistry and physics, but gives in the first 125 pages his own summary of chemical and physical principles. This is commendable, because it forms a connecting link with the students' preparatory studies, and further teaches him the author's point of view of these elementary

laws. Then the author proceeds systematically with the preparation in the laboratory and the manufacture on a large scale of the various non-metallic and metallic elements and compounds.

A distinguishing feature of the treatment is the setting forth, as clearly as possible, of the commercial side, including prices, statistics and commercial considerations. This is intended to give the student the business side of industrial chemistry, so that when entering the industry he may not be ignorant of the elementary financial foundations of manufacturing chemistry as a business and of the values and prices of the materials he uses and produces. It will make an excellent up-to-date text-book for students in chemical engineering.

* * *

Lead Poisoning and Lead Absorption. By Thomas H. Legge, M.D., and Kenneth W. Goadby. 8vo (15x21½ cm.), 308 pages, 15 illustrations; price, \$3.50 net. New York: Longmans, Green & Co. London: Edward Arnold.

The senior author is government medical inspector of factories in England; the junior author is surgeon to smelting and white-lead factories in London. The work covers the symptoms, pathology and prevention of lead poisoning, with special reference to its industrial origin and the processes involving this risk. The inhalation of lead dust is cited as the main cause, and exhaust fans, hoods or vacuum cleaners are named as the simple means by which the dust can be removed and the worker's health assured. The work is written in a highly scientific and exact style, using accurate medical phraseology. This makes it very satisfactory as a medical monograph, but limits its usefulness as respects the general reader. Every manager of works manufacturing lead products should be provided with the book, study its statements and follow its recommendations.

* * *

La Cementazione dell'Acciaio. By Dott. Federico Giolitti, Professor of Metallurgy in the Royal Polytechnicum of Turin. Octavo, 16x25 cm., 506 pages, 155 illustrations. Turin: Unione Tipografico-Editrice Torinese.

This brilliant Italian savant left the University of Rome two years ago to build and assume charge of the finest metallurgical laboratory in Italy, at the Polytechnic of Turin. The above work may be regarded as the first fruits of his labors and those of his students in this new chair.

There exists in no other literature so complete and masterly a study of the cementation process of manufacturing steel. It bears throughout the personal imprint of the enthusiastic investigator and able writer. Visiting his laboratory in Rome three years ago, we were wonderfully impressed by his enormous activity and the wonderful program of matters to be studied out when he should get into his new laboratory at Turin. Fortunate, indeed, is Italy in having such a professor and doubly fortunate those Italian students who have the inspiration of working under him.

As for the program of the book, 250 pages, divided into five chapters, deal with research into the chemistry of cementation. The contents of these chapters are: Early researches; up to the end of the XIXth century; the first seven years of the XXth century; the last five years; actual state of our present knowledge. In this section the author relates at length his own exhaustive investigations on the chemistry of the process, much of the information being here published for the first time.

In the second half of the book the industrial applications of the cementation process are discussed in a similarly thorough manner. The five chapters deal with: Total transformation into hard steel, partial transformation into hard steel (case-hardening), sub-divided into treatment by solids, liquids, gases, mixed agents and special agents, thermal treatment of cemented products; methods of control of the cementation; contents of patents on cementation. In this part the furnaces are illustrated by photographs and working drawings, and the author gives details of practice in various countries—American practice being well described.

We congratulate the indefatigable and genial author and also

his country on such a fine contribution to scientific literature and progress. Surely this is a victory of peace.

* * *

Coal: Its Composition, Analysis, Utilization and Valuation.

By E. E. Sommermeier, Professor of Metallurgy, Ohio State University. Octavo, 15½ x 23 cm., 175 pages, 8 illustrations. Price \$2.00. New York: McGraw-Hill Book Company.

The book has been prepared for the mechanical and power-plant engineer, the chemical engineer and chemist, and business men and operators buying and selling coal. The author has met this rather difficult program in a satisfactory manner.

The chemical part has particularly attracted our attention, by its very clear exposition of chemical principles and details involved in the analysis and combustion of coal, including a number of new and original points of view and calculations. There are full detailed directions for sampling, getting heat-values in calorimeters, analyzing flue-gases and interpreting the experimental results. The chapter on purchase specifications is short and condensed, but practical; it might have been amplified. About 300 analyses of coals, with calorimetric values, form a useful appendix. It is a well-written, useful and timely book.

* * *

Engineering and Metallurgical Books, 1907-1911. By R. A. Peddie. 206 pages, 4¾x7¾; \$1.50 net. New York: D. Van Nostrand Company.

A bibliography or catalog of the engineering and metallurgical books published in the years 1907 to 1911 inclusive. The books are classified under subjects, and a brief characterization corresponding to a sub-title is generally given. The names of both the American and English publishers are included and the prices in both British and American currency.

Apparently the work has been done with thoroughness and care, and will make a useful reference catalog for engineers and librarians.

* * *

Centenary Celebration of the First Commercial Gas Company. Lectures delivered at the Franklin Institute, Philadelphia, April 18 and 19, 1912. Octavo, illustrated; 174 pages. New York: The American Gas Institute, New York.

This book contains five carefully prepared lectures: By-Products in Gas Manufacture, by Prof. Charles E. Munroe; Commercial and Financial Aspects of the Gas Industry, by Hon. George B. Cortelyou; The Technique of Gas Manufacture, by Alfred E. Forstall, M.E.; Gas as an Illuminant, by Van Rensselaer Lansingh; The Use of Gas for Heat and Power and the Testing of Gas, by E. B. Rosa, Ph.D. Besides these first-class lectures there is a chronology of the use of gas, covering several pages, and description of an interesting Loan Exhibition.

The book is an appropriate and valuable souvenir of a dignified and memorable celebration.

* * *

Hygiene for the Worker. By Wm. H. Tolman, Ph.D., and Adelaide W. Guthrie. 12mo (12x18.5 cm.), 231 pages, numerous illustrations; price 50 cents. New York: American Book Company.

This inexpensive book is one of the Hygiene Series, edited by C. Ward Crampton. The authors are the Director of the American Museum of Safety and an assistant in the research department of that museum. It is intended for students in vocational, industrial and manual training schools and particularly for night schools.

An inspection of the nineteen chapters shows them to be carefully, accurately and attractively written, full of good common sense as well as of scientifically accurate information.

It is a good book to give to any worker, but particularly to the young; employers of labor may also find profit in reading it.

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Chemists as Captains of Industry

Elsewhere in this issue we print a report of the presentation of the seventh impression of the Perkin medal to Dr. James Gayley at a joint meeting of the national chemical societies of this country, held under the auspices of the Society of Chemical Industry on January 29.

The award of the Perkin medal to one who has highly distinguished himself in applied chemistry, has now become a delightful annual event in chemical society and club life. The Perkin medal has become an inspiration to all who have at heart the future of American chemistry, and the Perkin medal has become an education for the general public as to the scope and achievements of chemistry.

Let other countries be proud of their warriors, poets, and musicians, the United States is proud of its great captains of industry, and the Perkin medal has become a means of proclaiming to all the world how many of the great captains of industry have been recruited from chemistry. Here is a list of those who have received so far the Perkin medal: Sir William Perkin, the inventor of the dyestuff mauve and founder of the coal-tar industry; Dr. J. B. Francis Herreshoff, to whom the general chemical industry of this country and electrolytic copper refining owes so much; Dr. Arno Behr, of the great American industry of corn products; Dr. E. G. Acheson, the founder of the carborundum and artificial graphite industries; Dr. Chas. M. Hall, the founder of the American aluminium industry; Mr. Herman Frasch, of oil refining and sulphur mining fame, and now, Dr. James Gayley, the inventor of the dry-blast process, who began as a simple steel chemist and grew to be the first vice-president of the U. S. Steel Corporation.

This is certainly a brilliant list of names and a brilliant list of achievements. And yet, lest we forget, that is only the beginning.

Empirical vs. Exact Methods

It has been remarked that the oldest mining districts are the most conservative, where progress and change are accomplished only by laborious effort on the part of outsiders. In such places it is not uncommon to find metallurgical operations based on empirical knowledge derived from limited observation. The natives rely with satisfaction on tradition and custom, and rest contentedly on the use of inexact and unscientific methods without realizing their limitations. It is true that some empirical methods have a basis in fact and, in the hands of skilled operators and observers, serve a useful purpose. The control of the bessemerizing and poling of copper, and the estimation of the value of some gold-ores, are notable examples of arbitrary methods which, to the practiced eye, yield reliable results.

The vanning shovel is a device in the same class with the gold-pan. For years the practice of vanning has been

firmly entrenched in Cornwall as a method for determining the value of tin-ore and concentrate. The skill of the Cornishman with this device is proverbial. So great has been the reliance placed in the results obtained by the vanning shovel that the efficiency of milling operations and the buying and selling of concentrates are confidently based thereon, to the general exclusion of chemical methods of analysis.

There has been a growing suspicion that the vanning assay for tin is unreliable, together with a lack of confidence in the supposed efficiency of Cornish tin-milling when based on such assays. More recently this suspicion has been confirmed, and a rude shock given to the advocates of Cornish practice by one who has had an opportunity of studying the district and its methods. Fortified by the more exact methods of chemical analysis, this investigator has come to the conclusion that the vanning experts have been deceiving themselves as well as others; and that, so far from making a mill recovery of 90 per cent, the actual tin extraction throughout Cornwall to-day is between 50 per cent and 60 per cent. In competitive tests between the vanning assay and chemical analysis, the results of the former ranged from 22 per cent to 95 per cent of the latter, being more exact on high-grade coarse cassiterite.

In the light of these figures it seems almost incredible that a great industry should be controlled by such inexact methods; and more so that the custom should prevail in the face of better knowledge. The continuation of the practice in spite of the protests of those who are informed as to its deficiencies, shows how firmly rooted it is. Progressive metallurgists realize that the continued use of the vanning shovel is largely responsible for the backward condition of tin metallurgy in Cornwall, and they look for improvement in the latter as fast as accurate chemical analysis supersedes the arbitrary and inexact vanning assay.

Surface Quality in Steel

We have long been accustomed to think of quality in steel almost wholly in terms of elastic limit, yield point, ductility, hardness, resistance to fatigue, etc., while more lately we have had to consider, for certain classes of steel, magnetic permeability, hysteresis, conductivity, etc., and for certain other classes of steel the question of corrodibility has been emphasized by some producers.

A somewhat new factor has been looming large of late, on account of the extremely rapid increase in the use of steel sheets for metal furniture and for various automobile parts, particularly bodies and hoods. These sheets are required to take a high degree of finish. Various finishes are furnished by the sheet mills, leaving the surface in condition to take such coating as the fabricator desires to apply. It has been found that these so-called "finishes" given by the mill are not to be regarded purely as treatment superficial or entirely subsequent to the manufacture of the steel sheet. The character of the steel itself is found to have much to do with success in applying the "finishes." There is practically no literature on the subject.

The mills which have developed these finishes most successfully have not taken the public into their confidence. They have been learning rapidly, and the information has

not had time to percolate, but it has become perfectly clear that to our researches as to how to make steel strong, ductile or hard as the case may be, qualities which are purely of cross section, there must be added study as to how to make steel such that it will present, or yield itself to being given, a particular character of surface. This is practically a new field in the adaptation of steel to commercial use. The first man who modified the character of steel in order to contribute to surface finish has not come forward to claim the honor, preferring to endeavor to maintain a commercial secret, but we have reason to believe that what he did was done only very recently.

Technical Education

University extension courses of study have developed a successful means of disseminating knowledge on many subjects. Short courses in agriculture have proved valuable for farmers, being given during the winter months when work on the farm is not pressing. Correspondence courses in a variety of subjects have served a useful end. Reasoning by analogy, it has been suggested that "practical" short courses in mining and milling, given by professional schools of mines, might be beneficial in raising the standard of work in those industries. The proposal raises a question on which there is a wide difference of opinion.

Recently the University of Tennessee has announced extension courses, designed especially for workers in mining and quarrying. Physics, chemistry, mathematics, geology, mining, metallurgy and mechanical engineering are the subjects touched upon during a short course of six weeks duration, supplemented by correspondence work. The object of the course is to afford an opportunity to the man "on the firing line" to improve his work by the understanding of scientific principles. Another school, at Platteville, Wisconsin, has for its aim the training of young men in the practical details of mine and mill work, without pretending to turn out finished engineers. This training school produces technicians, but not technologists.

The principal argument in favor of the short course is that it renders a distinct service and ultimately raises the standard of work. There are, perhaps, stronger arguments against the short course in professional mining schools. Alumni almost a unit against the plan, believing that it would detract from the reputation of the school, and affect their prestige as graduates. The difficulty arises of differentiating for the public between engineering graduates and short-course students. Each might, with varying degrees of propriety, claim attendance at a well-known school; but without further explanation, the short-course student would give the inference that he was on a par with the engineering graduate. Even if such a result were not intentional, it is a possibility.

Except for instances in which local conditions seem to favor the short course, it would seem better to maintain the integrity of the school of technology, and leave the other field for the strictly training school.

The Digest of U. S. Electrochemical Patents

When this journal was started somewhat over ten years ago with the title "Electrochemical Industry," analyses of current electrochemical U. S. Patents naturally suggested

themselves as valuable, instructive, and interesting news matter. At the same time it was obvious that for anybody who wants to look up the patents on a certain subject, it would be very important to be able to find also the older U. S. patents which had been issued before this journal was started. This consideration suggested a new department—one decidedly unique in technical journalism—the "Digest of Electrochemical U. S. Patents prior to 1903, arranged according to subject matters and in chronological order."

For the preparation of this department this journal had the good fortune to secure the services of one most competent to do the work—Dr. Eugene A. Byrnes. And the Digest, prepared by Dr. Byrnes, has been published regularly ever since. It has been used extensively by those who understand its scope and object. But the fact that the subscription list of this journal has increased very rapidly in recent years far beyond the limits of the original constituency, suggests that to many of our more recent friends the idea of the Digest may not be sufficiently familiar and to them a concise summary of the immense amount of work of love stored up in this department in the past ten years should be welcome. This also gives us an opportunity to say a word on electrochemical U. S. patents in general.

When Dr. Byrnes was appointed principal examiner of Division 3, "Metallurgy," etc., in April, 1891, the comparatively few U. S. patents theretofore granted relating to electrochemical subjects were scattered through several different divisions and subclasses of the Patent Office, especially through Division 3 with classes on metallurgy, coating with metals, and distillation; Division 6 with classes on bleaching and dyeing, chemicals, hides and leather, and medicines; Division 25 with a class on paper making (bleaching); Division 26 with subclasses on batteries for the generation of electricity.

As the art of electrochemistry was then becoming quite active, it seemed desirable to collect these scattered patents into a single class, and Dr. Byrnes suggested to the Commissioner of Patents that such class be established under the title "Electrochemistry." Some argument was necessary to convince the Commissioner as to the need for this class, and the propriety of such title. However, in view of favorable recommendations of other examiners, and the fact that the word "electrochemistry" had been used to a slight extent by others, having been adopted by Becquerel as the title of his classic book, while the first edition of Borchers' "Elektro-Metallurgie" had just appeared, the Commissioner ordered the establishment of the new class, in May, 1892. The 390 patents which Dr. Byrnes had collected for this class were distributed into 27 subclasses, being, with two exceptions, those of the present classification.

On December 27, 1892, subclass 28, Diaphragms, was established, and the patents on batteries were received from Division 26, where they existed as subclass 6, "Batteries, Galvanic" and 72, Batteries, Secondary." On November 23, 1894, subclass 31, "Synthesis" and 32, "Synthesis, Ozone" were established, the old subclass 23, "Ozone," being abolished. The subclass of "Galvanic Batteries" was split up, prior to January 1, 1895, into subclasses 33-53, first published in the Official Gazette of that date. On January 31, 1897, new battery subclasses, 53-56, were established; also the following: 57—Refining Metals; 58—Chemicals. Alka-

lies and chlorine; 59—Chemicals, Bleaching and disinfecting liquors; 60—Chemicals, Chlorates; 61—Chemicals, Dyes and pigments; 62—Synthesis, carbides; 63—Reduction; 64—Electric furnaces; the old subclass 2, "Bleaching" being abolished. There has been no subsequent change in the classification of these patents.

At the date when Dr. Byrnes began to analyze these patents for the Digest in this journal, July 1, 1902, ten years after the class had been established, the number of electrochemical patents, aside from those on batteries, had increased from 390 to 1486; and on December 10, 1912, the number had increased to 3498.

The Digest, so far published, has covered the more important patents prior to 1903 according to subject matters in chronological order as follows:

Electrolysis of sodium chloride with mercury anodes (vol. I, p. 42, 78).

Diaphragm cells for the production of caustic alkali (vol. I, p. 114, 152, 196, 232, 268, 300, 340).

Bleaching and disinfecting liquids (vol. I, p. 368, 401, 433).

White lead by electrolytic methods (vol. I, p. 474, 518).

Dyes and Pigments (vol. I, p. 518, 555).

Chemicals (vol. I, p. 598; vol. II, p. 43, 80, 123).

Production of aluminum from molten electrolytes (vol. II, p. 165, 210, 253).

Production of sodium from molten electrolytes (vol. II, p. 296, 333, 384, 434).

Molten electrolytes, Miscellaneous (vol. II, p. 434, 474, 517; vol. III, p. 48, 88).

Electric Smelting and Reduction Furnaces (vol. III, p. 128, 164, 208, 246, 284, 322, 362, 406, 448).

Treatment of carbon and the production of graphite (vol. III, p. 482; vol. IV, p. 42).

Production of nitrogen compounds (vol. IV, p. 80, 120).

Production of ozone and miscellaneous gas reactions (vol. IV, p. 162, 204, 252, 290, 338, 423, 468, 514).

Carborundum (vol. V, p. 36, 70, 108).

Calcium carbide (vol. V, p. 154, 198, 248, 292, 336, 382, 430, 480, 516; vol. VI, p. 39, 86, 132, 170, 218, 264).

Electric furnaces (vol. VI, p. 306, 345, 389, 434, 476, 523; vol. VII, p. 49, 97, 140, 176, 235, 293, 373, 413, 459, 499).

vol. VIII, p. 57, 168, 216, 305, 380, 553, 605, 653, 706; vol. IX, p. 58, 116).

Ore treatment (vol. IX, p. 166, 230, 285, 333, 382, 420, 560, 618, 684; vol. X, p. 62, 122, 188, 322, 446, 502, 570, 704, 765, 825; vol. XI, p. 59).

After completion of the patents on Ore Treatment the following subjects remain to be taken up in the Digest: Refining metals (subclass 57); Plating, ornamenting, electrotyping, production of articles, and miscellaneous applications (subclass 1-8, 10-14 and 16-18); Diaphragms (subclass 28); Purifying liquids (subclass 24-26); Tanning (subclass 27).

With the continuously and rapidly increasing importance of electrochemical methods in so many fields of chemical, metallurgical and industrial endeavor, it becomes daily more and more important to be able to find out quickly what was already known in the early history of electrochemistry. Nothing is more convenient or better accessible to supply such information than Dr. Byrnes' Digest covering the field completely and concisely in the volumes of this journal.

Readers' Views and Comments

In Memoriam—Henry de Mosenthal

To the Editor of Metallurgical & Chemical Engineering:

Sir:—In the history of the modern explosive industry a chapter has been closed by the death of Henry de Mosenthal on the 17th of December last from heart failure. Even only a short sketch of his life and work should prove a source of inspiration.

Henry de Mosenthal was born in 1850 at Port Elizabeth, South Africa. He had been associated with Alfred Nobel, the inventor of dynamite, for some years before the formation, in 1886, of the Nobel Dynamite Trust Company, Ltd., of which he was appointed technical secretary from the very first. This position he held until his death.

He was a well known figure in the explosives industry of the world, and his business wisdom and great ability earned for him a high reputation in the business world and in the industry with which he was more especially connected.

De Mosenthal was a Fellow of The Institute of Chemistry of Great Britain and Ireland for twenty years and served for two periods of three years as a Member of the Council. He also belonged to the principal chemical and scientific societies, and will be remembered, not only on account of his connection with the manufacturing aspect of the explosives industry, but also for his valuable scientific work.

Most of his researches were contributed to the *Journal of the Society of Chemical Industry*, and among these may be mentioned the following:

1894, Vol. 13, p. 326—"Treatment of Gold Ore at the Witwatersrand (Transvaal) Gold Fields."

1899, Vol. 18, p. 443—"The Life Work of Alfred Nobel."

1904, Vol. 23, p. 292—"Observations on Cotton and Nitrated Cotton."

1907, Vol. 26, p. 443—"Observations on Cotton and Nitrated Cotton."

1911, Vol. 30, p. 782—"Observations on Cotton and Nitrated Cotton."

The subject of Alfred Nobel was also treated in a biographical sketch in the "Nineteenth Century" of October, 1898, entitled: "The Inventor of Dynamite," and of course, de Mosenthal's intimate relations with Nobel enabled him to write a complete, and one might almost say, classical, biography of Nobel's life and work.

De Mosenthal also contributed articles on "Nitro-Glycerine and Nitro-Glycerine Explosives" and on "Permitted Explosives" to the work published by the explosives section of the Seventh International Congress of Applied Chemistry and entitled: "The Rise and Progress of the British Explosives Industry" (Whittaker & Co., 1909).

De Mosenthal was an expert in microscopy and micro-photographic work, and indeed the earliest of his papers, that on the treatment of gold ore, was already illustrated by micro-photographs. His technique, patience and manipulative skill enabled him to obtain the most remarkable results.

His researches on cotton and nitrated cotton were intended to obtain a picture of the constitution of the cellulose molecule, and he early recognized that the colloidal condition of the substances he examined necessitated methods other than those employed for bodies of a crystalline character. De Mosenthal accordingly determined values for a large number of the physical constants of cotton and of nitrated cotton, and these, together with his deductions, have thrown a good deal of light on this debated question.

In his first paper he was able to show the presence of stomata in the cotton fibre which had until then remained undetected. In his subsequent papers the effect of polarized light on nitrated fibres is described, and the connection between the degree of nitration and anisotropic effect criticised. The density of cotton and of nitrated cotton was determined, and an important difference was shown to exist between the increase

in molecular volume with ascending degree of nitration in the case of cotton, when compared with that afforded by the simpler alcoholic nitrates.

The refractive index was investigated, films of the nitrated cellulose being employed, and from the values obtained, the molecular refractions of the di- and tri-nitrates were calculated. The optical activity of nitrated cotton was examined, and its dextrorotatory character confirmed. Work on the dialysis of solutions of nitrocellulose in acetone ultimately established that the substance could not pass through a sound membrane: nor was it possible to detect any osmotic pressure.

In the last paper of the series de Mosenthal applied the results of his work to the nature of the constitution of cellulose, and stated his reasons for considering that it had an open chain structure. Whatever the view which will ultimately prevail, as to whether this or a cyclic constitution is the more rational, the values of the constants determined by de Mosenthal, the description of his methods, and his impartial discussion of the results, constitute a material contribution to the advancement of our knowledge on this difficult subject.

De Mosenthal's papers were the result of much care and infinite thought. It is characteristic that on the day before his death, he discussed his ideas and future programme of research, and it has been arranged that they shall form the subject of a scientific investigation by two of his friends and colleagues as tribute to his memory.

As an illustration of his great versatility it may be mentioned that apart from his scientific work, de Mosenthal published several monographs of a popular nature on explosives in which he exhibited to a remarkable degree his talent for popular exposition of abstruse technical subjects, and for making them understood by the general public. Some of these monographs appeared in the papers of the Savage Club, of which institution he was an esteemed member.

De Mosenthal was a bachelor and a remarkable linguist. He knew the four or five languages which he spoke with a thoroughness as though each was his own, and in addition, he possessed the rare faculty of immediately grasping the vital point in a controversy. He was a trenchant, fearless, yet kindly critic, and he will be sorely missed by a wide circle of friends and colleagues whom it is not an exaggeration to say he could count in every quarter of the globe.

—A FRIEND.

* * *

The Tungsten-Molybdenum Thermo-Couple

To the Editor of Metallurgical and Chemical Engineering:

Sir: Permit me to correct the rather surprising claim made by Edwin F. Northrup, Ph. D., in your January issue, p. 45, to the effect that he was the one who first suggested this couple at a recent meeting of the Philadelphia Branch of the A. I. E. E. I am sure those present will recall, and if there had been a stenographic record it would certainly have shown, that the proposal to make a pyrometer of these two very infusible metals was made first in my discussion of the papers and not by Dr. Northrup, although he discussed the proposition afterwards together with several others, and told me after the meeting that the idea seemed interesting and that he would proceed to make a trial of one. He certainly did not mention the couple before I rose to suggest it, for if he had there would have been no object in my doing so.

One of the members, in reply to my discussion, stated that he had tried it and had found that there was no e.m.f.; but after the meeting he explained to me that what he had tried was not a couple of these two metals, but something different, namely one between tungsten and some metal other than molybdenum.

After someone has contributed publicly to the profession a suggestion which appears to be of value to his colleagues, it is

almost invariably the case that claimants of priority at once come forth; it is then so easy to say that one had done a thing before others announced it, and even to manufacture alleged proofs; in such cases, inquiry generally shows that the alleged prior claimant had something different, probably just short of the successful element, which he then persuades himself to believe was the same, in order to take away the credit from the one who first contributed the suggestion publicly for the benefit of his colleagues, instead of secreting it for his own personal benefit. But whether it is the same or not, such practice should be discredited and looked down upon by the profession, and should be considered as not being in accord with professional ethics. The true professional spirit at such meetings is to exchange views, opinions and suggestions for mutual benefit, and if a member withholds, for his personal benefit, something which is of value to his colleagues (excepting, of course, patentable devices or trade secrets), he is not giving to others what he expects others to give to him at such meetings, and he should therefore be debarred by a code of professional ethics from making prior claims after some less selfish member had contributed it publicly to his colleagues. By secreting such matters one sacrifices his moral right to claim priority afterwards.

In his article Dr. Northrup did not give the complete description which I had given at the meeting. After I had suggested the couple of these two metals because they had a very much higher fusing point than the usual copper alloys and even cast iron and steel, and could therefore be inserted directly into the liquid metals, the question at once arose with what material the wires could be protected from direct contact with the fused metal with some of which they might alloy; one speaker asked me how to make "the hole in the liquid metal," and I then stated that I had found that electrically fused magnesite (which is on the market) withstood steel temperatures very well, and that I had seen tubes made of this material which seemed to me were not too porous. Graphite might answer in some cases.

Of course, such a protecting tube would be essential for the success of such a thermo-couple when it is to be used to plunge directly into the fused metal so as not to have to depend on measuring the temperature by radiation from the liquid surface, which in arc furnaces must necessarily be higher (and probably considerably) than the body of the liquid metal itself, or which in resistance furnaces in which the heat is generated in the liquid itself, would be lower than the metal or the important contact surface of the metal and the slag. In such a furnace the top surface of the slag might even be relatively considerably cooler although the active layer in contact with the steel may be very hot.

Besides this tube I also suggested placing in it some inert material which would either generate a gas which would be inert to the tungsten and molybdenum (hence non-carbonaceous) or would at least consume what little oxygen there was left in the tube, although if the wires were made thick enough that slight amount of oxidation might not be serious. I also suggested glazing the wires with some enamel, if such an enamel could be found.

Since Dr. Northrup found later on that the curve of this couple intersected the zero axis at 1060°C . (see his article) the question arose how a direct reading instrument could be used for a range to both sides of this zero value, as that is the very range for which this thermo-couple was suggested and would be most useful. I then suggested that either the "cold" end might be raised to some known high temperature, which presumably would shift this zero point to a less objectionable region, or an instrument of the Weston type with the point for no current in the middle of the scale, might be used. Apparently the only objection to this would be that up to 530°C . the readings would be wrong (unless indicated by a supplemental scale); but as such a couple would presumably be used only for red and white heats, the eye could be trusted to show whether the temperature was on the upward or downward part of that peculiar curve.

Such an instrument as that here suggested would be of considerable importance for use with furnaces for melting metals, which is the purpose I had in mind; and as it seems promising it is hoped that some enterprising maker will take sufficient interest in it to put it on the market.

Philadelphia, Pa.

CARL HERING

* * *

To the Editor of Metallurgical and Chemical Engineering.

Sir: After reading Dr. Northrup's article on page 45 of your January issue on "Tungsten and Molybdenum—Their Thermal E.M.F.," I feel that I owe it to the two other manufacturers of pyrometers, who contributed discussions at the meeting referred to, to exonerate them from the odium of making a statement not borne out by Dr. Northrup's subsequent experiment.

First, let me remark that while I made the statement quoted I did not put the emphasis where Dr. Northrup's italics put it. Indeed, the statement has been torn from its context and so given a scientific rather than its obvious commercial meaning. The suggestion of a tungsten-molybdenum thermo-couple was made by Dr. Carl Hering, who complained that makers of pyrometers had so far failed to provide any means of measuring the temperature at the bottom of a bath of molten steel. I stated that the Thwing radiation pyrometer had been used successfully for measuring these and much higher temperatures, and if he would find a tube which would have the necessary mechanical strength to withstand the pressure at the bottom of the bath, I would measure the temperature at the bottom of his tube with ease.

My strictures on the Mo-W couple were based, not alone on its oxidizability, but on rough determinations of the e.m.f. of each of the elements against iron, which showed at the melting point of iron an e.m.f. which was but a small fraction of that given by the platinum couple.

Dr. Northrup's more exact determinations would indicate an e.m.f. of about 3 or 4 millivolts, at 1500° or 1600°C ., which would be high temperatures for a bath of steel. This is but one-fifth that of a platinum couple, a fact which would doubtless cause this couple to be rejected by any maker of commercial pyrometers, even if it had the useful uniformity and durability.

In the course of two years devoted largely to the investigation of thermal e.m.f. of all obtainable metals and alloys, I could not have failed to discover what any physicist would have assumed a priori, namely, that any pair of elements would, at some temperature, give some e.m.f. Had Dr. Northrup had the courage to challenge my statement on the spot, he would have had his suspicion that there ought to be some e.m.f. confirmed by me, but, in that case, we might have been deprived of his prompt investigation of the problem, which I, for one, should have deplored. By all means let us find a thermo-couple which is good at 2200°C ., if there is one, not forgetting, however, to provide a suitable protecting tube at the same time. I have seen a lot of good platinum destroyed in tubes built of "hopes" much less ambitious than those of Dr. Northrup, all in a very short time after they were inserted in a bath of steel.

CHARLES B. THWING

Philadelphia, Pa.

Waste tin is an important by-product of the northwestern salmon canneries. Scrap tin is sent to Germany, where it is used in the manufacture of toys, and strip tin is shipped to California for use in the manufacture of fruit and berry boxes.

Beds of magnesite of good quality are said to exist in Lower California in the region of Magdalena Bay. Analysis shows these deposits to contain 92 per cent carbonate of magnesium. The largest bed, on Margarita Island, has an area of 30 acres. A company known as La Compania de Desarrollo y Explotacion de la Baja California, of San Diego, Cal., has been organized to exploit the deposits

The Western Metallurgical Field

Alaska

Considerable interest attaches to the proposed operations of the Alaska Gold Mines Co., which has recently taken several properties in the Juneau district under its management. A 6000-ton mill is to be built at tidewater, and a 5000-hp. hydroelectric power plant is under construction. The average grade of the ore is \$1.50 per ton, and the anticipated cost of mining and milling is placed at \$0.75 per ton. Comparison is made with the costs at the Alaska-Treadwell to show that a cost of \$0.75 for Alaska Gold Mines is not excessive. Thus, in 1911 the cost of milling at the Treadwell was only 22.52 cents for the 240-stamp mill and 18.20 cents for the 300-stamp mill. The Alaska Gold Mines' 6000-ton mill should equal these mills in point of cheap cost of operation, and consequently 25 cents is considered a fair estimate for this item. This would leave only 50 cents for mining and delivery to the mill, whereas the Treadwell cost for the same operations is about \$1. To explain this discrepancy, it is claimed that cost of mining the Alaska ore will not be nearly as great as that of the Treadwell, while the power cost will be limited to interest, depreciation and maintenance of the hydroelectric plant. The entire proposition is one which depends for success on the treatment of a large tonnage at a small margin of profit, and requires an enormous investment.

The value of the mineral output of Alaska for 1912 is estimated by the United States Geological Survey at \$21,850,000, compared with \$20,650,000 for 1911. The increase is to be credited to the larger copper production in 1912, as gold fell off slightly and silver was only increased by a small margin. The value of all other mineral products showed a slight increase. Information at hand regarding placer operations indicates that gold recovery from this source fell off during 1912; on the other hand, gold from lode mining increased.

There was no abatement in dredge installation on the Seward Peninsula in 1912. During the year 39 dredges were completed or in process of erection, and about 30 were in operation last August. It is significant that these dredges are scattered over the peninsula, in every district where placer gold has been discovered. It is assumed that the production of placer gold from the peninsula has passed its minimum for many years to come.

Revival of the Augustin Process

Fifty years ago the Augustin process of treating silver ores probably was more generally known than it is to metallurgists today. Chloridizing-roasting, leaching with salt water and precipitation of silver on copper formed the basis of the process. Recently a mill has been completed at Park City, Utah, by the Mines Operating Co., for the purpose of treating low-grade ore from the Ontario mine. The first experimental work on this ore indicated probable success with cyanidation, but later it was decided to use the other process mentioned. Details of operation at this particular mill have not been made public, but the revival of the old process is being watched with interest, and it is hoped that success will be achieved.

Mercury in 1912

Preliminary figures collected by H. D. McCaskey, of the United States Geological Survey, from the individual producers show that the domestic production of quicksilver in 1912 was 25,147 flasks of 75 pounds each, valued at the average San Francisco domestic price for the year, \$42.04, at \$1,057,180. A comparison of these figures with the final published Survey figures for 1911 and 1910 shows a gain over the output of 1911 of 3891 flasks and over that of 1910 of 4546 flasks. Twenty mines were reported as producing in 1912, of which 16 were in California against 22 producers in 1911, of which 19 were in California.

American Mining Congress

The work of the last session of the American Mining Congress is reflected in the large number of important resolutions adopted pertaining to the mining industry. There was a nota-

ble absence of formal speeches and technical papers, and a corresponding increase in debate on pertinent topics. Among the more important resolutions adopted were the following: Urging the immediate construction by federal aid of two trunk lines of railway from the Pacific coast of Alaska to the great navigable water systems of the interior; protesting against any reduction of the present tariff of zinc and lead; recommending state legislation making it a misdemeanor for any ore-buyer to mix or in any way disguise the identity of a lot of ore before its value has been definitely agreed upon by buyer and seller; urging Congress to provide for federal assistance to state schools of mines; commending Congress for its appropriation to the Bureau of Mines for investigations in metalliferous ores, and asking further increased appropriations; favoring the enactment of laws extending the principle of eminent domain to aerial trams, electric lines and pipe lines; asking the national government to assist in construction of roads through national forests; appointing a committee on mine taxation; suggesting to the President the appointment as Secretary of the Interior of a resident of the public land states, or one familiar with the public land laws.

Activity in Rare Metals

The uranium and vanadium districts of southwestern Colorado and southeastern Utah were the scene of considerable activity during the last year. An inspection of both districts gives the impression that Colorado contains smaller but richer deposits, while those of Utah are more extensive but not so high in grade. The General Vanadium Co. has purchased a large number of claims in Colorado, and has become one of the principal producers. It is difficult to give a statement of the production, as there are several small producers who sell their ores to larger concerns; but it is certain that more work was done and more ore shipped during the past year than ever before. The grade of the ore is generally low; the districts are isolated; transportation is difficult and expensive; and for these reasons development is slow. As the region is quite dry and streams are few and small, additional obstacles are placed in the way of local treatment plants. There is need for a successful system of dry concentration which will raise the grade of the ore and thereby reduce the expense of transportation and subsequent treatment. One new treatment plant was erected and placed in operation last year. It is situated in western San Miguel county, Colorado, near the junction of Disappointment creek and the Dolores river, where the Dolores Refining Co. originally had a mill.

Metal Production Statistics

The preliminary statistics of the United States Geological Survey on metal production in 1912 give some interesting comparisons with 1911. The gold mining industry was about normal, but early figures indicate the smallest production since 1907, when the output was worth \$90,435,700. In 1912 it was apparently \$91,685,168, or over \$5,000,000 less than in 1911. The decrease is ascribed mainly to Nevada, and particularly Goldfield, where the production fell off about \$4,500,000. A further decrease is noted in Colorado, particularly from the Camp Bird mine, although an increase was noted from the Cripple Creek district. The Survey has adopted the plan of distributing gold production under several heads, as placer, amalgamation, cyanidation, smelting, etc., but these figures are not yet available for 1912.

The output of silver was the largest in the past 20 years, amounting to 62,369,974 fine ounces. Only twice since 1892, namely, in 1893 and 1911, has the production been above 60,000,000 ounces, and it is expected that final figures for 1912 will show the silver production to have been the largest in the history of the country. The increase is attributed mainly to the notable increase in copper production from districts in which the ore also carries silver.

Copper production in 1912 was the largest in the history of the United States, and six of the copper producing states, namely, Arizona, Michigan, Utah, Nevada, New Mexico and

Alaska, have each exceeded all former records of production. According to the best figures obtainable, the copper production amounted to 1,249,000,000 lb. in 1912, an increase of over 140,000,000 lb. over 1911. Arizona again holds first place among copper producers, with Montana, Michigan, Utah, Nevada, California and New Mexico following in the order named.

The preliminary figures on lead show a slight decline from the high record of 1911, being approximately 480,965 short tons of desilverized, soft lead, not including antimonial lead. Spelter shows a remarkable increase, stimulated by the high prices for the metal. The production of primary spelter from domestic ore in 1912 is estimated at 323,961 short tons, and from foreign ores at 14,669 short tons, a total of 338,630 tons.

The production of tungsten is estimated at 1290 tons of concentrate carrying 60% WO_3 , an increase over 1911. Uranium and vanadium show an increase, although it is difficult to get exact figures. The uranium production is equivalent to 26 tons of uranium oxide, while the vanadium output is equivalent to 300 tons of metallic vanadium. Quicksilver showed the largest production in six years, being 25,147 flasks of 75 lb. each. The principal increase was in California; in Nevada there was a largely increased output.

The Iron and Steel Market

January has brought mixed developments in the iron and steel market, but on the whole the developments are much more favorable than there was reason to expect. The only development which can be considered really unfavorable is that crude materials, coke, ore, scrap and pig iron, have shown an easier market tone. This is not particularly unfavorable, for January normally is a quiet month for all these materials, and nothing approaching a slump has occurred in any of them.

In the steel market proper conditions are all stronger than at the close of December. It is true that there has been a decrease in forward buying of finished steel products, but this is a function of the distance ahead to which material was already sold. Some market judges appear to think the decreased volume of forward buying betokens uneasiness on the part of buyers because of the change in the national administration, involving the certainty of serious tariff revision among other things, but it is really very difficult to imagine that were this not the case, January would have seen heavy contracting all along the line for third and fourth quarter delivery. The contracting for first half had already been done.

There has been an accentuation of the pressure upon the mills for deliveries. This is seen in two ways. Buyers who had already specified for materials are strenuously representing to mills that they desire better deliveries. Again, there is heavier buying, of early deliveries and at larger premiums. The producing mills are distinctly divisible into two classes, those that sold far ahead upon contract at regular minimum market prices, and those (usually the smaller mills) which held aloof in the expectation of profiting by the premium market which has ruled of late. The strongest pressure is in plates. While the Carnegie Steel Company names a price of 1.45 cents on the limited class of orders it will accept, involving delivery far ahead, and large independents are naming 1.50 cents for somewhat similar delivery, the small independents are able to sell plates in carloads to 100 or 200 tons at 1.75 cents for delivery in four to six weeks, and in similar or larger lots at 1.60 to 1.65 cents for deliveries March to June inclusive. In merchant steel bars similar conditions prevail, and in sheets the character of the market is the same, though the premiums are hardly as large.

Production is at record rate, represented by pig iron being made in January at the annual average of 33,500,000 tons, nearly 4,000,000 tons in excess of the average during 1912, and with steel production proportionately increased. From physical considerations the output cannot be expected to be much larger than this, and under the most favorable market conditions conceivable the present year's production cannot be expected to be above 34,000,000 or 35,000,000 tons.

Pig Iron

Southern pig iron has distinctly softened, as prompt iron of many brands is readily obtainable at \$13.50, Birmingham, while some brands of Tennessee iron are purchasable on this basis for delivery two or three months ahead, and regular Alabama iron for second quarter may easily be had at \$14. This represents a clear recession of 50 cents on all positions. History shows that general advances and declines in the pig iron market are led by Southern iron, but all advances and declines in Southern iron are not followed by corresponding movements in the general market. There has been, however, some weakness in Northern markets. Chicago, influenced greatly by Southern iron, which is mixed with Northern in that territory, has shown signs of weakness though quotable prices are not lower. The Philadelphia market has seen some shading, though no general decline. In the Pittsburgh market steel making irons have been firm, but in foundry iron there has been the unprecedented incident of Cleveland iron being offered at 50 cents under the valley parity. This is novel rather than irrational, however, since it is a fact that, considering freights, Cleveland can afford to assemble raw material and deliver pig iron to Pittsburgh about 25 cents a ton cheaper than the valleys, which have controlled the Pittsburgh market from time immemorial. We quote: Bessemer, \$17.25; basic, \$16.50; No. 2 foundry, \$17.50; forge, \$17; malleable, \$17.25, all f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh. No. 2X, delivered Philadelphia, \$18.50; No. 2 foundry, delivered Cleveland, \$17.75; No. 2 foundry, f.o.b. Chicago furnaces, \$18. No. 2 foundry, Birmingham, \$13.50 to \$14.

Steel

The famine in unfinished steel, referred to in previous reports has become if possible more pronounced. There is practically no steel available in the form of billets or sheet bars. Of course occasional lots appear, but they are the exception which tests the rule, having in mind particularly how quickly they are absorbed. Thus the receiver of the Southern Steel Company desired to dispose of 5000 tons a month of open-hearth billets over the first half, it being desirable on account of the financial position of the company to have the steel sold, and the material was all quickly taken, the United States Steel Corporation absorbing 15,000 tons for its Pencoyd plant, which should be supplied fully by the Carnegie Steel Company, while the balance was taken in small lots, chiefly by Eastern rolling mills. In Pittsburgh occasional small lots are sold. Ingots have become a market commodity, there being occasional excesses of melting capacity over blooming capacity at points, matched by occasional excesses of blooming over melting capacity at others. Prices cannot be quoted closely, because there is no definite market. In a general way it may be said that billets and sheet bars, if offered, would bring \$20 to \$30, f.o.b. maker's mill, Pittsburgh or Youngstown.

Finished Steel

Effective January 13, the American Sheet & Tin Plate Company advanced its prices \$2 a ton on all regular products except tin plate, and including blue annealed black sheets, tin mill black and galvanized sheets, carrying also corrugated roofing which are now sold on the basis of flat sheets plus extras. Many of the independents had already advanced their figures, and the advanced figures of the leading interest at once became the minimum of the market. Early in the month cold rolled shafting was advanced two points or \$2 a ton to 58 per cent off list in carload lots, delivered in base territory.

Regular prices for forward delivery are given below, plates, bars and sheets commanding a premium for almost any delivery before June, while small premiums rule on some other products. Prices are at Pittsburgh unless otherwise specified. Rails, standard sections, 1.25 cents for Bessemer 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.50 cents

Shapes, 1.50 cents.

Steel bars, 1.40 cents.

Steel hoops, 1.50 cents

Common iron bars, 1.65 cents, Pittsburgh; 1.60 cents, Chicago; 1.67½ cents, Philadelphia.

Sheets, blue annealed, 10 gage, 1.75 cents; black, 28 gage, 2.35 cents; tin mill sizes of black, 28 gage, 2.30 cents; galvanized, 28 gage, 3.50 cents; painted corrugated, 28 gage, 2.55 cents; galvanized corrugated, 28 gage, 3.55 cents.

Tin plates, \$3.60 for 100-lb. cokes.

Merchant steel pipe, ¾ to 3-in., 80 per cent off list.

Steel boiler tubes, 3½ to 4½ in., 71 per cent off list.

Standard railroad spikes, 1.85 to 1.90 cents.

Button head structural rivets, 2.20 cents; cone head boiler rivets, 2.30 cents.

The Non-Ferrous Metal Market

There were no unusual features in the non-ferrous metal market in January. Sales of metal were rather below normal, and quoted prices were in many instances discounted in order to obtain business. Domestic consumers seem to buy only for immediate needs and from small agencies. More business was transacted in lead on account of the favorable price, but later in the month this market was dull also.

Copper.—The principal producers have not realized their quoted price. Only a small business in Lake copper is reported. Sales for future delivery have been made at about 17.40 cents, New York, and the market was generally dull. The last quotations are 17¼@17¾ cents for Lake and 17.35@17.45 cents for electrolytic.

Lead.—This market has shown a firm tone, but business disappeared when higher prices were asked. The St. Louis market quotes 4.15@4.20 cents, with New York at 4.30@4.35 cents.

Tin.—Early in January there was a slight premium on spot metal, but this later disappeared. Visible supplies at the end of the year were smaller by 5,500 tons than at the beginning of 1912, and it is expected that this will result in an upward movement in the London market. January tin was quoted at 50¾ cents, New York.

Spelter.—Not much business has been consummated in this market, especially for immediate delivery. Future deliveries are discounted considerably. The market is practically unchanged at the following prices: 7.05@7.10 cents, St. Louis, and 7.10@7.20 cents, New York.

Other Metals.—No material change is noted in aluminium, the last quotations being 26@26½ cents, New York. Sales of antimony have been light, prices ranging from 9 to 10¼ cents for various grades. Business in quicksilver has been light, and prices remain at \$40 per flask of 75 lb., New York and San Francisco.

A New Honor for Dr. Acheson

Dr. Edward Goodrich Acheson has been made an honorary member of the Imperial Technological Society of St. Petersburg, Russia, before which body he delivered an address on the evening of Jan. 20.

American Museum of Safety

On Jan. 23 the American Museum of Safety, 29 West Thirty-ninth Street, New York City, held its annual exercises for the award of its four gold medals.

The *Scientific American Medal* for the most efficient safety device invented during the last three years and exhibited at the Museum's exhibit hall was awarded to the *Dräger Oxygen Apparatus Company*.

The *Travelers' Insurance Company Medal* for the American employer who in the judgment of the American Museum of Safety has done the most for the protection of the lives and limbs of workmen was awarded to the *New York Edison Co.*

The *Louis Livingston Seaman Medal* for progress and achievement in the promotion of hygiene and sanitation and the mitigation of occupational disease was awarded to the *National Cash Register Company*.

The *Rathenau Medal*, placed at the disposal of the Museum by the Allgemeine Elektrizitäts Gesellschaft, for the best device or process in the electrical industry for safeguarding industrial life and health, was awarded to Mr. *Thomas Alva Edison*.

Atlantic City Meeting of the American Electrochemical Society

The Spring meeting of the American Electrochemical Society will be held at Atlantic City on April 3, 4 and 5. Headquarters will be at the Hotel Traymore. Members and guests are advised to secure their accommodations early.

No complete program of papers, social functions, and excursions has been arranged so far, but it has been fixed that there will be a symposium of papers relating to the electrolytic decomposition of metals and electroplating, the different metals to be taken up by different authors.

Dr. W. Lash Miller, of the University of Toronto, Ontario, Canada, is the president, and Dr. Jos. W. Richards, of Lehigh University, South Bethlehem, Pa., is the secretary of the Society.

Chemists' Club Library

The library of the Chemists' Club in New York is now getting into fine shape and is being used by a steadily increasing number of members and visitors.

As one fruitful source of criticism of libraries in general is ignorance of the method used by the particular library which is visited by the critic, the library committee has issued the following advice "how to use the library":

"Thanks to the kind co-operation of Dr. Billings, director of the Public Library, and Mrs. Smith, librarian of the Academy of Medicine, any book can be located exactly on the shelf by a system that will never obstruct unlimited expansion.

"To find a book (say, Lunge, Sulphuric Acid, etc.), look it up in the card index by either author, title or subject. Note the class mark (Acid) in the upper corner of the card. Refer to the chart on the card case and find the shelf mark for this class (F 3.5). Go to the stack labeled with this letter (F) on the side marked with the first figure (1-3) to the vertical row of this number counting from left to right (the third) and the shelf of the last number (5) counting from the top down.

"To find all books on a general subject: Look for your subject (or the division that includes it) on the chart, note the shelf number and proceed as above.

"When more than one shelf is occupied by a group, the books are in alphabetical order by authors' names.

"The library is open to the public from 9 a. m. to 5 p. m., except Saturday afternoons, Sundays and holidays; entrance through the public hall and elevators to the third floor."

The Petroleum Industry in 1912

There was no considerable change in the quantity of petroleum produced in the United States in 1912 compared with 1911. Nevertheless, according to David T. Day, of the United States Geological Survey, the year was as full of remarkable incidents as is usual in the history of this article of commerce, which depends for its statistical position more upon the chances of new discoveries and less upon trade demands than any other commodity except gold.

As a rule, the Eastern fields declined in production, because it was impossible to keep up the great output of 1911 without large additional discoveries of new pools in the older fields. The Eastern decline was, however, offset by the increase in California, where the San Joaquin Valley fields (Midway, McKittrick, Maricopa, etc.), are still at the height of the gusher stage.

The total production in the United States in 1912 was 220,200,000 barrels (of 42 gallons) against 220,449,391 barrels in 1911. The chief producers were California (87,000,000 barrels), Oklahoma (52,000,000 barrels), and Illinois (28,000,000 barrels).

Presentation of the Perkin Medal to James Gayley

The Dry Blast Process as One of the Greatest Achievements in Modern Metallurgical Chemistry

THE Perkin medal was awarded for the present year, to Dr. James Gayley. The medal is awarded annually to a chemist residing in America "for achievements of high value in applied chemistry." The award is made by the Perkin Medal Committee which is constituted of representatives of the different chemical societies of this country. The former recipients of the medal were: Sir William Perkin, Dr. J. B. Francis Harreshoff, Dr. Arno Behr, Dr. Edward G. Acheson, Dr. Chas. M. Hall, and Mr. Herman Frasch. The award to Dr. James Gayley was made in recognition of his dry-blast process as one of the greatest achievements in modern metallurgical chemistry.

The formal presentation of the Perkin medal to Mr. Gayley took place at the meeting of the Society of Chemical Industry held at the Chemists' Club in New York City on the evening of January 24.

Prof. M. C. Whitaker in calling the meeting to order emphasized the presence of so many captains of industry who had come to honor Mr. Gayley. He then introduced Dr. Chas F. Chandler, the senior vice-president of the Society of Chemical Industry who made the presentation speech.

In giving a sketch of Mr. Gayley's life Dr. Chandler mentioned that James Gayley is the maternal grand-nephew of Sir Henry Bell, who established steam navigation on the Clyde, where he launched "The Comet" in 1812. He was born at Lock Haven, Pa., October 11, 1855, the son of Samuel A. and Agnes (Malcolm) Gayley. The following brief statement of the successive stages of his professional life are significant:

"Mr. Gayley began his professional life as chemist for the Crane Iron Company, Catasauqua, Pa., 1877-80. He was next superintendent of the Missouri Furnace Company, St. Louis, and later was the manager of blast furnaces, E. & G. Brooke Company, Birdsboro, Pa., 1880-85. In 1885 he became manager of the blast furnaces at the Edgar Thomson Works, and he was subsequently promoted to the position of manager of the Edgar Thomson Works, and he later became a managing director of the Carnegie Steel Company. In 1907, he was made first vice-president of the U. S. Steel Corporation, remaining in this position until 1909."

Dr. Chandler emphasized the enormous importance of the dry air blast and in concluding he presented the Perkin medal to Dr. Gayley with the following words: "It gives me great pleasure, as the representative of the Society of Chemical Industry, and the affiliated chemical and electrochemical societies, to place in your hands this beautiful token of the appreciation and affection of your fellow chemists."

Dr. Gayley replied in a few graceful words of thanks, followed by a most interesting story of the evolution of the dry-blast process. His speech follows practically in full:

The Story of the Dry-Blast Process

By James Gayley

"I wish to thank the speaker for this medal which he has presented to me with such gracious words. I wish also to express my thanks to the awarding Committee and the Societies which they represent in conferring this great honor upon me. I appreciate it still more because the committee have stepped aside from what is purely the chemical industry to another great industry, that of iron metallurgy, which, nevertheless, is one in which the application of chemistry is the controlling factor, and this recognition of the broader field of chemistry brings

with it a keen sense of appreciation of this rare honor and distinction conferred upon me. Again I thank you.

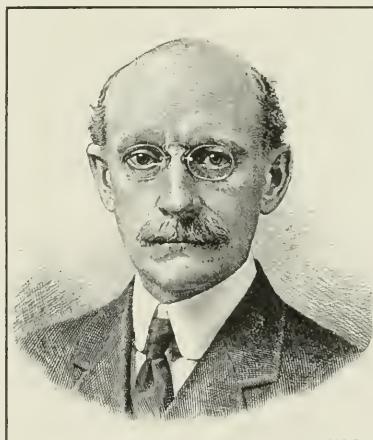
"The iron blast furnace is but the chemist's crucible on a gigantic scale—operated on chemical principles and the mechanical appliances are instruments of precision, constructed and adjusted to carrying out in efficient manner the chemical reactions involved.

"The 'rule of thumb' practice, which ended thirty-five or forty years ago, is well nigh forgotten—that period in which chemistry was not applied, and a ton of ore was a ton of ore all the world over, irrespective of its reducibility or chemical composition; when the iron master was more interested in the output of 1.8 iron than in the fuel consumption, and would tell you that 'lime made heat.' David Thomas, that grand old pioneer of the anthracite iron industry, summed up the requisites for successful furnace practice as 'wind and heat.' The application of chemistry to iron smelting has effected a wonderful transformation in the industry, so that today the chemist occupies the foreground in directing its operations.

"Moisture in the atmosphere has long been recognized as a variable and influential element in the manufacture of iron. Truran, in his 'Manufacture of Iron,' written in 1802, refers at length to the influence of moisture on the quality and make of iron, and the increased consumption of fuel in the summer season, but does not touch on the problem of its regulation or removal.

"Later, in England, it was proposed to extract the moisture by passing the air over lumps of chloride of calcium, but Sir Lowthian Bell showed the impracticability of that scheme on account of the rapidly diminishing power of absorption possessed by the calcium chloride.

"Charles Cochran, an eminent iron manufacturer of Great Britain, proposed and obtained a patent on extracting moisture from the air by bringing it into intimate contact with liquid calcium chloride flowing over chains suspended in a chamber; but later he declared respecting the benefits to be derived from extracting moisture that 'the game was not worth the candle.'



JAMES GAYLEY

Nevertheless, the moisture in the atmosphere was recognized as a disturbing element. Some of the earlier makers of anthracite iron in the Lehigh Valley stated that they frequently knew in the morning what kind of iron their furnaces were making by observing the weather conditions; yet, prior to the first dry-air installation in 1904, this variable element of moisture was accepted in a spirit of resignation, as something to be endured, like storm and sunshine, a condition beyond control.

"It was my privilege to be invited in 1885 to take charge of the Edgar Thomson blast furnaces of the Carnegie Steel Company, a plant which represented the most advanced construction in blast furnace equipment, and was supplied with the best fuel and ores. In the operation of these large furnaces, the influence of the atmosphere was forcibly impressed, as the steel works were being supplied with hot metal direct from the furnaces, for the Jones metal mixer had not come into use. It was a difficult matter to supply metal approximating uniformity even under the best conditions, and quite impossible under conditions of varying atmospheric humidity.

"For several years observations were made twice a day of the moisture in the atmosphere, and a direct relation was established between the quantity of moisture and the grade of metal, with particular reference to the silicon which is the heat-producing element in the acid Bessemer converter. In the summer season, the air blown into the converter was heavily charged with moisture and, in consequence, a higher percentage of silicon in the metal was normally required, while at the same time these atmospheric conditions operated on the furnaces to produce a lower-silicon metal; thus the operating of furnaces in conjunction with Bessemer converters on direct metal may well be presumed to have been a difficult and trying proposition.

"Under these conditions of manufacturing, the removal of the moisture from the atmosphere, or rather its reduction to a low degree, and one of practical uniformity, appeared to hold out the only means of effecting regularity, and at the same time to promise a considerable saving. It was a matter of constant observation that a sudden change in the weather, making a quick drop in the temperature and content of moisture, would cause the furnaces to work excessively hot, but this period of dryness could not be taken advantage of, for before any changes could be made effective in the furnace charge to meet current atmospheric conditions the moisture might have increased to its normal seasonable amount.

"Observations of the humidity in the atmosphere were made daily from 1886 to 1890, when the first experiments for moisture removal were begun. These observations were made at the beginning in the morning and evening, and later were made every hour. Observation was also made of rainstorms to determine the moisture before, during and after the rain, and it was noted that the moisture usually dropped quickly as the rain began to fall. Throughout this period of moisture tests, the working of the furnace was observed in its relation to the amount of moisture in the atmosphere.

"A modern furnace consumes about 40,000 cu. ft. of air per minute, and for each grain of moisture per cubic foot there enters the furnace 1 gal. of water per hour for each 1000 cu. ft. that is, if the moisture was 1 gr. per cu. ft. 40 gal. of water would enter the furnace per hour. The variation of moisture from month to month in the Pittsburgh district is shown in the following table:

	Average Temperature Degrees Fahrenheit	Weight of Water Per Cubic Foot of Air Grains	Quantity of Water Entering Per Hour Into a Furnace Using 40,000 Cubic Feet of Air Per Minute Gallons.
January	32.6	3.18	87.2
February	31.7	1.83	73.2
March	45.0	3.40	136.0
April	51.0	3.60	120.0
May	61.6	4.80	162.0
June	71.6	5.03	237.6
July	76.2	5.60	224.0
August	71.6	5.16	206.1
September	70.4	5.68	227.2
October	56.4	4.00	160.0
November	40.4	3.35	94.0
December	36.6	2.25	90.0

While the moisture in January averaged 2.18 gr., yet the total variation for the month was from 0.36 gr. to 2.55 gr., and while June shows an average of 5.94 gr., yet the extreme variation was from 4.8 gr. to 8.50 gr., or, in other words, the moisture entering the furnace varied from 192 to 340 gal. per hour. During the midwinter period the moisture may vary as much as 150 per cent in the same day, but in midsummer it rarely varies more than 25 per cent.

"Having secured extensive data on the varying humidity of the atmosphere over a period of five years, the problem of reducing the moisture and making it uniform next presented itself.

"The experience of English engineers was against the absorption process with calcium chloride, and my predecessor at the Edgar Thomson Works, Mr. J. H. Cremer, had constructed an apparatus to dessicate in the same way, but he obtained very unsatisfactory results. As our observations showed a reduction of moisture through a reduction of atmospheric temperature, the reduction of temperature through mechanical refrigerators presented itself as the obvious method of removing the moisture; and on this basis the first experiment was undertaken in 1890 in a crude way.

"We began by passing air through two chambers in succession; in the first chamber the air came in contact with pipes cooled by water, and in the second chamber anhydrous ammonia was expanded through the pipes. As the air came from the blowing engine it was quite warm, but after passing through the chambers it was very cold; each chamber was only 6 ft. long and the volume of air treated was small, indeed.

"Various forms of apparatus were constructed from the period of 1890 to 1895, and tests were made as opportunity offered. During this period my duties had been changed from that of superintendent of the furnaces to general superintendent of the whole works, thus adding to my duties that of the steel works and mills. There was in consequence not much time free to make experiments, for the demands of a large steel works on its manager's time were persistent and exacting, and experiments and improvements along other well-defined lines represented more immediate profit and demanded thought and attention. But the general plan for a final test on a large scale had been worked out, which was to be tried later. In 1896, I left the active management of the works to enter upon the general business of the Carnegie Steel Company in connection with the ore department and the assembling of all raw materials. If the opportunities for experimenting were rare when stationed at the works, they became more so when located in the general office.

"I therefore secured the services of a young engineer, Mr. Walter, from a refrigerating company, and when they learned the nature of my experiments they kindly loaned me a small ammonia compressor to carry on the work. We used two air-drying chambers, each 4 ft. square and 8 ft. long, and filled with pipe coils, and equipped with the necessary refrigerating appliances. A blast engine with an air cylinder, 3 ft. in diameter and capable of supplying 3500 cu. ft. of air per minute was installed, and from this engine and its small air supply the data was worked out for treating 40,000 cu. ft. of air, which was the quantity required to supply a modern blast furnace.

"I mention that we had to work out the necessary data for a practical demonstration, and the suggestion might arise, why did we not avail ourselves of the experience of refrigerating firms? Indeed, we tried. The proposition was put up to several important makers of refrigerating machinery, but without any substantial benefit. To refrigerate the air for cold storage rooms was a specification they readily understood, but the treatment of a hurricane of air was an entirely different problem to them. We were, therefore, compelled to work out the data ourselves and make our own card of estimate.

"It is unnecessary to give in detail the many and varied experiments made in treating the air, nor to describe how certain obstacles, which seemed to block our progress, were successfully overcome. In our list of experiments we tried rarefac-

tion with refrigeration, but we did not obtain satisfactory results.

"For quite a period the chamber into which the cold air was discharged accumulated a large quantity of snow, and the air flowing in produced a veritable snowstorm. This was delightful on a hot July day, but unless we could overcome this snowstorm it would be disastrous to the process, for the snow would soon become aqueous vapor. We tried to extract it from the air, but without success.

"The essential thing was not to produce it, and in this we were finally successful. We were trying to refrigerate to a very low and unnecessary degree and the frost did not become fixed on the cold pipes; by increasing the temperature of the brine circulating through the pipes the snow became fixed and promptly disappeared from the cold air blast. We worked generally under conditions of the highest humidity in the Pittsburgh district, which we secured by admitting steam to the air inlet when necessary.

"About the year 1900 we had plans worked out in detail for a plant to treat 40,000 cu. ft. of air per minute, and I then applied to my company for an appropriation of \$100,000 to make this installation, and here my most difficult work began—to persuade my associates that moisture in the air could appreciably, even in a small degree, affect the working of a blast furnace.

"It could hardly be expected that men who came up through the ranks of the steel works or office should appreciate the influence of atmospheric changes on a blast furnace that to all external appearances was crude and rough, but which in its interior adjustment was really on a most delicate balance. Money was readily appropriated for the mechanical appliances of the furnaces, or for securing uniformity in the physical and chemical composition of the ores, coke, and limestone, and a variation therein of 10 per cent to 20 per cent was promptly corrected, but the atmosphere might vary from 20 per cent to 150 per cent even in a single day without giving any concern, for few realized that the weight of air consumed per ton of iron was 50 per cent greater than all the other raw materials combined.

"It was not until 1903 that I secured an appropriation to construct the first plant, although I had offered two years before to assume personally half the expense.

"Early in June, 1904, the dry-air plant, which was erected at the Isabella furnaces at Pittsburgh, was in proper working order and ready to be put on the furnace. It was necessary to apply the dry air gradually so as not to disturb the furnace adjustment, and as there were three blast engines we arranged to first supply dry air from one engine, making the blast one-third dry air. We decided to apply this dry air at 9 a. m. and I had the weight of ore charge increased by 5 per cent, and this increase was timed to reach the combustion zone of the furnace at 11 a. m. The furnace manager protested against the addition of 5 per cent, contending that a 2 per cent increase was about the limit, and if the dry air did not prove efficient he would then have a very cold furnace on his hands. Notwithstanding the objection, the increase was made. When the dry air was turned on at 9 a. m. the furnace soon showed the effect of it, the gas became grayer, a sure sign of the furnace working hotter, and when the 5 per cent increase in ore had reached the hearth it was not discernible that the furnace had been given an increased duty.

"The operations of the furnace were conducted on a conservative basis, and the long time spent in working out and developing the air-drying plant was rewarded, for the plant started and continued to perform its work without a hitch. By the sixth day the furnace was being supplied with two-thirds dry air and one-third natural air, and 9 per cent increase in ore charge had come to work; and the furnace condition was splendid. I might add here that the 9 per cent increased weight of ore charge represented the same percentage increase in output and a like reduction in the amount of fuel used per ton of iron.

"On the afternoon of that sixth day in the use of dry air

I left the works well satisfied with the progress made and confident of the future by reason of the many little indications easily recognized by practical furnacemen that a still heavier duty would be responded to.

"I had not gone far when the sound of an explosion reached me, and, on turning, I saw great clouds of steam rising from the works. I hastened back, and feeling my way through steam and dust, I was dismayed to find that the refrigerating chamber had collapsed.

"There it lay, a tangled mass of concrete and distorted pipe coils.

"The building gave way because during a part of the time employed in erecting the dry-air plant there was a bricklayers strike and the works manager not wishing to delay construction built the refrigerating chamber of concrete. The dry-air blast conduit leading from the top of the chamber to the blast engines was constantly vibrating from the impulse of air and the vibrations of the engines. These vibrations were in turn communicated to the dome of the building, weakening the base of the concrete and caused the walls to collapse, which in their fall broke the steam pipe leading to the compressors. The debris was cleared away, and in less than two months the plant was again started.

"We began as before and in two weeks' time had secured a fuel economy of 15 per cent, and then by small increments we gradually worked up to a point of 20 per cent efficiency. That is, we effected a fuel saving of 20 per cent and increased the output to the same extent. This practice was continued for about six months, but the manager thought it was operating too close to the danger line to take care of irregularities in the raw materials, and the fuel economy was adjusted to 15 per cent. Even with dry-air blast there is a disinclination to work up to the high limit of efficiency, for the prior experience of every furnace manager with natural air had shown the necessity of keeping a good margin of safety."

Mr. Gayley then exhibited two models of apparatus for drying air. In the first model, which represented his first installation, there is a chamber filled with pipes and the air is cooled by indirect contact with a cooling fluid, which is brine. The temperature of the brine is such that the air entering the chamber at 80 deg. is discharged at 23 deg.

The second model represented the two-stage process in which the air is brought into direct contact with refrigerated water in the first chamber and is reduced to about 35 deg. in temperature, then it passes to the second chamber filled with pipe coils through which brine circulates and the final temperature is obtained. The two-stage method is more cheaply operated and costs one-third less to build.

"It was not our purpose throughout our experiments to work out the most economical method of drying air for the first installation. What we aimed at was to construct a plant that would efficiently and continuously produce dry air in order to determine its economical value in the furnace."

Mr. Gayley then showed a chart which illustrated the variability of moisture in natural air and the regularity of dry air. During the thirty-one days of July, 1910, the moisture in the natural air averaged between 31 and 8.22 per cubic foot of air, while the moisture in the dried air of the dry-blast process was uniformly equal to 0.822 of moisture per cubic foot of air. In other words, with dry blast 800 gal. of water were delivered to the blast furnace every twenty-four hours, while with natural air the amount of water delivered would have averaged between 3400 and 7800 gal. per twenty-four hours.

"The records obtained at several works from the use of dry air, and which in each case cover a considerable period are as follows:

"Works A—Obtained a decrease on coke consumption of 10.5 per cent and an increase in output of 23 per cent.

"Works B—Obtained a decrease in coke consumption of 6 per cent and an increase in output of 15.3 per cent.

"Works C—Obtained a decrease in coke consumption of 9.8 per cent and an increase in output of 11.8 per cent.

"Works D—Obtained a decrease in coke consumption of 10.5 per cent and an increase in output of 16 per cent.

"The results obtained at different plants are modified by the raw material used, and the personal equation of the management enters into it.

"The results given above are taken from works in three different countries. *The claim I personally make for the process is that with an increase in output of 10 per cent the saving in fuel per ton of iron will be reduced 10 per cent, and I consider this conservative.*

"One of the advantages obtained through dry air is the elasticity it permits in the furnace operations to meet business conditions. If trade is dull the furnace can be operated to secure the maximum of fuel economy with a small increase in output, but when trade is 'booming' and pig iron is in demand, and increased output is more desirable than a saving in fuel, the furnace can readily be made to respond.

"The manager of the Dowlais Works, at Cardiff, Wales, experimented with one furnace to determine what results could be obtained under each condition of operating.

"For a period the furnace was operated principally to secure increased output and obtained an increase of 26 per cent with a fuel saving of 12.3 per cent.

"In the next period they operated principally to obtain decreased fuel consumption and obtained an increase in output of 12 per cent with a decrease in fuel of 17.4 per cent.

"In addition to the advantages from dry air already mentioned, it is found that the furnace works with greater regularity and in consequence a more uniform metal is produced, and a required grade of metal can be made with precision.

"In low-silicon irons the sulphur does not increase as it does when natural air is used.

"The saving in coke effects a corresponding saving in the limestone necessary to flux the ash of the coke, and there is less phosphorus entering the metal.

"As the furnace charges settle more regularly, there is less fine ore carried over with the gases.

"A manager of blast furnaces using dry air recently said that they found it to be an 'educator' in their practice, as it had taught them the advantage of securing regularity in many other directions.

"I shall not attempt herein to set forth any of the theories advanced to account for the fuel saving, which has been found to be in excess of what is necessary to dissociate the moisture. That feature has been ably discussed by Dr. Henry M. Howe, Prof. J. W. Richards, Mr. J. E. Johnson, Jr., and many others in this country, and also by metallurgists in England, Germany, and France; and their full discussions can be found in the transactions of the American Institute of Mining Engineers, the *Journal of the British Iron and Steel Institute*, and metallurgical journals.

"Towards a full consideration of the dry-air blast, a few points may be referred to briefly:

"It did not come into its work as did the Neilson hot blast in 1828 when the art was crude and the appliances of the furnace were poorly adapted to the work. But to the Scotchman Neilson all credit is due for effecting a fuel saving of 30 per cent.

"The dry-air blast was tried out when the equipment of the furnace was as perfect as human skill could make it. The chemistry of furnace operations was well understood and its management was on a skillful and scientific basis.

"Again, it was not possible for dry-air blast to have an experimental stage. Nearly every device or process can be tested in an experimental way and on a comparatively inexpensive scale. Bessemer could blow air through molten metal in a pot and demonstrate the value of the pneumatic process, but it would have been of no value to build a small plant to refrigerate a part of the air, nothing conclusive would have been shown. Nor would it have been of any value to treat the whole air supply of a diminutive or toy furnace. To efficiently demonstrate its value it had to be applied to a furnace that was equipped and operated according to the most ad-

vanced state of the art, and not only was it essential that the whole air supply should be treated, but also that the method and means of treatment should measure up in capacity and efficiency and operate as continuously as any of the other accessories of a modern blast furnace. To achieve this it required time, patience, and many experiments, and now with its accomplishment, in removing irregularity from the most important and invariable factor in the winning of iron from its ores, it apparently removes the last barrier toward the attainment of ultimate economy in the present field of practice.

"This, in brief, is the story of the dry-air blast."

Addresses by Prof. Howe, Dr. Hart and Dr. Raymond.

Prof. **Henry M. Howe**, of Columbia University, followed with a brilliant address on the metallurgy of the dry-blast process, spiced with sarcasms on the light which the development of the dry-blast process has thrown on the value of expert opinion.

A saving of fuel of 20 per cent or 10 per cent had been considered as preposterous, as absolutely impossible. But then it became a fact. "Let us learn the lesson of humility." Now as the heat required for heating and dissociation of the moisture is so very small, how can the big saving in fuel, due to dry blast, be explained? Dr. Howe emphasized the supreme importance of temperature. There is a critical temperature in the blast furnace reaction and accordingly there is hypercritical and hypocritical heat and hypercritical and hypocritical work, and any change of the ratio of hypercritical to hypocritical heat has its effect.

We hope to publish Professor Howe's address in a future issue.

Prof. **Edward Hart**, of Lafayette College, followed with an address on "Dr. Gayley's Interest in Education." Professor Hart told of what Dr. Gayley has done for various institutions of learning in this country. His speech was delightful, as he told of many humorous and intimate reminiscences from Mr. Gayley's early life. While Professor Hart was Mr. Gayley's teacher in chemistry, yet he emphasized that Mr. Gayley had always been highly original and was in many ways a self-educated man. One of Mr. Gayley's chief traits of character was his consideration for other people's feelings.

The session was concluded by an eloquent and highly poetical speech by Dr. **Rossiter W. Raymond** on the present time as a "new age."

Old Freibergers in America.

On Friday evening, Dec. 20, at the Hofbrau-Haus, Broadway and Thirtieth Street, New York City, a number of old students of the Freiberg Bergakademie sat down to dinner. This meeting was called for the purpose of forming an association in America to be known as the "Old Freibergers in America."

After the dinner a business meeting was held and the following officers were elected: President, R. W. Raymond, Freiberg, 1861; vice-president, Gardner F. Williams, Freiberg, 1868, and secretary-treasurer, C. L. Bryden, Freiberg, 1907.

It was decided to hold two meetings a year, one on March 25, the anniversary of the founding of the Akademie, March 25, 1765, and the other on Dec. 20, this to be the annual meeting.

The following members were present: Dr. R. W. Raymond, 1861; Gardner F. Williams, 1868; P. J. Oettinger, 1867; Stuart M. Buck, 1868; T. Waln Morgan Draper, 1876; Baron Alfred von der Ropp, 1882; Franklin Guiterman, 1877; F. G. Corning, 1879; R. Boice, 1908; Albert Meyer, 1908; R. M. Payne, 1909; Dr. E. E. Lungwitz, 1886; F. H. Sistersmans, 1885; George M. M. Godly, 1900; Walter V. Rohlfis, 1903; H. H. Knox, 1886; H. A. Wilkens, 1880, and C. L. Bryden, 1907.

All old Freibergers who have not already done so are requested to send in their names and addresses to the secretary, C. L. Bryden, 1015 Myrtle Street, Scranton, Pa.

Utilization of Waste Wood

Raw Materials

By John E. Teeple, Ph.D.

For the purposes of this article we will include under the term "Raw Materials of Wood Waste," saw dust, ends, slabs and edgings from lumber mills, stumps from lumbering operations, and fallen or dead timber in the woods, caused by storms, and not suitable for lumber. We do not include the tops and branches of trees cut for lumber and left in the woods, because no suitable method has yet been worked out for rendering them commercially valuable, and we do not include young trees, particularly of short-leaf pine such as are used for making paper pulp to some extent, because it is permitted to grow they would eventually be suitable for lumber.

In kind, waste wood may be the long-leaf pine of the South Atlantic and Gulf States; short-leaf pine of the same general region, and various other pines intermediate between them in characteristics: Oregon pine or Douglas Fir of Oregon, Washington and British Columbia; Norway or red pine of Ontario, Manitoba, Minnesota, Wisconsin and Michigan; and hardwood of many parts of the United States and Canada.

This gives us at least four forms, viz., saw dust, slabs, stumps and fallen timber, and at least five classes of wood.

It is obvious that the recovery of valuable materials from short-leaf pine, saw dust for example, is an entirely different problem from that presented by long-leaf pine stumps, which is again different from the treatment of hardwood slabs. In fact, this simple classification into four forms and five kinds of wood presents of itself twenty distinct problems.

The situation is further complicated by differences in each class; long-leaf pine stumps for example differ widely in composition, depending on age, soil, elevation of the land, latitude, and probably on the season of the year when the tree was cut.

When any proposition for utilizing wood waste is presented then, the question of form and kind of raw material and location becomes exceedingly important. The only safe assumption is that conditions in any one place are not identical with those in any other place. The individual problems presented are practically as numerous as the individual locations and conditions. These differences must be taken into account and studied carefully before any specific plant can be designed, or any particular process recommended intelligently for the case in hand.

Given a general class of conditions, one can outline a general plan of plant, or predict the probable success or failure of a general idea when applied to those conditions. He cannot go much farther than this without a careful examination of the specific raw material and location. Too often success or failure depends on specific adaptation to individual local details, and not on general ideas applied to general conditions. This is especially true in this industry, dealing as it does with bulky materials not easily transported, not permitting much in the way of handling charges, yielding comparatively small amounts of products and those products not universally salable, often none of them salable within hundreds of miles of the point of production.

We cannot then give too close attention to raw materials, and it may be necessary or advisable in a particular case to reject one-half or nine-tenths of the waste offered in order to work successfully with the remainder.

No matter what form or kind of wood is presented within the limit specified above, we may for our purpose consider it to be composed of three parts: First, water; second, structural frame work, and third, gum resins. The first part, water, may form anywhere from 25% to 50% of the weight of the wood. It is often given a high value in the prospectus, but the demand for it seems to be rather light. The second constituent, the structural part of the wood, consists essentially of cellulose, various lignocelluloses, and similar products chiefly carbohydrates. This makes up the wood itself. It is the frame work, the part that gives it texture. It is, of course,

present in all woods, and, in many of them, particularly in the hardwoods, there is no other important constituent present excepting this cellulose framework and the water. The third constituent of the wood is resin or gum resin. This affects the color and weight of the wood, and is a deposit in the interstices of the woody framework.

When extracted from wood, the gum resins may be easily separated into turpentine, pine oil, and rosin. In pine woods, the first or volatile part of the resin is turpentine, or a mixture of terpenes, the kind and quantity of the different terpenes varying in different species of pine. The second part, still volatile but boiling at a somewhat higher temperature than the turpentine is pine oil. It consists essentially of hydrolytic and oxydation products of the terpenes. Most of the pine oil constituents contain oxygen. The third part of the gum resin deposit is rosin. This is the solid part not easily volatile with steam. Rosin is a mixture of different resins in different species of pine. The gum resins were the first products that led to the utilization of pine wood for uses other than lumber, and they still give the main value to waste wood. Considering the importance of the gum resins, it might be well to devote some attention to their origin and formation if we had any definite information about them.

If we make an incision into the pine tree it will nearly always produce a flow of sap or oleoresin or gum resin of a semi-liquid character, consisting of terpenes and rosin, no appreciable amount of pine oil. At the same time there is an increased deposit of gum resins in the body of the tree itself in the neighborhood of the wound. It seems probable that the oleoresin flowing from the tree is a combination of the natural sap of the tree and of another secretion formed to protect and reinforce the tree in the neighborhood of the wound. In case this wounding or chipping is continued from year to year, as is done in tapping trees for turpentine, the amount of deposit in the tree in the neighborhood of the wounded part becomes very large. The wood becomes so pitchy, or fat, that the butt of the tree is not suitable for lumber, particularly on the outside, and we have box faces or fat slabs which form a part of the waste of mills cutting long-leaf pine that has been chipped for turpentine.

When a long-leaf pine tree is cut in the woods, leaving a stump, the same process seems to go on in the stump. The sap wood gradually rots away, and the heart of the stump becomes richer and richer in gum resins as the years pass, producing finally what are known as fat light-wood stumps. This same thing occurs when a tree is blown down by storms, or if it dies standing, particularly where it has previously been tapped for turpentine. In some cases the stump becomes rich in gum resins down below the surface of the ground throughout most of the length of the tap root. In other cases there is no appreciable deposit of gum resin below the level of the ground. In the case of Douglas fir the deposits of gum resin are mainly in pockets produced by wind shakes, instead of throughout the framework of the tree itself, and this makes the yield of gum resin from that species very much more irregular than is the case with the long-leaf pine.

The fact that pine oils are important constituents of the gum resins extracted from old wood while they are scarcely present in the oleoresins obtained from the live tree by chipping it is probably due to slow processes of oxydation and hydrolysis going on in the stump or wood in the forest.

In the matter of cost: Waste from mills can usually be had at an exceedingly low figure, sometimes for nothing. The amount of waste usually considerably exceeds the amount required for fuel, and it often becomes a source of expense to take care of the remainder. The privilege of collecting fat light wood in the woods may usually be obtained at a cost of 10c to 25c per cord. Stumps can ordinarily be obtained for nothing, and if they are to be pulled out or cut off far enough below the ground so as to make the land tilable, the owner is often willing to pay \$8.00 or even \$12.00 per acre to have them removed.

The utilizable constituents of wood waste then are essentially the framework, which we call the celluloses for briefness, and the gum resins. The latter can be separated easily into turpentine, pine oil, and rosin. These are the only products existing in the wood in commercial quantities, or obtainable from it without using temperatures high enough to effect decomposition of the celluloses, rosin or oils.

The hardwoods contain no appreciable amount of gum resins, so the products derivable from them are, roughly speaking, only the products that could be derived from any other type of cellulose, for example they may be distilled to produce acetic acid, wood alcohol, tar and charcoal; they may be treated with alkali for conversion into oxalic acid; they may be treated with sulphuric acid for conversion into ethol alcohol. Some of them may be treated with soda, sulphite or sulphate for conversion into paper pulp. The yields in any or all of these cases, of course, will vary widely, depending on the nature of the hardwood.

The short-leaf pine contains a very small amount of gum resins, and also ordinarily contains more water than the hardwoods. The gum resins being in small amount are a disadvantage, because they contaminate the other products, and the yields both in quantity and in quality are liable to be inferior to those obtained from hardwoods, under similar conditions. So far as I know short-leaf pine waste is not being utilized at the present time, excepting for the production of ethol alcohol and paper pulp.

Long-leaf pine, Douglas fir and Norway pine may be classified together in this general way: If they are poor in resinous materials, then they can only be treated for the products obtainable from hardwood, and the products are liable to be inferior in quantity and quality, as stated under short-leaf pine. On the other hand, if they are rich in resins, this gum resin may be extracted direct and converted into any of the forms into which rosin, pine oil, and the terpenes are convertible, or the whole mass of wood may be treated together by distillation or otherwise, in which case we get various mixtures of products similar to those obtainable from hardwoods, with other products obtainable from the resins. This last method, unfortunately, has been the one most commonly adopted in the past, with the result that products of wood distillation too often have a reputation for indefinite and varying composition. In most cases it would be just as easy to separate the gum resins from the celluloses beforehand and to convert these separately into definite, uniform, marketable goods.

No account has been taken in this article of the mineral constituents of wood, nor of the various extracts which may be obtained from certain woods, like the oak and chestnut. The recovery of mineral constituents is not liable to become a factor in the near future, and extracts are not ordinarily produced from materials that would generally be classed as "waste wood."

The chief intent of this article has been to call attention to the wide variation of character in raw material presented, and the necessity of applying in each case the type of treatment which will be specially adapted to the character of the raw material.

New York City

The 1913 calendar of the **Brown Instrument Company**, of Philadelphia, Pa., shows a picture of the U. S. S. *Wyoming*, largest warship in the world, making 21 knots on her trial trip. There are Brown pyrometers used on the *Wyoming* for indicating the stack temperatures.

Mr. William E. Hartman, consulting engineer and expert in the construction of gas ovens and coke ovens, has been retained by the Gas Machinery Company, Cleveland, Ohio, with its oven department. Mr. Hartman was formerly associated with the H. Koppers Company, of Chicago, first as gas engineer in the construction of by-product coke ovens, and lately as managing engineer. A very interesting paper by Mr. Hartman on by-product coke ovens in America was published in our Vol. X, p. 521.

Sewage Purification

A Description of the Chemproco Sewage Disposal System

By James Millar Neil

Among sanitary engineers it is generally conceded that sewage consists practically of a mixture of 998 parts of water, one part of mineral matter and one part of organic matter, also that when reference is made to sewage purification, the same refers solely to this 1/10 per cent of organic matter which also includes the germ life.

Unfortunately, heretofore, no method in actual commercial operation eliminates completely all of this 1/10 per cent of organic matter contained in sewage, the difficulties encountered being the elimination of the soluble organic compounds and separation of the colloids from the treated sewage.

An ideal sewage purification process should possess the following advantages: The operating costs must be low; the plant required must have a low initial cost; the space required for the installation must be small; any reagents used must be low in cost; the resultant sludge must be easy of disposal; the effluent must be clear and harmless; the method of separating the solids from the liquid must be rapid and effective; under certain conditions the excess reagents in the sludge must be easily and cheaply recovered for use; the treatment operation and final products must be odorless.

The great defect in all present methods of sewage purification is that of filtration, and it is surprising to find how little ingenuity has been brought forward in solving this problem—every attempt at solution has been in the direction of sand filters, etc., with the corresponding technical difficulties pertaining to their satisfactory working and the enormous area of land required.

Sewage today can, however, be rapidly or, in fact, instantaneously purified, meaning thereby the complete separation of all of the solid matter contained therein, precipitation and separation of the organic matter, and the discharging of an effluent clear, odorless, and harmless. This result is obtained partially through the combination and application of the well known Moore suction filter, and a novel method of precipitation and separation of the organic and solid matters contained in sewage.

The operation of the process requires the use of suction pumps and vacuum filters, whose surfaces are permeable to liquids and impermeable to solids. The type of vacuum filter preferably employed is what is known in the arts as the Moore movable suction filter, as it allows of the separated solids being easily removed from the sewage treatment tanks at will.

By means of this process it will be found that not only are all of the conditions fulfilled for an ideal sewage purification process, but that its operation is simplicity itself.

The process consists essentially of the following steps or stages:

1st. The passage of the crude sewage through a screen or screens in order to remove the floatage and other large bodies contained therein.

2nd. Aeration in order to supply the amount of oxygen requisite for the complete precipitation of the organic matter and at the same time to maintain the contents of the sewage treatment tank in a state of agitation and thereby break up and granulate the solids, also for the purpose of commingling the precipitable and fatty matters with the precipitant so as to render them in a condition suitable to collection on the filtering medium employed.

3rd. Treatment of the sewage preferably while in a state of agitation by the requisite precipitating chemical reagents.

4th. Separation of the liquid from the solid matter in suspension in the treated sewage by means of filtration, instantaneously affecting the precipitation of the unprecipitated organic matter which heretofore has not been accomplished by any sewage purification process.

5th. Discharging of a clear, odorless and harmless effluent.

6th. Discharging a cake containing the impurities of the sewage and also containing the excess of chemical reagents used in the process.

7th. Recovery of the chemical reagents from the discharged cake for reuse in the process.

Before describing the individual steps or stages as they occur, a brief outline of what is meant by a Moore suction filter will facilitate the understanding of the operation of the process.

Fig. 1 is an illustration of a Moore filter leaf, which consists of a rectangular frame of tubing, enclosed by sheets of filter cloth suspended vertically from a rigid bar or header, the end sections or side tubes of the filter frame and canvas extending through the bar or header as shown in the illustration. The side tubes of the frame have no perforations, whereas the bottom pipe is usually slotted or channeled.

The filter cloths or canvas are reinforced by means of rows of stitches and stiffening or separating strips preferably of wood, which maintains the rigidity of the canvas or filtering medium when either pressure or vacuum is applied. As shown in the illustration, one of the side tubes is connected to a main

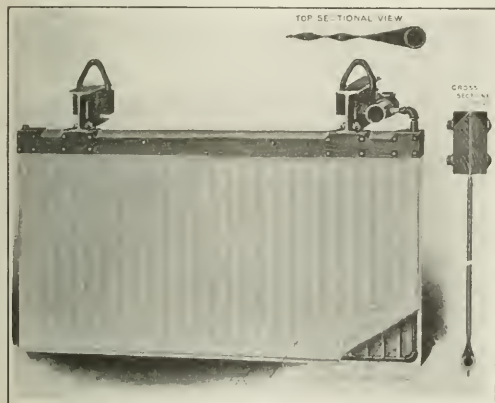


FIG. 1.—MOORE SUCTION FILTER LEAF

pipe or vacuum header connected by means of a flexible tube to the suction end of the vacuum pump. These leaves are made in any desired size, according to the filtering area required. This form of filter leaf gives an enormous filtering area, occupying very little space.

These filtering leaves are usually in practice coupled up in any desired number, and as such are known as a "basket." Fig. 2 illustrates a basket ready for use.

Fig. 3 illustrates a basket of leaves in place in a treatment tank, and a basket suspended out of the treatment tank by means of a traveling crane.

Fig. 4 is a side view of a complete sewage treatment plant, and shows the different steps of the process as set forth in the following description:

The operation of a Moore filter leaf is extremely simple. When the vacuum or suction is applied, clear liquid is drawn through the filtering cloths or canvas, depositing on the surface the solid matter in suspension in a cake of uniform thickness. The clear liquid in passing through the filtering medium is drawn through the slotted or channeled bottom pipe, thence through the side tube or pipe to the header, to the suction or vacuum pump.

In the operation of sewage purification, the initial step is that of preparing the chemical reagents used in the treatment tank, which is done by adding sufficient milk of lime until the tank contains an excess of lime, for example: 100 grains calcium oxide per gallon has been found to give excellent working results, and under certain conditions it may be advantageous to add other reagents such as calcium carbonate, sand, etc., as such reagents sometimes accelerate and improve the rate of filtration. In certain cases the addition of metallic oxides and hydrates also are found to greatly aid in the precipitation in the treatment of sewage from industrial plants.

The process of screening the sewage does not require any special or detailed description in this paper, as full details and data regarding the numerous types or methods at present in use can be obtained by reference to any standard treatise on sewage disposal.

The sewage to be treated is directed into the treatment tank, first passing through any approved system of screening in order

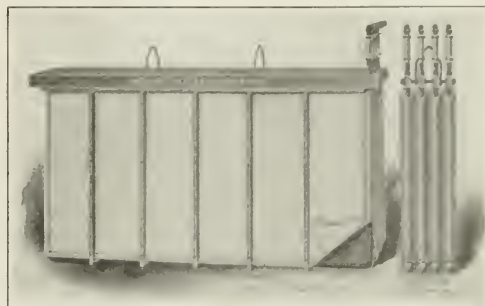


FIG. 2.—FILTER BASKET

to remove the floatage and other large bodies. The sewage on entering the treatment tank is agitated (and maintained in a state of agitation) for the reason explained, and into the treatment tank is introduced the requisite number of "baskets" of filter leaves which are submerged. Suction is then applied to the filtering mediums, whereupon a continuous flow of the agitated mixture is drawn toward the outer surfaces of the filter leaves, and on which the excess precipitant is deposited combined with the other solids in suspension. The liquid is then drawn through the film or cake of precipitant, thence through the filtering medium to the pump discharge.

By employing an excess of precipitant in the treatment tank, a film, layer or cake of the precipitant is instantaneously formed on the surface of the filtering medium, and it is found that the organic matter which heretofore it has been impossible to

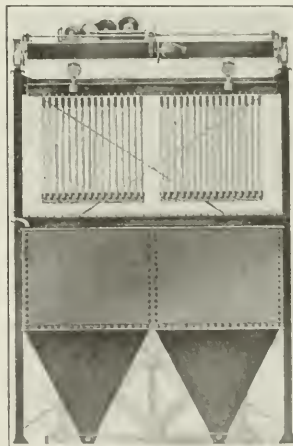


FIG. 3.—BASKET OF LEAVES IN PLACE IN TREATMENT-TANK

precipitate is thereby precipitated, and retained and combined with the other solid matter in the sewage adhering to the outer surface of the filtering medium. In practice it is found that even though this excess is present, the actual consumption of the precipitant is not greater than the amount actually required to effect the complete precipitation of the organic matter contained in the original sewage. In this simple manner the

complete precipitation of the foreign matter is effected instantaneously, resulting in the continuous discharge of a clear and harmless effluent.

The filtering operation may be continued until the deposit of solids on the surface of the filtering medium becomes so thick as to unduly interfere with the volume of liquid being drawn through, which will be indicated by the vacuum gauge and discharge from the suction pump, in which case the filtering medium or "basket" may then be raised from the treatment tank by means of the traveling overhead crane and moved to the dumping point; suction being maintained in order to hold the cake of solids on the surface of the filtering medium during this removal step. If it is desired to reduce the moisture in the solids, the "basket" may be held suspended in the air under suction before dumping.

In order to remove the cake of solids from the surface of

the accumulated solids from the treatment tank, which latter may be done in the manner already described.

If at any time during the treatment of the sewage the effluent shows any trace of organic matter, this can be at once remedied by adding fresh precipitant to the treatment tank.

Owing to there being a continuous flow of sewage into the treatment tank, which in some cases carries valuable ingredients, this latter method of operation results in the accumulation of these ingredients in a fairly concentrated and desirable condition for subsequent treatment, whereby if desired, the oils, greases, etc., can be economically recovered, and the residue used as a filler for fertilizer.

A special feature of this method of sewage purification is the destruction of harmful bacteria without the aid of any germicide. However, in exceptional cases hypochlorite will effect the complete destruction of all harmful bacteria; it is

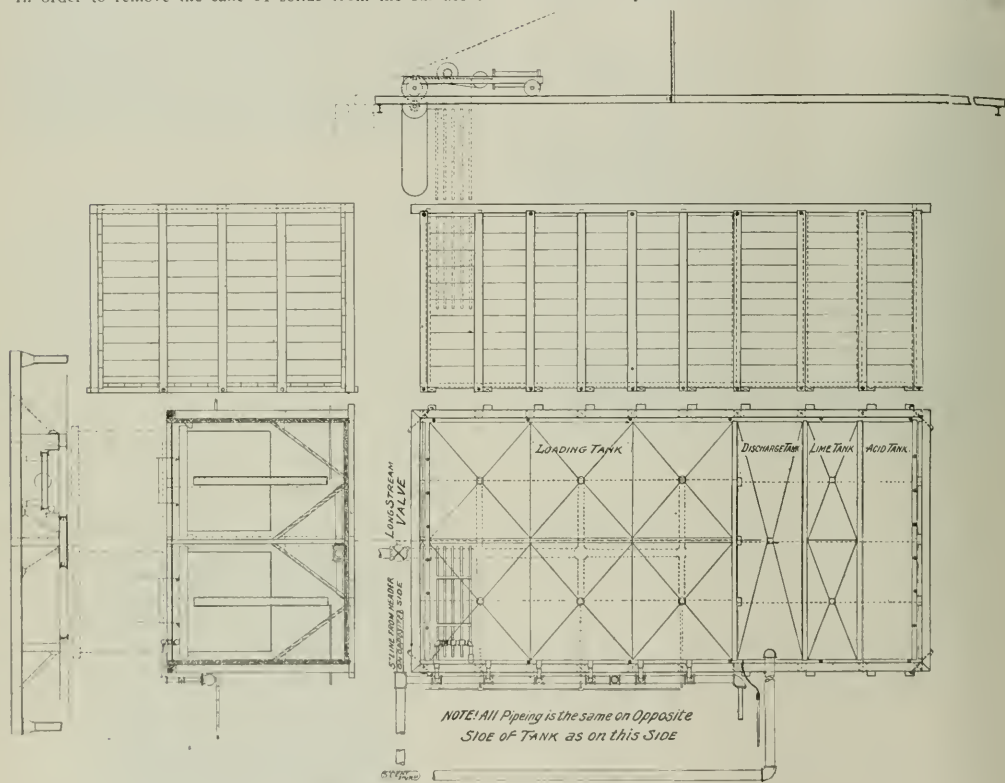


FIG. 4.—ARRANGEMENT OF FILTER PLANT FOR SEWAGE TREATMENT

the filtering medium, the suction is stopped and the cake is discharged therefrom by applying pressure by means of air or water within the filtering medium. After dumping or discharging the cake, the filtering medium of "basket" may then be immediately returned to the sewage treatment tank, and the filtering operation carried on indefinitely as already described.

Instead of raising and moving the filtering medium whenever it becomes coated the suction may be momentarily stopped, pressure applied and the solid matter thrown back or discharged into the treatment tank, and there become disintegrated on being subjected to the forces of agitation in the tank. After thus discharging the cake suction is again applied and the operation of filtration resumed; filtration continued until the surfaces of the medium are again sufficiently coated when the cake of solids may be discharged into the tank. This operation may be repeated until it becomes desirable to remove

preferably added to the sewage as it enters the treatment tank.

Numerous tests of the effluent obtained by this process from sewage rich in organic matter and harmful bacteria have shown the effluent in every case to be entirely free from organic matter and harmful germs.

Another unique method of operating the process, which it is sometimes found advantageous to employ, is to have a separate loading tank into which a "basket" of the filtering medium is first immersed for the purpose of forming a tum or cake of the precipitant thereon, by applying suction; when this cake has been formed on the surface of the filtering medium, the latter is then transferred and immersed in the treatment tank and the operation continued as previously described. This method is extremely effective in cases where wool scourers, tanneries, etc., add their wastes to the sewage.

Numerous demonstrations have proven that as soon as a

slight film of precipitant is formed on the surface of the filtering medium, a complete precipitation and separation of all the organic and foreign matter originally present in the sewage takes place and a clear, odorless and harmless effluent is obtained.

In operating the filters, the cake should be discharged in any of the various operating steps mentioned as soon as the vacuum gauge indicated a vacuum of 15 in., beyond which the rate of filtration rapidly diminishes.

The relative amount of moisture usually present in the discharged cake runs approximately 40 per cent, depending on the length of time the cake has been subjected to the drying action of the suction, while suspended in the air.

The chemical reagents used vary in accordance with the composition of the sewage undergoing treatment. In practice, sewage purification can be accomplished by the use of lime, copers, alum and caustic lime mud, the latter a calcium waste product in the alkali and paper trade.

In the construction of a plant wooden tanks have been found to be suitable, thereby doing away with the expensive iron or cement tanks at present used by disposal plants.

In comparison to the ground area required for installations of the different systems in use for sewage disposal, the great advantages of this process is clearly shown by the following figures:

Sand Filtration.—This system requires an area of one acre for about 30,000 gallons per 24 hours, operating under favorable conditions.

Continuous filters, the sewage having been previously treated either by chemical precipitation or by subsidence in settling tanks or in a septic tank, are assumed to be able to treat about 500,000 to 2,000,000 gallons per 24 hours.

The Chemproco (Chemical Process Company) system requires 230 square feet of surface area per 1,000,000 gallons per 24 hours. Besides this advantage, it is the only process delivering an effluent free from suspended matter, odorless and harmless.

The Chemproco system also possesses the following advantages over all others. It can be operated within the city limits without proving a public nuisance, and if desired it can be installed at the exits of the different sewers in cities where land values and local conditions render all other sewage disposal methods impracticable.

The process is operated by means of ordinary centrifugal pumps, driven by any desired motive power. The filtering medium with ordinary care will last from one to three years before requiring replacement.

No skilled labor is required: one man per shift can handle from one to thirty million gallons of sewage per day.

Preference is given to the recovery of the chemical reagents used in the Chemproco system from the discharged sludge. Experience has demonstrated that the regenerated reagents are not only more energetic in their precipitating qualities than the original chemical reagents, but that the rate of filtration is greatly accelerated.

The method of regeneration consists in calcining the sludge in a rotary kiln.

The operating cost of the Chemproco system ranges from \$5.50 to \$7.50 per million gallons of sewage.

Formastat is the trade name of specular hematite ore Fe_2O_3 , which after having been purified is sold as an imperishable paint pigment. It is particularly adapted for a paint for preservation of iron, steel, metals, woodwork, or any artificial surface exposed to the elements and the influence of destructive gases. The simplest formula for using it in making up 1 gal. of paint is 8 lb. formastat, 5 lb. pure boiled linseed oil, and $\frac{1}{4}$ lb. japan dryer.

Other formulas, as well as notes on application to aluminum, gold, and copper powders, together with general rules on the use of formastat, may be found in a pamphlet just issued by the manufacturer, the Formastat Mining Company, Twenty-first and Locust Streets, St. Louis, Mo.

Method of Producing Sound Ingots

Further Details of the New Process of Sir Robert Hadfield and Results Obtained with It in France.

In our November issue, 1912 (Vol. X, p. 738), we gave an account of a paper by Sir Robert Hadfield before the (British) Iron and Steel Institute on his new method for producing sound ingots.

We are obliged to Sir Robert Hadfield for the following information concerning the application of his method on a large scale.

By one plan the ingots are cast in molds placed upon bogies, movable trucks, or platforms, in the position A shown in Fig. 1. After casting the ingots vertically on bogies, the number on each bogie being as may be found most convenient, say four or more ingots on each bogie, advantage is taken of the time which elapses during transit of the ingots from the casting pit to the stripper to maintain the upper surface of the steel in a molten condition. This is effected by a series of air pipes, for example, three in number, with jets or holes on the under surface of such pipes, placed preferably horizontally arranged in such a manner as to cause jets of air to impinge at a suitable distance from the upper surface of the ingots. These holes or jets are made throughout the entire length of the overhead pipes, the length of such pipes being governed by the distance traversed under them by the bogies. In the figure shown the pipes are of sufficient length to extend over the tops of three rows of six bogies, each bogie containing four ingots.

One method of working is as follows:

As soon as possible after the ingots are cast, a quantity of charcoal sufficient for the purpose is placed on the top of each ingot, also a layer of insulating slag as described in the article referred to above. Although charcoal by means of the combustion effected by the heat of the molten steel delays to a certain extent the freezing of the molten steel, a blast of air applied to the charcoal generates a considerable amount of extra heat, this being of great advantage, as it helps to delay the freezing of the molten steel to such an extent that the good results before stated are effected in a manner which is impossible where such air blast is not used. The layer of slag, which may be termed an insulator, also prevents loss of heat and the charcoal is also prevented, or to a large extent so, from mixing with the steel, at any rate until too late to cause any interference with the quality of the product below the head portion.

If a series of bogies extending the length of the overhead pipes is used, the air blast may be working during the whole length of the piping, and if it should happen that only one or two bogies are in transit under the pipes it is not necessary for the air to be escaping throughout the entire length of the piping, but the air may be shut off in sections, and only allowed to escape in the section in which the bogies or ingots happen to be at the time being. Means may be provided for controlling the issue of air from each pipe to the ingots. Thus, each pipe may be formed at its lower side with a long slot or with a number of slots or apertures, and be provided with means, such as sleeves, which may be adjustable in a longitudinal or rotary manner, or in both a longitudinal and rotary manner, and by means of which the points of issue of the air to the ingot can be adjusted to suit requirements. Thus the air may issue between the adjacent ends of plain sleeves, or the sleeves may be furnished with suitably spaced nozzles, through which air can be caused to issue as required. As, however, it would be found that with normal working a continuous line of bogies would be passing underneath the air pipes—that is to say, there would be no break between the continuity of the bogies—the combustion, which is intensified and effected by the air jets on to the charcoal, would be effectively performed and carried out whether the ingots on the bogies were moving or at rest, owing to the continuity of the air holes or jets in the pipes which exert the pressure necessary for proper combustion of the charcoal being at what-

ever position the bogies and the molds may happen to be, provided they are within the range of the overhead air pipes.

It will thus be seen that if the passage of the bogies from the casting pit until they arrive at the stripper is relatively slow, the overhead air pipes need not be so long as in the case where the passage of the bogies from the casting pit to the stripper is at a more rapid rate, for it will be evident that in the latter case the overhead pipes would have to be of greater length than if the bogies were traveling under them at a slower rate.

Approximately the time that is necessary for the blast to operate on the upper surface of the molds, that is to say, from the time they are cast until the steel is sufficiently set to avoid any danger of piping, would be in the case of rail ingots from

casting for each individual ingot or for a group combined on one bogie or other suitable movable or stationary casting arrangement. The blast is continued as long as required; then the car or bogie is returned for further ingots to be cast.

It might be desired also to have a combination of bogies both worked electrically and by overhead pipes.

Fig. 1 shows the method of arrangement, which can be suitably varied to suit the local conditions. Although the sectional ingot shown is with the small end down, it can, of course, if preferred be cast with the small end up.

In Fig. 1 *A* is the plan of the casting department; *B* are drying stoves for heads, and *C* is a general plan of air pipes or blast arrangements, which are also shown in elevation by *D*.

Another method would be to make the casting plant circular

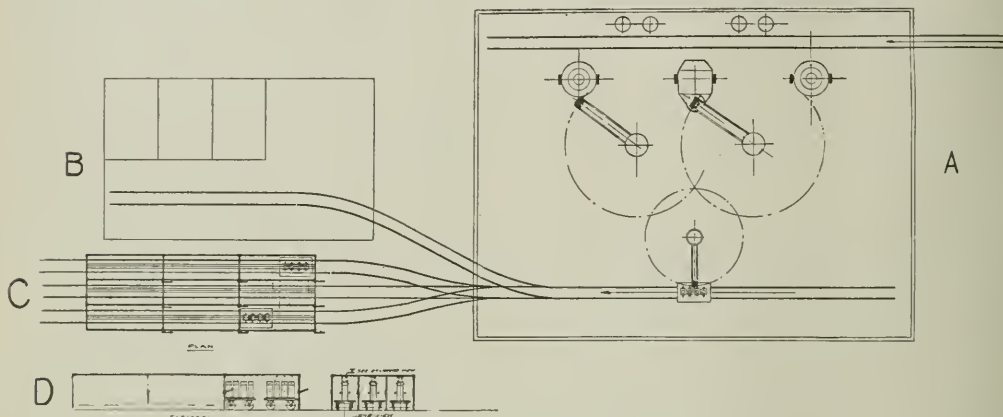


FIG. 1.—PLANS OF HADFIELD METHOD OF PRODUCING SOUND INGOTS

about fifteen to fifty minutes, according to the size of the ingot, though this time is only stated approximately. Also when the rate of progress is known at which the ingots move from the casting pit to the stripper—that is, the time from which the ingots are cast until they can be safely removed from their molds—the length of the overhead pipes can be easily calculated.

Three air pipes may also be used, these pipes being arranged at such a distance from each other as will enable the three rows of jets to play upon the molten steel which forms the heads of the ingots.

The height from the track to the upper side of the runner box should be approximately the same in every case, so that the air jets have the same heating effect on every ingot.

When the bogies arrive at the stripper the ingot molds are lifted and the ingots removed from the molds in the usual manner preferably retaining the ingots in a vertical position during the process of stripping. Considerable modification of this arrangement can be made as may be desired.

Ingots may be cast in this way with either their large or small end at the bottom, or with their large or small end at the top, whichever may be preferred, or practically parallel ingots may be produced of any size, section or shape which may be desired.

This method may be also applied to the production of castings of various types.

Another modification to effect the same object is by casting the ingot in molds upon bogies to which are attached electrically worked fans or blowers, the required electrical energy being taken from suitable overhead or underground cables. A blower of suitable type is mounted under the car at the side, or in such a convenient manner as may best suit the required purpose (see *F* Fig 2). As soon as the ingots are cast, the blower is put into operation, either while the car is standing still or as it moves along from the casting house. In this method a suitable supply of blast is obtained immediately after

—that is, the rails would be in a circular form outside the casting plant, and the bogies on which the ingots were cast could go around this if it were preferred to have the plant in this form.

In Fig. 2 *E* is a section of an ingot mold standing upon the bogie top, with air pipes in position at *X*; *F* the bogie arranged with the blower attached underneath as described.

The foregoing in a general way represents the methods of obtaining the desired advantages of preferably first obtaining sound steel which would pipe, then properly feeding the steel in such a manner as to give perfectly sound ingots with very little segregation portions, so that the double advantages are obtained of sound steel combined with much less waste than ordinarily experienced. Sound rails, bars or other forms are the result, with consequent advantage and increased safety to the user.

To obtain a sound and safe rail, the steel must itself be sound and free from blowholes. This requires the use of steel which will pipe or settle. As rail steel is now made, the maker has no proper means of adequately dealing with this property of fluid steel, namely, the "piping," often very deep, produced by "piping or settling steel." He therefore tries to avoid such steel, specially as it is dangerous unless the pipe is properly dealt with and fed, for the reason that in the methods hitherto generally adopted the pipe often runs down the center of the ingot for nearly one-half or even two-thirds of its length. By the Hadfield system piping itself is avoided, proper feeding being effected. Thus ingots are obtained in which soundness is ensured, that is to say, for example, in 10,000 ingots every one can be made sound and free from piping, blowholes and segregation.

It is suggested that this system should be thoroughly tested by the railroad companies in the following manner: Hadfield should supply a specified number of tons of ingots, say from 10 to 100 tons, as may be desired, made under this system; these should be rolled for the railroad, making the test by the

rail mill supplying the ordinary rails. Specimen rails selected from the batch would be tested by drop test or other mechanical means, in the manner carried out for the acceptance of rails under the present specifications.

Under the Hadfield system, too, the advantage is obtained of the investigator's long experience in alloy or special steels.

It is possible to produce, quite cheaply, special steel rails showing most excellent shock-resisting qualities, high dura-

size, weighing about 25 tons. Figs. 3, 4, and 5 show the results obtained.

M. Charpy stated that the Hadfield method of feeding the upper portion of the ingot in his special manner, namely sand head, air blast, charcoal, and insulating layer of slag, at-

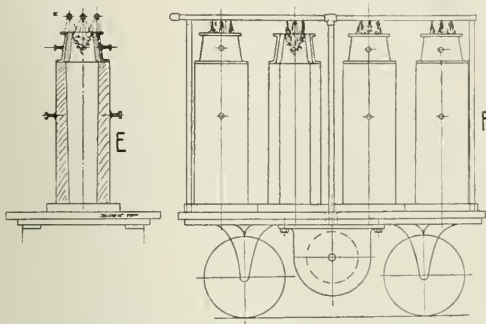


FIG. 2.—MODIFIED PLAN OF HADFIELD METHOD FOR PRODUCING SOUND INGOTS

bility and other advantages, and this at little extra cost, that is, it is unnecessary to use expensive and costly steels such as high nickel, manganese or other similar types.

It is estimated that by the Hadfield method the cost per ton of sound steel fusible ingots is even less than that of fusible ingots made by the ordinary method, because the extra cost of the Hadfield treatment is more than counterbalanced by the fact that the discard with the Hadfield method is only 8 per

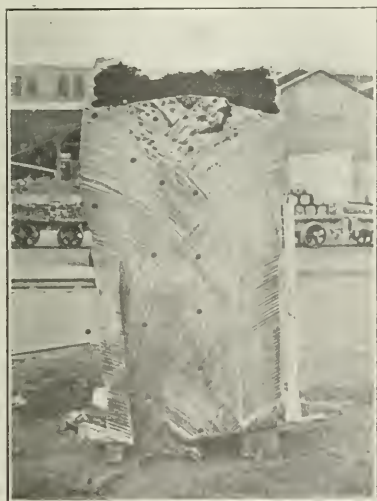


FIG. 3.—SECTION OF UPPER PORTION OF 25-TON INGOT

cent against 15 to 25 per cent with the ordinary method.

* * *

Quite unknown to Sir Robert Hadfield, the well-known French metallurgist, Monsieur G. Charpy, of the Cie des Forges et Acieries de Chatillon-Commentry et Neuves-Maisons, Montluçon (Allier), after reading the Iron and Steel Institute papers on this subject, decided to test the methods for himself, and proceeded to make an ingot of comparatively large

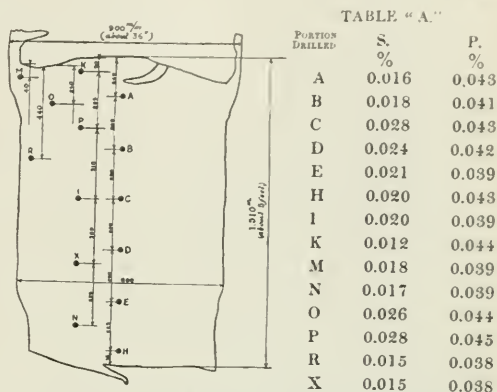


FIG. 4.—ANALYSIS OF INGOT AT DIFFERENT POINTS

peared to him to constitute a particularly simple and practical solution of the problem met with in steel manufacture, and to overcome the difficulties met with as regards piping, also reduce segregation and avoid the large percentage of waste material.

Fig. 3 represents a photograph taken by M. Charpy of the upper part of an ingot weighing 25 tons. This ingot had been treated by the Hadfield method, its composition being 0.40 per cent C, 0.02 S, 0.04 P, 0.45 Mn. The material also contained a small percentage of nickel.

Fig. 3 shows the satisfactory nature of the method. The portion of the ingot shown in the photograph represents about one-fourth of its total length. M. Charpy found that in order to eliminate all faults and sponginess in the ingot in question it was only necessary to cut off about 12 in. to 12½ in. from the upper part, in other words, about one-twentieth of the ingot, as against one-third which is the proportion of discard ordinarily required.

Moreover, the analyses taken of the material at different parts of the ingot showed this is free from segregation. Fig. 4 shows the parts from which the analyses given in table "A" were taken. Drillings were taken on the center line at the

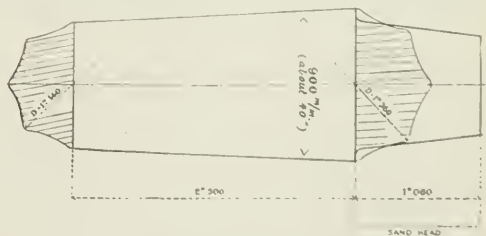


FIG. 5.—DIMENSIONS OF 25-TON INGOT

different parts shown, commencing at the top, marked "A," also at the other parts as lettered.

Fig. 5 shows the dimensions of the 25-ton ingot referred to. The waste and segregated material in this ingot only amounts to about 5 per cent of the total weight.

The original (British) Iron and Steel Institute paper of Sir Robert Hadfield on his process was published in our November issue, 1912, p. 738.

Some Notes on the Manufacture and Properties of Carbon Bricks.*

The resistivity of amorphous carbon against high temperatures, abrupt changes of temperatures, and not the least against chemical corrosion by ashes and slags make it especially valuable for manufacturing refractory material for smelting furnaces.

For this purpose carbon is used in form of coke which should be as much free from ashes as possible. The raw material, prior to molding, is made up in different ways. According to an older method, which is scarcely in use any more, the crushed coke is mixed with a small proportion of clay, this being in suspension in water. In the new method, now universally applied, the dried coke is crushed to grains of 1 mm. size, sieved and thoroughly mixed in the pug-mill with about 20 per cent in weight of heated tar.

The plastic mass thus prepared is filled, in successive layers, into wooden or iron molds, by taking care to create the upper surface of the last applied layer in order to secure a perfect bond between the different layers. In this operation care should be taken to fill the molds up completely, and to drive the mass into all the corners so as to obtain a brick with plain and even surfaces and sharp corners.

By following this precaution only a narrow joint will be left between adjacent surfaces of the bricks when laid aside in a wall. The drying of the wet bricks requires about two weeks, which is effected in the open air, after which they are ready to be burnt.

The bricks are then set in a producer-gas fired chamber-kiln and are enclosed in fireclay saggars. The spaces left in these saggars are then completely filled up with pulverized coke. The use of this filling is to protect the carbon of the bricks from any contact with air and combustion gases of the kiln, and to prevent combustion of the carbon.

In the average a burning temperature corresponding to Seger cone 12 is required. The tar content of the brick carbonizes during the burning and the additional carbon thus formed unites with the coke to form a strong and homogeneous mass. The specific gravity of carbon bricks varies from 1.5 to 1.9; the ash contents, in average, amounts to 12 per cent.

As stated above, these carbon bricks are exclusively used in metallurgical plants. They render excellent services for lining lead melting furnaces and other metallurgical furnaces where oxidation of the molten metal is to be prevented. Special attention has been given to carbon bricks in the construction of some parts of blast furnaces, where they are used for lining both walls and bottom. According to numerous reports carbon bricks showed in this instance a most satisfactory behavior, which, as in many other instances, depends mainly upon careful manipulation and treatment.

As few and as narrow joints as possible should be made in laying carbon bricks, otherwise the molten metal would enter the joints and cause loosening of the lining.

A mortar made up of pulverized coke with 20 per cent of good fireclay has been successfully used. This mixture, after being ground to a minute powder, is stirred up with pure water, i. e., which should be free of any soluble salts, to the consistency of a slip and this just before being used as mortar. It is essential for guaranteeing durability of the masonry to allow the same to dry slowly and thoroughly.

After all the water contained both in mortar and bricks is expelled, the furnace then may be put in service, otherwise mortar as well as bricks will be dislocated by a too sudden evaporation of the water. The solidity of the carbon brick masonry requires the application of a special and proper outside shell extending over the bosh and the hearth portions of the furnace.

The opinions about unlimited solidity of carbon bricks in metallurgical furnace practice differ, according to observers and circumstances. As to Dr. Ing. C. Geiger, in No. 41 of

Stahl und Eisen, 1912, carbon bricks showed especially good results in the production of high-carbon gray iron. Again, the fitness of this kind of bricks for foundry iron has repeatedly been confirmed; also successful results have been obtained with them in connection with Thomas iron.

These favorable reports have, however, been opposed by the testimony of other metallurgists who observed evident failures in several instances, so that the carbon bricks soon disappeared when in contact with so-called white pig iron and Thomas iron. A complete explanation of these failures has not yet been given, so far as the actual observations go. According to Mr. v. Schlippenbach this is probably due to the greater indirect reduction of the iron oxides by the carbon or that the iron with normally low carbon percentage readily combines with that of the carbon lining.

In order to elucidate this question further it would be a valuable undertaking for manufacturers of refractories as well as for metallurgists if accurate observations and systematic tests would be made and published.

Notes

The order in which the stamps drop in the Homestake mills is 1-3-5-2-4, or what is the same thing, 1-4-2-5-3. This order has been tested with other variations, such as 1-5-3-2-4 = 1-3-4-2-5, and has been found to give a slightly greater daily tonnage.

The extent to which gold penetrates copper plates in amalgamation has been investigated by Clark and Sharwood (Bulletin 98, Inst. Min. & Met.), who found that beyond a depth of 0.005 in. from the face of the plate nearly all the value lay in dents and pinholes. The percentage of gold found in layers next the surface, totalling about 1/64 inch, was 98 per cent. The value of gold contained in a square foot of outside plate was \$0.31; of a chuck-block plate, \$1.11.

Potash explorations in Nevada and Nebraska have disclosed deposits of potential value in both states. In Nevada the Silver Peak marsh was drilled and found to yield salts containing as much as 3.5 per cent K_2O . Certain shallow lakes in the sand hills of western Nebraska also contain potash in solution as well as in the surface of the lake beds.

Developed ore in the mines of the main reef series of the Witwatersrand is estimated at from 100,000,000 to 110,000,000 tons. At the present crushing capacity of Rand mills this ore constitutes a reserve sufficient to last for three years.

Protecting Exposed Metal Work in Battery Rooms.—No metal work in a battery room, especially over the cells, should be exposed to the action of the acid fumes and gases. The structural iron work of the building can be very well protected by encasing it with expanded metal work upon which a coating of cement plaster is placed. This if kept well painted with a good lead paint will prove a very satisfactory way of protecting all structural metal work. Pipes and small portions of metal work can be protected by sheet lead, or if they are not in too close proximity to the cells can be protected sufficiently by frequently cleaning and painting with an acid resisting paint such as Hydrex preservative paint. This paint is very thick and heavy, and forms a strong, heavy protective coating, and does not contain any coal tar to corrode the metal. It is claimed to be acid, gas, and sulphur fume-proof. The battery rooms in all new electric sub-stations of the New York Central Railroad are painted with this preservative paint. One great danger of exposed iron work in a battery room if it extends over the battery, is that the sulphate of iron formed by the action of the fumes may fall into the cells and impair the purity of the electrolyte, thereby seriously damaging the plates.

The *Biddle Bulletin* is the title of a new trade publication issued by the Biddle Hardware Company, of Philadelphia, Pa., to give information on new developments in the Monel metal market.

* Translated from the *Thomson Zeit.*, by Leon V. Schleich.

Ore Sampling Without Machinery

A Series of Instructive Illustrations
Explaining the Essential Feature
of Ore Sampling. By Jesse Scobey

CORRECT sampling is the basis of ore valuation; an assay can represent only the proportionate value of the sample tested. These two propositions are recognized as fundamental by competent technical men, and yet occasional evidence is found which indicates that these principles are violated even by those who should know better, not to mention those whose lack of training and experience lead them into costly errors. Perhaps the self-evident nature of the requirements of correct sampling is one reason why comparatively little attention has been paid to the subject in technical literature. Hence no apology is required for reiterating some facts which cannot be too strongly

mixed with the proper amount of suitable fluxes, and smelted in a sufficiently large furnace, it would yield exactly 10 oz. of silver, about the size of six silver dollars. This would be a perfect assay; but manifestly it is impossible to work on so large a scale, and the practice of assaying limits the amount of ore used to an assay-ton, which is a quantity slightly more than 1 oz. It is, then, essential that this small amount of ore should be exactly representative of the ton. By delicate balances the actual amount of silver smelted from the amount of ore used is weighed, and by proportion the amount per ton is calculated. The whole secret in the proper valuation of a ton



FIG. 1.—A PILE OF 400 LB. OF ORE



FIG. 2.—HAMMER CRUSHING

borne in mind. An assay is worthless unless the sample is representative, as many a man has learned with regret.

It frequently happens that laymen who are interested in mines, or who desire to investigate mining propositions in a tentative way, resort to the practice of taking their own samples for assay. The results may be disappointing or highly encouraging, but are worthless unless the sampling has been performed properly. If the ultimate result is unsatisfactory, it is customary to claim a discrepancy in the assay without realizing that the error possibly lies in the sampling. This is an old subject, well understood by some, but may be repeated for the benefit of those not familiar with the technique of mining.

In order to appreciate the relation of assaying to correct sampling, let us consider briefly the conditions surrounding the former operation. In assaying, the results are reported in ounces of gold or silver per ton of ore; that is, a ton of the ore in question would contain one or several ounces of gold or silver. With silver worth 64 cents per ounce, an assay of 10 oz. silver per ton means that a ton of such ore would contain silver to the value of \$6.40. If a ton of this ore should be

of ore is, then, simply to get a correct sample of it, weighing not more than a few ounces.

Sampling without machinery¹ requires not only manual labor, but the exercise of care in order to approximate the accurate results which can be obtained with mechanical appliances. If a ton of ore can be sent to a testing or sampling works equipped with crushers and samplers, the work is simple indeed. Many mine operators, engineers and others engaged in mining have not these facilities, and do not care to incur the expense of sending ore to the testing works. In order to accomplish the same result by hand-sampling there need be only a full realization of the fact that the ore must be broken finer and finer at each successive stage of the process, as the remaining sample becomes smaller.

The illustrations herewith will convey the idea better than a lengthy description. The pictures show the successive steps in sampling a lot of crude ore.

Fig. 1 shows a pile of 400 lb. of ore. It was broken from a

¹Brunton has described and illustrated the possible errors in coning and quartering. *Transactions A. I. M. E.*, Vol. XL, p. 567.

vein, the value of which is to be determined by assay. When a representative sample is desired, it is inadvisable to start with less than 400 lb. of ore.

Fig. 2. Using a sheet canvas, an anvil or heavy piece of iron and a heavy hammer, break each piece of ore to less than

pile or bed several inches thick. Then throw one shovelful to the "sample" pile and three shovelfuls to the "reject" pile, making a sample which shall be one-fourth the size of the original pile. Each piece of ore in the "sample" should be one-fourth or less of its original size.



FIG. 3.—SHOVELING INTO A CONE



FIG. 5.—CRUSHING SAMPLE WITH HAMMER TO NUT SIZE

fist-size. Scatter the pieces well over the canvas to be sure that all are broken.

Fig. 3. With a square-pointed shovel throw all the ore into a central pile or cone, dropping each shovelful precisely in the

Fig. 5. We now have about 100 lb. in the sample. Take a hammer and again break each piece of rock to less than nut-size. Thoroughly spread and mix this portion as before.

Fig. 6. The sample is again carefully coned, and flattened by



FIG. 4.—SAMPLE PILE OF 100 LB. AND REJECT PILE



FIG. 6.—FLAT CAKE

center. This is a precaution that must be observed in order to distribute the coarse and fine ore uniformly on the pile.

Fig. 4. By working around the pile with the shovel, draw the center to the sides in order to mix evenly the fine and coarse particles. The cone is thus transformed into a flat

tamping down the top. This gives a flat cake which can be marked into quarters by means of a board and shovel.

Fig. 7. By shovelling away opposite quarters we leave a sample weighing about 50 lb., while the 50 lb. of "reject" can be discarded.

Fig. 8. The same operation described under Fig. 5 must now be repeated, breaking each piece to pea-size or smaller, spreading the ore over the canvas to be sure that all pieces are broken.

Fig. 9. Shovel the sample into a cone again, carefully clean-

at the left on the table; reject one portion and again divide the other. One portion of this last division will weigh about 2 or 3 lb.

Fig. 12. Repeat the grinding operation shown in Fig. 11 until the sample will pass an 80-mesh screen. Divide the sample



FIG. 7.—REDUCING SAMPLE TO 50 LB



FIG. 9.—CONING AGAIN

ing the canvas with a broom so as to be sure that no fine ore is lost.

Fig. 10. Flatten the cone, mark into quarters, and with a small fire-shovel remove opposite quarters. Remix the remain-

twice as before, when the final portion will weigh about 8 oz and will fairly represent the original 400 lb. This is a safe and sane sample for the assayer.



FIG. 8.—BREAKING TO PEA SIZE



FIG. 10.—REDUCING SAMPLE TO 10 OR 12 1/2 LB

ing ore; cone and quarter again. The sample will now weigh about 10 or 12 1/2 lb.

Fig. 11. This sample must now be ground on a bucking-board until fine enough to pass a 10-mesh screen (openings about 1/20 in.). Pour this fine ore through a divider, shown

The procedure just described may be subject to modification for different grades of ore, richer ores requiring finer subdivision in the successive steps so that the whole mass may be more intimately mixed. The following table shows the maximum size of particle that should exist in the different portions

as the quantity retained for the sample decreases. Column A refers to a medium-grade ore; column B to a high-grade ore, and column C to a very rich ore.



FIG. 11.—REDUCING SAMPLE TO 2 OR 3 LBS.

Quantity of ore to be sampled.	A Size.	B Size.	C Size.
100 tons to 10 tons.....	3 1/4 in.	3 in.	2 1/2 in.
10 tons to 1 ton.....	2 1/2 in.	1 1/2 in.	1 1/4 in.
2000 lb. to 200 lb.....	1 in.	3/4 in.	3/2 in.
200 lb. to 5 lb.....	5/16 in.	3/16 in.	5/32 in.
5 lb. to 8 oz.....	1/32 in. or 16-mesh screen	1/64 in. or 30-mesh screen	1/164 in. or 80-mesh screen



FIG. 12.—REDUCING SAMPLE TO 8 OUNCES

Consideration of the foregoing will show that the cost of concentrations and accurate sampling is greater than is sometimes conceded to be a reasonable charge. Engineers and laymen should recognize the importance of a properly taken sample and should insist upon it; otherwise there will be a mis-

conception of the value of the ore, and dissatisfaction when it is shipped to the testing works, mill or smelter.

A "mill test," so-called by the old-time prospector or miner, is nothing more than the shipment of about 400 lb. of ore to an assayer who, with proper facilities, crushes and samples the lot to obtain a representative portion for assay. A common charge for this work is from \$1 to \$5. Anyone can do it in the field by following the above procedure, realizing that time, labor, and care are essential.

The Henry E. Wood Ore Testing Company, Denver, Col.

An Application of the Electric Resistance Furnace to the Determination of Oxygen in Iron and Steel

By R. H. McMillen

The fact that iron and steel always contains more or less oxygen has long been known, and about thirty years ago Ledebur¹ called attention to it and gave a method for its determination. It is only recently, however, that the requirements in the manufacture of high-grade steels have become so exacting that the determination of oxygen in steel and other materials has come to be one of the routine determinations required of a steel laboratory.

The Ledebur method, which is well known, consists in heating the sample of iron or steel in nitrogen to remove all moisture without oxidizing it, then reducing the oxides at a red heat by hydrogen and absorbing and weighing the resultant water. Cushman² has shown that the drying of the sample in nitrogen is unnecessary, his results being but slightly higher than those by the original method. When used with electric resistance furnaces, this method is very satisfactory for the determination of oxygen in iron and steel, tungsten³, and other non-volatile metals. Even this method, however, will fail to show all the oxygen in metals containing oxides of silicon, vanadium, titanium, and other elements whose oxides are not reduced below 950 deg. C.⁴

The following modification of the Ledebur method has been found to give most satisfactory results:

Apparatus.

The apparatus consists of two electric resistance furnaces, such as are employed in many steel laboratories for combustion carbon determinations and capable of maintaining a temperature of 950 deg. C.

They are designated as No. 1 and No. 2 in Fig. 1. Both furnaces are equipped with heavy walled, fused quartz tubes 7/8 in. inside diameter by 24 in. long. The function of furnace No. 1 is to heat the sample under investigation, while that of No. 2 is to heat the hydrogen so that it may combine with any oxygen that may be carried over from the hydrogen generator. Several spirals of platinum gauze are placed in the quartz tube of this furnace.

The quartz tubes of these furnaces are connected in the rear by a U tube (not shown in the illustration) having close fitting glass stop-cocks. This U tube is filled with phosphorus pentoxide opened by small glass beads. The phosphorus pentoxide absorbs any water which may be formed in tube of furnace No. 2, and insures absolutely dry hydrogen to pass over the sample in furnace No. 1. The front end of the quartz tube in No. 2 furnace is connected to the drying and purifying train leading from a hydrogen generator.

No. 3 contains small pieces of pumice stone saturated with concentrated sulphuric acid, No. 4 soda lime, No. 5 stick potassium hydrate, No. 6 50 per cent solution potassium hydrate, No.

¹Sauerstoffbestimmung im schmelzbaren Eisen. Stahl und Eisen, Vol. 2, page 193.

²The Determination of Oxygen in Iron and Steel. Journal Ind. and Eng. Chem., Vol. 3, page 372.

³Tungsten powder often contains a rather large percentage of oxides. Some commercial samples investigated by the writer recently have shown an oxygen content corresponding to 12% WOs. It is probable, however, that the whole of the oxygen is not combined with the tungsten.

⁴See The Determination of Oxygen in Iron and Steel by Reduction in an Electric Vacuum Furnace by W. H. Walker and W. A. Patrick. Met. and Chem. Eng'g, Vol. 10, p. 629. Original Communications of Eighth Internat. Congress of Applied Chemistry, Vol. 21, p. 139.

7 25 per cent solution of pyrogallol made alkaline with potassium hydrate.

Hydrogen is generated in a sixty-four ounce Kipp apparatus by the action of chemically pure hydrochloric acid on pure mossy zinc.

The water formed by reduction of oxides in the sample is absorbed in U tube No. 9 of same construction as No. 8, and also filled with phosphorus pentoxide opened with small glass beads. A guard tube, No. 10, is attached to No. 9 and contains concentrated sulphuric acid. A small wash bottle of Drexel form is used for this purpose. Porcelain boats of sufficient size to hold at least twenty-five grams of the sample are employed.

Preparation of Sample

Great care must be taken in preparing the samples so that they be free from rust or foreign oxides. In the case of iron or steel samples both the fine and coarse drillings are rejected, only those between twenty and thirty mesh being used. These

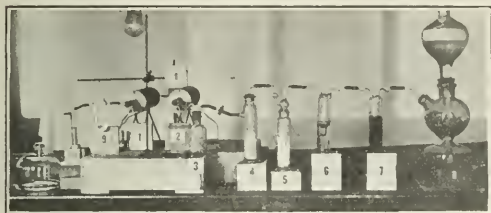


FIG. 1.—APPARATUS FOR DETERMINATION OF OXYGEN IN IRON AND STEEL BY A MODIFIED LEDEBUR METHOD

are dried for at least one hour over concentrated sulphuric acid before using. Samples of tungsten powders are dried to a constant weight in the drying oven at a temperature of 105 deg. C.

Procedure

Twenty-five grams of the prepared drillings are weighed into the dried porcelain boat and placed in tube of furnace No. 1. Connection is made to weighed U tube, No. 9, through which, just previous to drying for fifteen minutes and to weighing, hydrogen has been passed for ten minutes. Guard tube No. 10 is connected. Hydrogen is allowed to pass through the whole apparatus at a rapid rate for fifteen minutes. For the first run it is advisable to allow the hydrogen to sweep through the whole apparatus for thirty minutes to insure complete expulsion of oxygen from the train and tubes. The train and quartz tube of furnace No. 2 can be kept constantly filled with hydrogen by closing cocks of U tubes No. 8 whenever the flow of hydrogen is interrupted.

The electric current is turned on and continued for one hour after the maximum temperature has been reached. During this time the flow of hydrogen is cut down to about seventy bubbles per minute, this rate being maintained until U tube No. 9 is removed. At the end of the hour the current is turned off and furnaces allowed to cool, accelerating the cooling with a blast of compressed air. When the quartz tubes in the furnaces show no visible redness, cocks of U tube No. 9 are closed and a pinch cock is inserted over the rubber tube connecting U tube No. 9 to the quartz tube of furnace. In this manner oxygen is kept from the heated tube thereby avoiding danger of explosion. U tube No. 9 is disconnected and carefully wiped with soft dry cloth then desiccated over concentrated sulphuric acid for fifteen minutes and finally quickly weighed. Weight of water, after deducting blank, multiplied by 0.8888 divided by 25 equals oxygen.

A blank should be run frequently, adhering to all details as to time of heating furnaces, desiccating U tube, etc. Usually the blank found varies between 0.0015 and 0.0025 gram.

For samples of tungsten powder the same procedure is carried out except that a smaller sample is taken varying from one to ten grams according to amount of oxygen present.

The following table gives results on several samples in duplicate by the above described procedure

No.	C	Fe	Mn.	Sul.	Phos.	Si.	O.	Dug. i. cate.
1	Crucible steel	1.15	.31	.018	.011	.21	.03	.037
2	Crucible steel	1.17	.31	.015	.010	.22	.03	.035
3	Crucible steel	1.14	.33	.016	.009	.2	.044	.045
4	*Basic open-hearth steel	.07	.06	.019	.008	.18	.113	.115
5	Basic open-hearth steel	.08	.22	.016	.008	.21	.172	.17
6	Swedish wrought iron	.06	.17	.019	.012	.1	.345	.353
7	Domestic wrought iron	.69	trace	.009	.007	.06	.069	.76
8	Domestic wrought iron	.73	trace	.009	.007	.06	.090	.89
9	Acid open-hearth steel	.36	.66	.040	.046	.03	.641	.042
10	Bessemer steel	.46	.72	.041	.095	.09	.658	.060
11	Tungsten powder	..	..	..	..	..	2.27	.57
12	Tungsten powder	..	..	..	..	..	1.37	1.34
13	Tungsten powder	..	..	..	..	..	.55	.56
15	Tungsten powder	..	..	..	..	..	..	1.45

Numbers 1, 2 and 3 were three ingots made under as nearly the same conditions as possible. It is not intended that the above table should be typical as to the oxygen content that exists in the different classes of steel. In many samples of crucible steel it is much lower than those cited.

*This sample fractured badly in rolling.

Crescent Laboratory,

Crucible Steel Company of America.

Swedish Methods of Packing Ferrosilicon

Two Swedish manufacturers of ferrosilicon export their product to the United States. Their methods of packing the product have been secured by Consul Stuart J. Fuller, Gottenborg, and transmitted to the United States Bureau of Commerce and Labor. The method of the Gullspanns Elektrokemiska Aktiebolag are as follows:

"For packing we principally use iron-bound wooden cases in two sizes, one (for 50 per cent ferrosilicon) containing about 130 kilos (287 lb.) and one (for 75 per cent) holding about 110 kilos (353 lb.). The boxes have always been unlined and were made in the beginning of un rabbeted $\frac{5}{8}$ -in. stock, but since last year we have employed rabbeted stock, as the loss of contents in transit was found to be too great with un rabbeted boards. Water-tight packing being required for Germany, we pack goods for that country in second-hand kerosene barrels; these are, of course, both water and gas tight, but no fault has been found with them. On all our very latest shipments we have, at the request of foreign sellers, begun to pack ferrosilicon in sheet-metal cylinders containing about 170 kilos (375 lb.), and have made four holes of 7 mm. (0.276 in.) diameter in the top to provide against excessive pressure in the cylinders.

"It will be seen from the foregoing that the regulations in force are different for different countries (e. g., for transport on some English railways each package must have three 1-in. holes to permit the escape of any gases that may form). In our opinion, the dangers connected with the transportation of ferrosilicon can never be entirely avoided until uniform international regulations are adopted. In this connection it should be noted that danger is present only when the material has a silicon content between 35 and 65 per cent; under and over these proportions ferrosilicon is much more stable. So far as we are informed, accidents have never happened except with ferrosilicon of about 50 per cent silicon content. If, as we hope, an international regulation regarding packing and transportation of high percentage ferrosilicon can be established, regard should be had for the last-mentioned fact, and different regulations adopted for 25 per cent, 50 per cent, and 70 per cent goods or those of higher percentages.

Four Styles of Package Used by Another Firm

"The Aktiebolaget Heroult's Elektriska Stål thus describes in detail its methods of packing ferrosilicon:

"Until recently we used wooden barrels as containers for ferrosilicon. These barrels consisted of:

"(a) Old oil barrels and tar barrels, etc., holding about 500 kilos (1100 lb.) of 50 per cent ferrosilicon. These barrels we would describe as tight with exception of the cover, which is made of rough boards laid edge to edge without rabbets and held together by two laths. Across the cover are then nailed

two pieces of hoop iron, the ends of which are turned down and nailed to the sides of the barrel.

"(b) Wooden barrels, which hold about 700 kilos (1543 lb.), shaped as cylinders and made of 1-in. rough and un rabbeted boards, which have the edges cut aslant so that they fit or overlap each other in the sides of the barrel. The bottom and the head are made as described under (a).

"(c) The last few months we have used barrels taking about 300 kilos (660 lb.) and made in the same way as those described under (b), but of $\frac{3}{4}$ -in. boards instead of 1-in.

"(d) Barrels holding about 300 kilos (660 lb.) and consisting of cylinders of 0.6-mm. (0.0236-in.) sheet iron. The rolled or folded seam is reinforced with rivets. The head and bottom are made of wood, as described under (b), but of rough boards, and are placed so far inside the cylinder that the rims of the same can be turned over the bottom and cover about $\frac{1}{4}$ in. Head and bottom are fastened by nails driven through the wall of the cylinder and into the wood.

"We use only barrels, which can be rolled and therefore are more convenient to handle when moved than would be the case with another form of package—boxes, for instance. Besides this, the round form is stronger with the same thickness of walls.

Advantages and Disadvantages.

"With reference to the packing according to (a) and (b), these packages proved too heavy to handle with ease, and therefore we changed to packing (c). As, however, the packing (c) costs more for a certain quantity of ferrosilicon than the packings (a) and (b) and also increases the weight of the tare and consequently the freight that has to be paid per ton of iron, we have during the last few weeks partly adopted the packing (d).

"The forms of packing (a), (b) and (c) have this disadvantage, that when the packages have been stored for some time the wood has dried and shrunk, causing cracks between the staves as well as between the staves and the head or bottom, and these cracks have been large enough to let some of the ferrosilicon powder fall out. Some ferrosilicon has always a tendency to fall apart into powder or dust, and such powder is also formed when the packages are handled. Therefore, when the barrels are rolled, chiefly in transit, some of the pulverized ferrosilicon can fall out through the cracks, causing disagreeable controversies between seller and purchaser concerning the net weight.

"We consider that this inconvenience is most easily avoided by using the packing (d), for in this the walls do not allow any powdered ferrosilicon to escape, and the rims of the cylinder, turned over the edges of head and bottom, also prevent loss of powder between the wood and the metal wall of the cylinder. According to our opinion, this advantage is obtained with packing (d) without preventing ample ventilation in the barrels. Should it, contrary to our supposition, be shown that this ventilation is insufficient, holes can easily be made in the center of head and bottom, and these holes can be covered with wire-cloth, perforated metal sheets, or strong bagging."

Tungsten-Molybdenum Thermocouple.—In Dr. E. F. Northrup's article in the January issue, page 45, the sentence beginning in the eighth line of the second column should read: "At 1000 deg. C. (instead of 100 deg. C.) the oxidation is negligible, etc."

Cement Floor.—"The best hardening preparation for a cement floor is composed of equal parts of care, cement, and water used when the floor is built," states Leonard C. Wason, president of the Aherthaw Construction Company, of Boston, in a recent paper. "Various advertised products have been tried out, and there is considerable divergence of opinion. Certainly none of them have had uniform success, as could hardly be expected, under varying degrees of intelligence in their handling. So far as our experience goes we do not know of anything better than a floor made with properly selected stone and cement, laid intelligently and properly protected."

Specifications for Open-Hearth Steel Blooms, Billets and Slabs for Forging Purposes.

The Association of American Steel Manufacturers has just issued a set of "Standard Specifications for Open-Hearth Steel Blooms, Billets, and Slabs for Forging Purposes."

The need for standard requirements governing the chemical and other properties for forging billets has been apparent for some time and the new specification may be expected to be used widely in this field. A feature that is distinctly new is the table of allowable variations on check analysis of carbon, manganese, and other elements. In giving this table out for general use the Association of American Steel Manufacturers is taking a bold step toward standardization. Mr. A. A. Stevenson is the president and Mr. Jesse J. Shuman, care of Jones & Laughlin Steel Company, Pittsburgh, Pa., is the secretary of the association.

The new specifications are as follows:

I. Manufacture.

1. *Process.*—The steel shall be made by the open-hearth process.

2. *Discard.*—A sufficient discard shall be made from the top of each ingot to secure freedom from injurious piping and undue segregation.

II. Chemical Properties and Tests.

3. *Chemical Composition.*—The steel shall conform to the following requirements as to chemical composition:

Element.	Class.	Least Range to Be Specified.	Allowable Variation on Check Analysis Outside of Range Specified.	
			Under.	Over.
Carbon	Up to .10, .11 or .12	...	...	.02
	.10 to .25	.05	.02	.05
	.26 to .60	.10	.02	.05
	.61 to .90	.15	.02	.08
Manganese	Up to .60	.20	.05	.05
	.61 and over	.30	.05	.05
		Least Maximum to Be Specified.		
Phosphorus		.040	...	.010
Sulphur		.045	...	.010

4. *Class.*—The class shall be determined by the upper limit specified for carbon and manganese.

5. *Ladle Analyses.*—To determine whether the material conforms to the requirements specified in section 3, an analysis shall be made by the manufacturer from a test ingot taken during the pouring of each melt. If requested, a copy of this analysis shall be given to the purchaser or his representative.

6. *Check Analyses.*—Check analyses may be made by the purchaser in accordance with the standard methods of sampling for check analysis adopted by the Association of American Steel Manufacturers, and these analyses shall conform to the requirements specified in section 3.

III. Workmanship and Finish.

7. *Chipping.*—Chipping shall be done in such a manner as not to cause any imperfections when the billets, blooms and slabs are properly forged.

8. *Finish.*—Billets, blooms and slabs shall be free from all injurious defects.

IV. Marking.

9. *Marking.*—The melt number shall be legibly stamped on each billet, bloom and slab.

V. Inspection and Rejection.

10. *Inspection.*—(a) The inspector, representing the purchaser, shall have free entry at all times while work on the contract of the purchaser is being performed, to all parts of the manufacturer's works which concern the manufacture of the material ordered. The manufacturer shall afford the inspector, free of cost, all reasonable facilities to satisfy him that the material is being furnished in accordance with these specifications.

(b) The purchaser may make the tests to govern the accept-

ance or rejection of material in his own laboratory or elsewhere. Such tests, however, shall be made at the expense of the purchaser.

(c) All tests and inspection shall be so conducted as not to interfere unnecessarily with the operation of the works.

11. *Rejection*.—Billets, blooms and slabs which show injurious defects while being finished by the purchaser may be rejected, and the manufacturer shall be notified.

12. *Rehearing*.—Samples tested in accordance with section 10 (b) which represent rejected material, shall be preserved for one month from the date of the test report. In case of dissatisfaction with the results of the tests, the manufacturer may make claim for a rehearing within that time.

Recent Advances in the Spray Process for the Production of Metallic Coatings

By M. U. Schoop

In spite of the extraordinary simplicity of the principle which underlies the spray process for the production of metallic coatings, which I described in an article in this journal in July, 1910 (vol. VIII, p. 404), careful experiments extending over several years were necessary to transform the original idea of the invention into a practically successful process.

In order to be practically successful, the apparatus must be capable of being started or stopped at any time within a few minutes and the necessary materials must be easily procurable anywhere. Further, the apparatus must be easily movable so that even objects of large dimensions, wherever they may be located, can receive a cheap and durable metallic coating.

This goal has now been reached. Coatings of all metals, of the easily fusible tin and lead, as well as of copper, iron, platinum, can be made with the spray process.

The first experiments were made with a Flobert rifle. Experiments with small cannons followed whereby powdered lead or tin was sprayed. In some of the experiments the charge of ground metal in the cannon was carefully heated and melted by means of an alcohol lamp. Though these experiments were rather primitive, they showed, nevertheless, that the problem could be solved in two ways. Either molten metal could be sprayed or metallic powder could be used if only the powdered metallic particles were given a large kinetic energy. For this reason the first patents alluded expressly to the use of either fluid metal or metallic powder (Swiss patent 49,278, of November 26, 1909).

Spraying Liquid Metal

It is evidently difficult to design an apparatus in which metal is molten and sprayed so that it can be easily and quickly put into operation. It is even more difficult to make it movable, while with a stationary apparatus the application of the process would be considerably limited.

It is not difficult to build apparatus of this kind on a small scale so that large molten baths are avoided. But such apparatus involve essential and inherent difficulties, of which one of the most troublesome is the tendency of the molten metal to alloy with the crucible and the spray nozzle. The formation of alloys seems to be aided considerably if the molten bath is under pressure and if high temperatures are used. With easily fusible metals, crucibles of soft steel can be used, while with aluminium, brass, and copper only graphite or carbon fulfills to some extent the requirements. The spray nozzle cannot be made of copper nor of metal.

We succeeded in overcoming these defects by employing as container for the molten metal a closed armored crucible, to the interior of which either a pressure or a vacuum may be applied at will by simply turning a handle. If the molten bath is put under pressure the molten metal is ejected in form of a capillary uniform filament, while when a partial vacuum is applied the ejection of metal stops. This pneumatic device has the advantages that the apparatus can be started or stopped at any time. My French patent 411,100 contains a diagram which illustrates the operation of this apparatus.

This apparatus was the final result of a large number of experiments and undoubtedly represented a considerable advance since it was now possible to spray and deposit metals at any melting point. These pneumatic melting crucibles, however, are not particularly easy to handle or to transport.

In this article it is not possible to describe in detail the experiments which have been made in my Zurich laboratory with this melting crucible. It may only be stated that the conditions of operation, especially the amount of pressure and vacuum in the crucible, the design of the spray nozzle, the pressure and the temperature of the metal to be sprayed, etc., must be different for different metals. For every metal the application of the process must be worked out separately.

A characteristic feature of the metallic coatings which are obtained by spraying liquid metal seems to be that the metal increases in hardness and loses in ductility, and according to our experiments it does not matter much whether compressed air or steam, or hydrogen, or nitrogen gas is used. It might have been thought that the phenomenon is due to oxidation, but the same results are obtained if non-oxidizing gases are employed so that any oxidation of the metal from the spray nozzle to the surface to be coated is avoided.

Spraying Metallic Powder.

In a series of further experiments metallic powder or metallic dust was sprayed with great force. In this case the process may be aided by heating the metallic powder or the gas or the surface to be coated. If, however, the kinetic energy of the metallic particles is large enough, no flame or other method of heating is required with certain metals. This surprising result is probably due to the fact that the metallic particles shot upon the surface with great speed (great pressure) get heated momentarily and are thereby welded together.

In the apparatus described in German patents 252,565 and 253,510 the metallic powder is blown in an adjustable quantity by means of compressed air to the spray nozzle and is then blown through a concentrically arranged blow lamp flame. With lead, tin, and zinc, compressed air and illuminating gas are used. With metals which are not so easily fusible oxygen must be employed.

In this process it is important to avoid overheating as this would cause evaporation of the metal. On the other hand, if the temperature is insufficient or not uniform the different particles do not weld together into a uniform and intimate coating.

Very promising results have been obtained so far, especially with tin and zinc coatings, as described in the *Chemiker Zeitung*, 1911, No. 155, p. 1434. It may be mentioned that the new zinc coating process has been introduced on a large scale in several factories in Belgium and France and also in the French Navy.

Coating with lead by this process is also successful if non-oxidized lead powder is used. This condition, however, is not easily fulfilled in practice since lead powder oxidizes when in contact with air.

The Latest Development of the Spray Process.

An observation which was made incidentally in the spring 1911, led finally to a new method which is different in principle from our former methods. The fundamental idea is to melt from a metallic strip or wire only such a small quantity of metal as is to be sprayed at any moment. Fig. 1 illustrates the principle; *c* is the surface to be coated; *a* is the metal from which the coating is to be made. It is placed within the guide cylinder *b* and is being pushed uniformly forward so that its lower end is molten off by means of the flame *e*.

The drops which form in rapid succession are finely subdivided by means of the "transportation wind" blown through the pipes *d* and are thereby shot upon the surface *c* where they weld together and form a beautiful metallic coating. The thickness of the coating is practically proportional to the time of the spray operation. The gas pipe for the flame and the pipe for the "transportation wind" may also be arranged concentrically, as shown in Fig. 2.

The pressure to be used and the chemical nature and temperature of the flame depend primarily upon the nature of the metal to be sprayed and the same is true for various other factors like the speed with which the metal to be sprayed is pushed through the guiding cylinder, further, the quantity, temperature, and pressure of the "transportation wind," the distance of the spray nozzle from the surface to be coated, etc.

It is well known that with the oxyacetylene blow pipe and with the electric arc temperatures of 2500 deg. or over may be

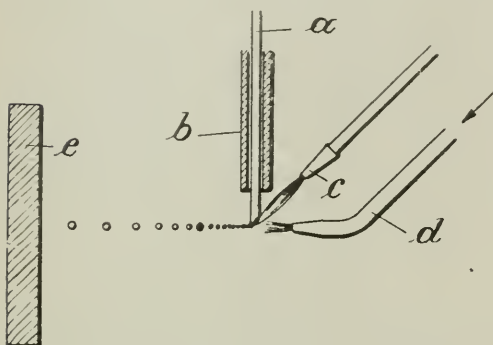


FIG. 1.—PRINCIPLE OF METAL SPRAY APPARATUS

produced. The application of this process involves neither theoretical nor practical difficulties in the production of coatings of brass, copper, nickel, iron, gold, and even platinum, and it permits the production of beautiful coatings free from oxide on any surface whatsoever. The italics are used with purpose to avoid any misunderstanding and to indicate that even inflammable substances, like wood, paper, celluloid, woven fabrics, and even explosives can be coated by this process. As an example, it may be said that a bunch of safety matches taken from its box and brought without any further preparation into a spray of brass was covered within two minutes on all sides with a brass coating without the heads of the matches being ignited.

Fabrics of every kind, including fine Brussels laces, can be coated with copper after they have been spread on a board. The Army department of a large country makes experiments at present with balloon material which we coated in Zürich on one side with a brass film of 0.002 millimeter (0.08 mil.) thickness.

The surprising fact that inflammable or fusible materials can be coated by this new process is probably explained by the considerable reduction of the temperature of the very finely divided particles caused by the sudden expansion of the com-

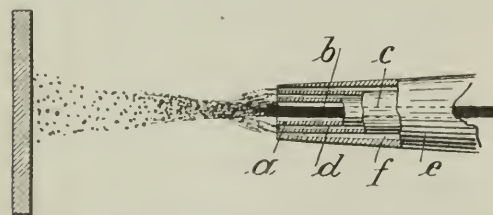


FIG. 2.—PRINCIPLE OF MODIFIED METAL SPRAY APPARATUS

pressed air in the moment of spraying. Moreover, the first thin film formed acts later on as a protecting layer.

I do not intend to discuss the theory of the process any further, but one fundamental difference between this new process and the older process using metallic powder may be pointed out. In the new process, the temperature of the blow lamp flame is hardly much different from the melting point of the metal, since the spray takes place immediately after melting. On the other hand, in the old process in which metallic powder

was used, the spray coming out of the spray nozzle consists of an intimate mixture of metallic powder and air or gas and each metallic particle is surrounded by an insulating film of air which is, of course, highly heated so that the heat efficiency is necessarily reduced.

Other advantages of the new process are that there are no more losses of metal and that the blow lamp can be easily adjusted so that the melting and spraying process takes place in the reducing zone of the flame.

Fig. 3 is a photograph of a "metal spray pistol" which is based on the new principle. Its design is the outcome of experiments, which were continually carried out for more than a year and a half in which my associate of many years, Mr. F. Herkenrath, participated. *a* is the wire of the metal to be sprayed; *b* are two guide rolls which are geared to the compressed air turbine *c*. By means of the handle *d* the supply of the gas as well as of the compressed air is cut off.

The compressed air supplied to the apparatus first sets the small turbine *c* in motion (30,000 to 35,000 revolutions per minute) by means of which the wire is pushed with an adjustable and uniform speed into the spray nozzle. From the turbine the compressed air passes in form of a concentric cylinder to the opening of the spray nozzle and aids the blow flame in subdividing and spraying the metal.

When certain metals like iron and copper are sprayed there is a clearly discernible smell of ozone. We have not yet found a certain explanation of its formation.

Fig. 4 shows the use of the metal spray pistol in practical operation. It is suitable for making coatings of brass, copper,

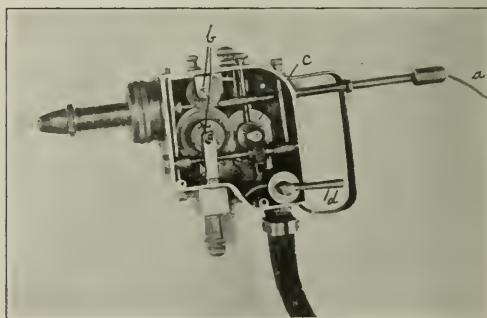


FIG. 3.—METAL SPRAY PISTOL

steel, etc., and is operated with hydrogen, oxygen, and compressed air. The box on the floor contains the coils of wire.

The metal spray pistol is particularly suitable in all cases where complicated surfaces or surfaces with large dimensions and in a somewhat remote position are to be coated. For instance, it is only a question of time when ships of the navy and of the mercantile marine will be coated with an adherent layer of copper. Experts consider the solution of this problem to be of eminent importance for the whole ship building industry and a practical solution of this problem has been sought as long as ships have been in existence.

Radiators for heating houses when coated with brass look good and also have a better heating efficiency than radiators coated as usual with a heat-insulating paint. In the electrical industry the process is suitable for many different purposes, as in the manufacture of resistors of all kinds, for making carbon conductive, for coating contact points in order to reduce the resistance, etc.

The many fields in which the process can be used and will be used, might best be illustrated by enumerating some of the things which have been sent to me in the course of time to Zürich to experiment with. It is a real museum which has nothing to compare with and which comprises brass-coated doll heads, sharpnail igniters, balloon material, metallized paper for transatlantic transportation, machine parts coated inside with

zinc or lead, beer kegs coated with tin, linen screens coated with aluminium for moving picture shows, cables coated with zinc, art products in which the metallic coating produces a special decorative effect, etc. It is no exaggeration to say

The Technical Analysis of Ferrovandium and Its Products

The Determination of Aluminium in Ferrovandium

By Wm. W. Clark

The writer will give a series of articles on the determination of the different constituents of ferrovanadium and its allied products. The methods given are in daily use, and have proven accurate, and rapid enough for routine work.

The determination of aluminium in ferrovanadium is the first, the method given having been in use for about three years. It is fairly rapid in comparison with other aluminium determinations, taking about 3 hours for a single determination, while a single operator will run 12 in an 8-hour day.

A number of methods were taken up and discarded, those that at first sight looked the best adapted for the material being given with the objection to it.

The precipitation of the aluminium from the neutral or acetic acid solution as phosphate with thiosulphate of sodium, and the precipitation with organic precipitants were discarded as useless, as vanadium precipitated with the aluminium in each case and under all conditions. The precipitation of the iron with sodium hydrate, and the determination of the aluminium in the filtrate is a long, tedious operation. The iron forms a very bulky precipitate, from which the aluminium is separated with difficulty, taking several reprecipitations, and a large bulk of wash water. The method while accurate takes about 12 hours for a single determination.

The fusion of the alloy with sodium peroxide, and the determination of the aluminium in an aliquot portion of the filtered solution, gives low results, as some aluminium is always held out of solution by absorption in the iron precipitate, and consequently an accurately measured part of the aluminium is not taken with the aliquot portion.

The method finally accepted as being accurate under all conditions, and rapid enough for a busy laboratory is as follows.

Dissolve 0.50 grams of the finely powdered material in 30 cc. strong hydrochloric acid and 5 cc. strong nitric acid, evaporate to a skim, add 20 cc. strong hydrochloric acid, transfer to a separatory funnel, washing out the beaker with a small amount of dilute hydrochloric acid (2 parts acid to 1 part water). Add 75 cc. ether, cork, and shake for 2 minutes. Allow the two layers to separate, and draw off the lower portion into a 400 cc. beaker, being careful to get the drainings from the sides of the funnel. Evaporate the dissolved ether, being careful not to ignite it, add 1 gram sodium chlorate, boil 1 minute, dilute to about 300 cc. with boiling water, and add sufficient excess of sodium hydrate solution to insure all vanadium and aluminium passing into solution. Add bromine if nickel is present until the precipitate turns black, boil 2 minutes, then add 25 cc. hydrogen peroxide, boil 2 minutes, and filter on a close filter using a platinum cone to prevent the paper from breaking, and washing the precipitate with hot water. Boil the filtrate and if any precipitate forms filter again on a close hard filter. Acidify the filtrate with dilute hydrochloric acid, add 25 cc. hydrogen peroxide, then ammonia until the solution just smells faintly of it, dilute to about 800 cc., and boil until the solution no longer smells of ammonia. Filter on a loose filter washing the precipitate with hot water until the washings are free from chlorides, ignite in a weighed platinum crucible at a low heat, finally blasting for a few minutes. Cool, add a few drops of sulphuric acid, 10 cc. of hydrofluoric acid and evaporate to dryness to remove any silica and ignite over a blast lamp until all sulphates are decomposed. Weigh and subtract from this weight any blank of alumina in the reagents. The resulting weight of Al_2O_3 multiplied by 106.60 gives the percentage of aluminium in the alloy.

The upper layer of ether left in the separatory funnel is distilled in an electric still, the ether vapor condensed and used again with very little loss.

The silica could be removed from the solution at the start by evaporation to dryness with hydrochloric acid, and subsequent solution in hydrochloric acid, and filtering, but a number of



FIG. 4.—APPLICATION OF METAL SPRAY PISTOL.

that every second day something new comes up, although it is also true that the inquiries received are often senseless and that valuable suggestions of new and useful applications of the process are the exception rather than the rule.

Zürich-Höngg,
Switzerland.

The metallurgical sources of gold recovery in 1911, according to figures just given out by the United States Geological Survey, were as follows: Dredging, 10.0 per cent; other placer operations, 13.1 per cent; amalgamation, 23.0 per cent; cyanidation, 26.1 per cent; chlorination, 3.8 per cent; smelting, 22 per cent. These proportions probably will not change greatly for 1912, although the recovery by chlorination may be entirely eliminated, and the recovery by cyanidation may increase. Dredging also will increase its output.

The filtration of finely divided precipitate may be accomplished by the aid of a thin film of collodion, according to Zsigmondy, Wilke-Dorfmüt and Galecke, in the *Berichte*. To prepare these films a 6 per cent collodion solution is diluted with twice its volume of ether and five times its volume of absolute alcohol, and poured in a thin film over a glass plate. When the ether has evaporated so that the collodion is no longer sticky the plate is plunged into water. After 10 or 15 minutes the film can be separated from the glass, and can be preserved under water. To prepare the filter, first place an ordinary ashless filter paper on the filter-funnel, and then cover the paper with a piece of the collodion film. See that the film covers the paper and is pressed against the sides of the glass funnel. Use in the usual way with a water-jet pump.

tests have shown that the removal of the silica in the crucible by means of hydrofluoric acid, gives the same results, saves considerable time, and removes any blank of silica from the reagents.

The lower portion drawn off from the separatory funnel should be a deep blue or green color, and if brown, as it sometimes is from the presence of nitric acid, must be evaporated again with hydrochloric acid, and the ether separation repeated with fresh ether.

The precipitate from the soda precipitation is small, allowing the aluminium to be washed out very easily. It contains all the nickel, manganese, and a small portion of the iron. The bromine is first added in this precipitation to make the precipitate of nickel granular, thus saving considerable time filtering, and the peroxide is added later to precipitate any manganese in the solution. The nickel can be determined from this precipitate, by dissolving it in hydrochloric acid, precipitating the iron with ammonia, and determining the nickel in the filtrate by titration with silver nitrate and potassium cyanide.

When oxidizing with hydrogen peroxide in the final precipitation an excess must be used, as this is the main point of the method. Vanadium will precipitate with the aluminium turning it a light yellow, if the peroxide is not in excess. This yellow color can be removed by adding more peroxide and ammonia and again boiling. The precipitate of aluminium hydroxide filters very rapidly, and will contain no vanadium when washed free from chlorides with hot water.

If the alloy contains an appreciable amount of phosphorus, the solution in the platinum crucible, after removal of the hydrofluoric acid, is diluted, sodium ammonium phosphate solution added, and the aluminium precipitated from the almost neutralized solution with sodium thiosulphate, filtered, washed, ignited at a low temperature in a porcelain crucible, and weighed as $AlPO_4$, which, after the subtraction of the blank, is multiplied by 44.38, giving the percentage of the aluminium in the alloy direct.

Following are some results on a synthetic solution of the approximate analysis of the ferrovanadium solution to which a known amount of aluminium was added in the form of aluminium potassium sulphate, and on a standard ferrovanadium.

Synthetic solution		Standard Ferrovanadium
Al. added.	Al. recovered.	Aluminium.
1.00%	1.02%	0.82%
	1.00	0.78
	0.98	0.82
	1.02	0.83
	0.99	0.79
0.50	0.49	0.79
	0.48	0.81
	0.52	0.83
	0.53	0.79
	0.50	0.81

American Vanadium Co.,
Bridgeville, Pa.

The only important manufacturer of pig iron in Mexico is the Cia. Fundidora de Hierro y Acero, of Monterey, Nuevo Leon. In 1911 this company produced 71,337 tons of pig iron, being an increase over 1910 of 26,246 tons. Imports of iron into Mexico during the fiscal year 1910-11 were made from Great Britain, United States, Germany and China. The duty assessed on foreign pig iron by the Mexican government is 0.02 peso per gross kilo, equivalent to about 0.45 cent per avoirdupois pound; on wrought iron and steel in ingots, 2.5 pesos per 100 gross kilos, or about 0.56 cent per pound. In addition to this duty a surtax of 5 per cent is assessable under the provisions of the budget of June 3, 1912.

The price of standard topographic maps issued by the United States Geological Survey was increased on January 1 from 5 cents each to 10 cents each, wholesale lots being sold at 6 cents when the order amounts to \$3 or more.

The Manufacture of Chilled Iron Car Wheels

A New Titanium-Boron Ferro-Alloy and an Apparatus for Testing Car Wheels

By Dr. Frederick C. Weber

In making a plain chilled iron car wheel from the usual mixture of pig iron, generally foundry No. 3, old car wheels, and steel and malleable iron scrap, there are always a small number of wheels which are found to be of very inferior quality, when subjected to testing and inspection.

A wheel should stand a thermal test of not less than two minutes before cracking. The drop test of dropping a weight onto the wheel is practically worthless, for there is no strain on the wheel flange during service which is similar to this drop test strain. The flange test should be a direct pressure strain on the flange and should show a direct resistance directly proportional to the weight of the car wheel and the load it is intended to carry.

No standards embodying the above are established yet. How many tons of strain there are on the flange of a wheel in service is variable. It depends on the velocity of the train, the degree of the track's curve, the width of the gaps at the rail joints and at the frogs and crossings, and on the way the car is loaded.

When making direct pressure flange tests the loaded car load weight is to be subtracted from the direct pressure breaking strain or this strain is to be compensated for by adjusting a vertical pressure for each weight car wheel. This can be done in a flange testing machine, of suitable construction.

The above factors collectively determine how much strength must be possessed by a wheel flange to be safe during service. I have arbitrarily placed the required strengths at the following figures based on such data as I have been able to gather. There are four car wheels which can be considered as being standard sizes, having a weight of 625, 675, 725 and 800 pounds respectively. The thickness of the flanges is proportional to the wheel's weight or better, proportional to the car load strain. The first weight is that of a wheel for cars carrying loads up to 30 tons, the second for cars carrying 30 to 40 tons, the third for cars carrying 40 to 50 tons, and the fourth for extra heavy cars of 50 tons capacity and over.

The breaking flange strengths should be taken irrespective of the varying load bearing upon the tread of the wheel. The first wheel should have a minimum flange strength before breaking of not less than 55 tons, the second of 65 tons, the third of 75 tons, the fourth of 85 tons. All of these strengths should be for between the brackets. For over the brackets the strengths should be 5 tons higher in each case.

The testing machine must be adjusted to exert a vertical load pressure of 12, 15, 18 and 20 tons on the several weight wheels to represent the loaded car pressure on the tread. And the flanges must show the above resistances for flange strengths before breaking with these load pressures bearing on the tread of the tested wheel. All wheels should, of course, average higher for all weights for all lots.

The above figures for strengths are purely arbitrary, but reasonable and fair and can be met in all foundry practice now. These figures should be the minimum for wheels to pass inspection.

Car wheels which are made from the above mentioned mixture will stand a flange test as above, all the way from 35 tons to 60 tons for test lot averages, and as high as 90 or even 100 tons for an occasional wheel. But quite a number creep in which are as low as 25 tons. It is this marked unevenness which is the most serious feature of a chilled iron car wheel. It is the low test wheel which gives out in service and which it is the aim to eliminate.

Under a freight car using a four-wheel truck there is a distribution of weight per wheel of practically one-eighth of the car's weight loaded or unloaded. In railroad operating practice this figures out as there being practically one-third of the total weight for the empty cars when figuring loaded train

weights. It gives a down pressure of practically 20 tons per wheel due to jolting on the tread of the wheel for the standard unit 50-ton loaded freight car, for the smaller cars less.

The factors to consider when calculating strengths required for flanges are as given above plus the centrifugal force in rounding curves and the pounding of the wheels over the gaps in the rails at the rail joints and frogs and crossings.

Iron which is of low grade and of uneven composition is more or less weak at all times. It is this uneven iron which causes so wide a range in the wheel's strength. Then there are the brown spot flaws in the throat between the tread and the flange which when present means a broken wheel every time in service.

This brown spot flaw I consider to be an incipient check due to viscosity of the molten iron at the time of pouring and it not filling the mold solidly. This incipient check at the moment of forming becomes oxidized to Fe_2O_3 because the iron is red hot. During service of the wheel the check becomes enlarged to a well defined crack and this enlargement becomes covered with Fe_2O_3 .

Then the chilled iron car wheel is subject to the formation of what are technically known as "bones." These bones I consider to be of two distinct kinds. First, bones of a segregation of slags, and second, a bone of a hard spot of steel. This steel bone hard spot comes from the steel scrap fed into the cupola with the mixtures, and the steel not becoming fully melted and incorporated with the rest of the metal in the cupola. The segregation bones of slag get into the wheel by slag flowing through the spout of the pouring ladle and by the segregation of slags within the body of the metal.

These two bones can be avoided first by exercising care to keep the work in hand by keeping slag from flowing into the pouring ladles from the bull ladle and second by scavenging or rather cleansing the molten iron.

Here a change in foundry practice is clearly indicated. The bull ladle must not only be mounted on trunnions, but it must be so constructed that it can be emptied from below either by having a vertically placed skimmer or be tapped below so that no slag will flow into the pouring ladles from that on top of the metal in the bull ladle. There must be some slag floating on the molten metal to cover it in the ladles to prevent oxidation and nitrogen absorption, but in the pouring ladles a film at once forms acting as a protection and then the pouring ladles are soon emptied. That will eliminate the plain slag bone.

The other, the segregation slag bone, is perfectly eliminated by means of a ternary alloy of iron, titanium, and boron.

The steel hard spot bone must be kept out by using no scrap steel in large pieces and then the scrap must not be covered with dirt or clay which will form a crust over the scrap and prevent it from melting and mixing into the mass of metal in the cupola.

It is at the frogs and crossings where the bones play havoc, for the softer metal at the sides of the harder metal of the bones becomes crowded past the bones by the cold flow of the softer metal by the pounding and then it is only a question of time when the iron having become tired, breaks.

The strain on the flange varies through a wide range with the train in motion. The track is never dead level. When rounding curves the outer rail being higher equalizes some but the train speeds are never the same, so consequently the speedier the train the more pressure on the flange.

Then there is the factor of the temperature to consider, for instance, when the thermometer reaches 40 below zero with the attendant effect of making the iron cold short.

Making a chilled iron car wheel to meet these various requirements is not exactly a boy's job. The essentials are a tread and flange far harder than the rail to minimize abrasion and having a toughness needed to withstand the pressure produced by centrifugal force on the flange, in rounding and pounding around curves at all speeds and temperatures.

It can be done! The iron's composition and handling as influenced by the various processes to which it is subjected, are the factors which must be considered in making a wheel which is to meet the severe calls made upon it when in service.

The very first factor for improvement is to empty the bull ladle from below to eliminate the plain slag bone, and to neutralize in the pouring ladles any added cleansing or scavenging compounds or as the foundrymen say dope.

No steel or malleable scrap should be added to the mixture which is in any way dirty. As a matter of safety all steel or malleable scrap should be cleaned in a rattle-box first. This will surely eliminate the steel bone.

The coke and limestone and the pig iron used in the cupola must be analyzed so that the right amount and kind of limestone flux can be added to the mixture so as to make a silicate slag. To correct any thickness of the resulting slags from using old wheels of uncertain composition some fluorspar, usually about 1 per cent, must be added to the mixture along with the coke and limestone.

But now comes the vital determining factor of causing the iron to have the essential quality of a fine, even, live, homogeneous grain with no interference with the irons taking the chill and additionally having imparted that strength which goes with purer iron while making it more liquid in the pouring and through these absolutely to eliminate the brown spot flaw, the segregation bones, and so eliminate the low test wheel.

An alloy—or rather my alloy, for I was the first to make it—composed of iron, titanium, and boron imparts the above essential qualities. The alloy contains Fe74, Ti12, B8, AlSi, etc., 6 per cent. This alloy combines with the occluded nitrogen which produces brittleness, it being fluxed out. The boron fluxes out the slags from within the molten metal and incidentally brings up some phosphorus and sulphur. While the trifle of aluminium in the alloy reduces whatever oxides of iron and manganese may be in the iron which along with the slags in the iron makes the iron viscous or pasty.

A curious action of the boron is that at the temperature of molten iron the boron determines that some of the graphitic carbon enters into combination, it becoming combined carbon. While at the temperature of molten steel, all excess of boron over that required to flux out slags is converted to B_2C_3 and this acts to throw carbon out of combination, making it graphitic.

The iron treated with this alloy has a live appearance. Its color is of a lighter gray, finer grained. It is homogeneous throughout and the chill looks more fibrous and steely. During all testing of wheels treated with this alloy so far not a single brown spot flaw nor a bone has been found nor has a low test wheel appeared. None were found on flange testing as low as the minimum demanded, while a number were found breaking as high as 110 and 117 tons. And that for medium weight wheels No. 800 lb. wheels were made or tested.

To use this scavenging alloy means that the foundry methods and practice must be adapted to its requirements. The alloy being a fluxing alloy demands that it be thrown into the bottom of the pouring ladles, the ladles being red hot and empty or nearly so of the previously poured metal. The iron from the bull ladle flowing onto the alloy mixes it in and the fluxed out impurities of the treated metal rising to the surface promptly furnish the required protective film of slag.

The slags formed during melting the mixture and flowing from the cupola cover the molten metal in the bull ladle if they should flow from there into the pouring ladles and then come into contact with the cleansing alloy, they would at once flux between themselves and the alloy would be waste!

The alloy is not neutral to the slags like a metallic alloy, as for instance, ferrochromium. The cleansing alloy will flux out the impurities from the iron only when the iron and the alloy are in shape for the alloy to act on the impurities in the iron. The aim is to cleanse and scavenge the iron and not to flux any floating slags.

The alloy will when correctly used flux with the impurities of the iron into which it is incorporated up to its saturation point. Of course, the cleaner the iron the less alloy will be required to obtain a given result. But the very fact that it scavenges the molten iron by a true fluxing action means the production of a far more uniform iron.

All the way from $\frac{1}{2}$ lb. to 5 lb. per wheel were used. In some as little as $\frac{1}{2}$ lb. was required to show a good improvement in the iron. Even this small amount of the alloy eliminates the brown spot flaw.

Boron was found in the treated iron when 3 lb. and over per wheel was used. The 3 lb. per wheel were the strongest wheels.

It has been found in practice that the alloy must be thrown into the bottom of the pouring ladles in fine powder, not coarser than will pass a 12-mesh sieve. It dissolves very readily in the molten iron and is best thrown into the pouring ladles as fast as they are fully emptied from pouring a wheel. The alloy then has the necessary time for action from one pouring to the next.

The increased strength of the iron is that increase which is the result of its being cleaner, more homogeneous, and of finer, even grain, and the improvement is such that the cleansed metal is very decidedly better than non-cleansed metal. This alloy is the best scavenger so far used.

The testing of a wheel for its flange strength must be done with a proper regard to its being at least approximately correct. No such thing as putting a wheel into a frame and putting a hydraulic ram at the hub by guess at centering will show anything one way or another.

The frame must be a rectangle or a rhombus with its greater diameter placed horizontally. Its upper side must have an inserted vertical pressure block.

Sliding up and down in a rectangular orifice in this block's lower face there must be a recess accurately fitting the car

rod. This buffer block must be of sufficient thickness to stand up under the heavy pressures used. The block's faces must be in parallel planes.

The center of the interspaces between the brackets on the wheel must be correctly marked so as to enable placing and fastening directly over a mark on the frame and rail. And just so for over the brackets. The wheel must be fastened centrally over the piston of the hydraulic ram.

The vertical ram piston pressing on the vertical pressure block which represents the car's weight can be accurately adjusted to the several weight wheels by forcing the ram piston a little beyond the required pressure and then by means of a fine thread opening drip cock drawing off enough of the liquid as will make the needle show the desired pressure.

In this way only can wheels be tested so that the tests are of any value. And incidentally this avoids the guesswork method of centering the buffer block and so prevent slipping one over on a too confiding unsuspecting inspector by changing the leverage to get a resulting low or high test at pleasure.

There are a number of details connected with this work. The centering rod must be removed as soon as the wheel is firmly held in place by the buffer block so that the wheel hangs free while the pressure is applied.

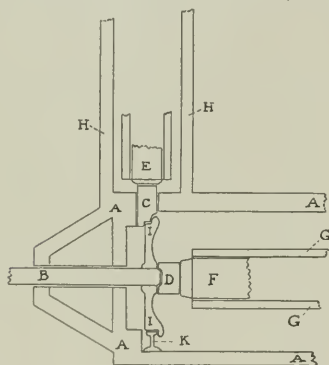
Then a washer of soft sheet iron or of annealed sheet aluminum can be placed between the piston faces and the blocks so as to compensate for any being out of line. The pressure on the vertical piston will be up to 2000 lb. per in. for a 6-in. piston and up to 3000 lb. on a 14-in. horizontal piston, when flange testing.

These pistons are best connected to one force pump by a proper system of double extra strong pipes with valves so inserted that the pressure is first put on the vertical pressure block and adjusted to the weight of wheel undergoing test. The pressure at 2000 lb. can be run up to 28 tons if desired. The pressure is then turned onto the horizontal breaking piston.

The pump is calculated for a working pressure of 3000 lb. which for the 14-in. piston gives an effective pressure of 230 tons for a direct acting breaking strain. This is more than three times the strain on a wheel flange in service and more than double the strain called for to pass a wheel by inspection.

A chain hoist on a traveler can be rigged up to handle the wheel for placing in position in the testing machine. And a pinch bar comes in handy for adjusting to guide marks.

Chicago, Ill.



SKETCH OF TESTING APPARATUS FOR CAR WHEELS

A frame, B centering rod, C vertical sliding block, D buffer block, E vertical piston, F horizontal piston, G and H cylinders, I wheel, K railroad rail.

wheel's flange and tread accurately tangent throughout an arc the chord of which must extend back for $\frac{1}{4}$ in. onto the tread for additional resistance.

The opposite lower side must be a rail or a rail head on which the wheel rests. This rail must be reinforced at the head for greater resistance. In this manner the wheel being free horizontally, the pressure is exerted on the flange in a similar manner by pressing on the rail head as obtains with the wheel in service. A wheel to make a flange test must have the central orifice centrally reamed out at right angles to the plane of the wheel. The hub must be smoothed up by planing or turning on a lathe accurately at right angles to the central orifice.

An iron or better, a steel rod, must be put into this central orifice which will just allow of its easy introduction and withdrawal. This rod must be tapering at the end for about an inch and the taper must project so that a buffer block can be placed over the taper between the hub of the wheel and the piston of a hydraulic ram. This accurately centers the buffer block over the hub of the to be tested wheel.

This buffer block must also be turned in a lathe so as to be round and true and be just large enough to cover the hub. It must have a recess made accurately to fit over the taper of the

Pennsylvania far outranks all other states in the value of its mineral output. In 1911 this state contributed, exclusive of pig iron, 24.7 per cent of the total mineral output of the United States. The reason for Pennsylvania's undisputed leadership lies primarily, according to the United States Geological Survey, in its great production of coal. It is almost exclusively the source of anthracite, and produces over one-third of the total bituminous output. Pennsylvania ranks second, next to New York, in the value of its manufactures, and stands first as a producer of cement, coal, coke, pig iron, lime, mineral paints, sand and gravel.

Tungsten is used chiefly in making steels that will hold their temper when heated, but it is more generally known as supplying the filament of incandescent lamps. Diamonds are used for drawing tungsten wire for the latter purpose, and wire of 0.0006 in. in diameter is now drawn in quantity. New uses are being found for tungsten in making electric contacts and targets for Roentgen rays.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in October, 1912, was 62. The total quantity of ore milled during that period was 2,252,040 tons. There were 9,075 stamps in operation, with an average duty of 8.56 tons per 24 hours. Tube mills in commission numbered 281. The yield for the month was 768.681 fine ounces gold, being about 60,000 ounces more than during the same period of 1911. The yield per ton milled was 6.64 dwts.

Organization

An Outline of Essentials from Engineering Practice

By T. C. Roberts

ORGANIZATION is one of the problems that all executives feel that they are thoroughly familiar with. They feel that their organization is laid out on exact scientific lines. They feel that if they are not getting the results they would like to get that it is not due to any fault of their method of organization, but due to some individual weak link in the chain in the form of a personal factor, and in a great many cases they are right.

There are numerous books written on the subject of organization, and it has been widely discussed, but 99 per cent of the friction or dissatisfaction in your organization is due to your lack of proper organizing ability. You who read this will probably say that it does not apply to your organization, because you have no friction or dissatisfaction among your men. But, let me tell you that you have. It may be, and probably will be, with your best men a carefully guarded secret, and one that they would be very reticent about divulging or discussing, even if approached upon the subject. It is

there just the same. Sometimes it is due to some one man in the organization, who may have reason to feel that he is abused, discriminated against, not allowed to advance according to his ability, etc., and sometimes he is right.

It is a fact probably based on the old adage that "familiarity breeds contempt," that a great many—I am even going to say that most executives—fail to appreciate the ability of their own men. Each and every one of us can verify the truth of this statement by reviewing our own experiences, and no doubt remembering some instance where we were pushed aside to make room for some one who had no greater ability than we had; and we could see what those higher up probably did not see; and that was, that the fellow who supplanted us received a considerable part of his experience after he got on the job. He probably, though, was a man of considerable ability and would

soon be able to introduce new and better methods into the organization, and if we will be absolutely fair in our reminiscences we will probably be able to recall some fellow whose place we took, who, if given the same support that we were accorded, would have been a better man than we were, and who probably proved it in some other organization.

The mistake that is made the greatest number of times is one of not outlining each man's duties, and demanding that everybody keep on their own side of the line. How often do you go to some man in an organization and ask him who has charge of certain work, and receive the answer that "if things were as they should be that he would have charge of that work, but as it is some one else has charge of it." Now, Mr. Manager, there is something for you to think about. That man is in one of several classes, and that answer was prompted from one of several causes. It was a case that was probably never brought to your attention. It might have been the only time

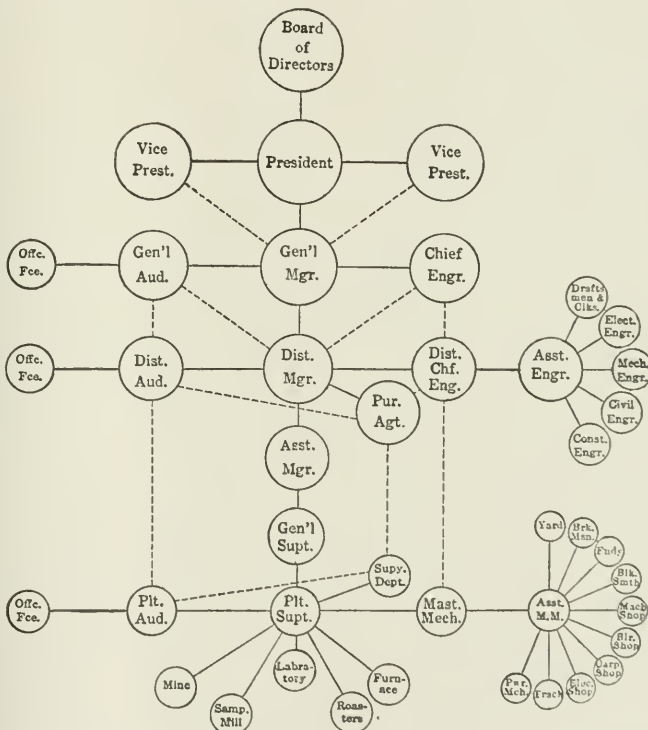


FIG. 1.—ORGANIZATION CHART SHOWING BUSINESS PASSING TO DEPARTMENT HEADS THROUGH ASSISTANTS

that that particular man ever gave utterance to what he really felt, yet how could you know that the feeling existed? I will tell you how. If you are a competent organizer you will be able to figuratively put yourself in the place of every man in your organization, and will analyze what your feelings would be if the system were applied to you.

A great many times your system is too flexible, and it would do you good to make an organization chart showing how many paths a report could take in getting to you. In other words, how many chances the man below has to get around the man above. And, how many times you offended some foreman by going around him to some man below, thereby licensing things to get out of their right channel, and making it possible for a weak character to tell somebody, in strict confidence, that when the "Old Man" wants to really know anything about the work-

ing of things he ignores those above and comes directly to him. Which remark, made in the strictest confidence, gets back to the man in the middle, who immediately and rightly gets peevish with you and the man below.

New blood in an organization brings new ideas, and a great many times new improvements, because the new man in trying to familiarize himself with the conditions sees a chance for improvement. While the new men may bring in new ideas, the

Another method, and I believe the most successful one, is to carefully define the duties of every foreman or superintendent, and where there is a chance of the duties of one department to overlap another the line should be very carefully drawn. This, you may say, is impossible, but it is not. By giving careful attention to the matter, you will find a point where the line can be drawn with satisfaction to all parties concerned. Do not assume that any two men will work in harmony in a case of this

kind, because they will not. If a man is ambitious to make a record that will some day land him in a higher position, it is human nature, when opportunity presents, to show the management how much he knows about the other fellow's job.

In other words, we all try to show that we are fitted to handle anything that is offered, and it is this very thing that will forever keep "Damon and Pythias" brotherly love out of the business organization. And, you can not blame any one of a dozen foremen in a large organization, and who are on an equal basis as far as salary and importance of position are concerned, for endeavoring to show the management that he is better fitted to fill a superior position than any one of the other foremen in his class. And, if in calling attention to his ability he gets on the other fellow's territory you can not expect the other fellow, who is equally ambitious, to not retaliate in kind, and there is a condition to watch.

One of the hardest conditions to meet is the impatience of men to wait a long time for promotion, and it occasionally creates trouble, because men striving for recognition and success in the line that they are engaged in are not prone to sit down contentedly and wait for a vacancy to occur, but they are liable to call attention to the other fellow's failings in the same breath in which they sing their own praise.

A properly arranged organization chart, showing the line of succession, will do as much toward keeping the business in its proper channel as any other one thing, and these charts can be posted throughout the works so that no man would have an excuse of ignorance for going out of the proper channel, except in case of emergency. Also a chart should be drawn in lines of

two different kinds, one showing the course to take normally, and the other showing the course to take in case the proper channel is closed.

Some managers prefer to have all reports from the works come up in line of succession through their assistants, as shown in Fig. 1, and some prefer to have the reports come direct to them, and they then assign to their immediate assistants the work that each one is to check (Fig. 2). It takes only a few minutes' time to lay out such charts and they are very valuable, as has been proven by experience. Continuous lines show path to be used under normal conditions, dotted lines show path used in cases of emergency.

In addition to the general chart, each department should have

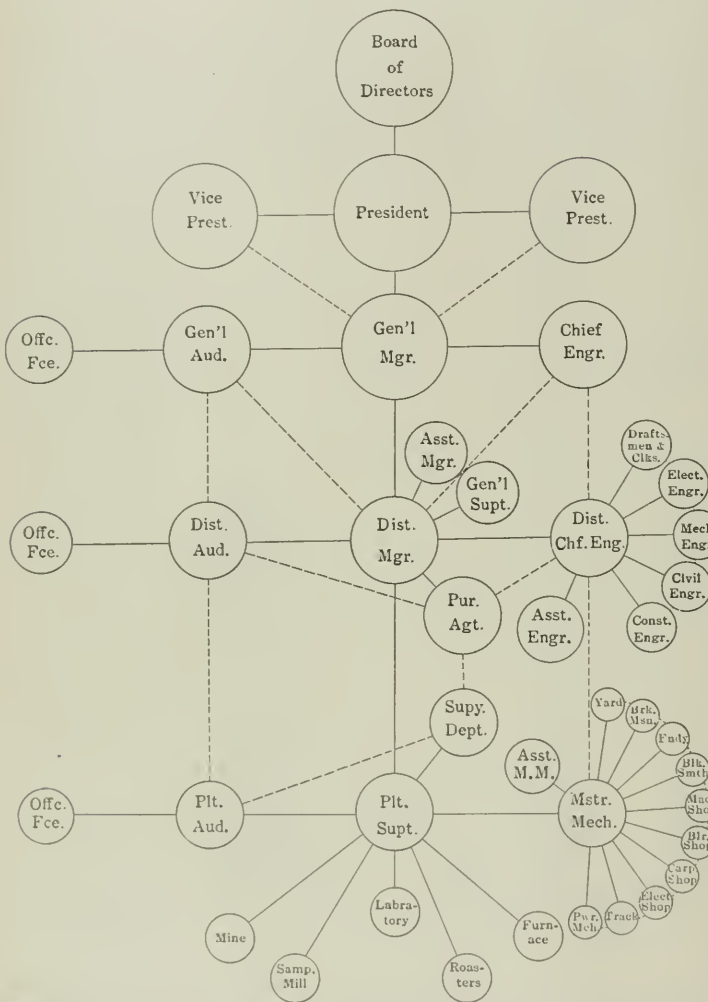


FIG. 2—ORGANIZATION CHART SHOWING ASSISTANT DEPARTMENT HEADS ACTING ONLY ON CONSULTATION WITH DEPARTMENT HEADS

old men in charge of various departments, by taking an occasional trip through other plants, would be just as quick to see and bring back ideas that would more than pay for the expense of the trip.

Different managers use different methods in keeping in touch with what is going on. Some do not believe in giving too much authority or putting too much confidence in any one man, and adopt the theory that if they want to know what is going on in one foreman's department that they should ask the foreman of some other department to make them a report on the other fellow's work, and yet this class of managers think they know all about maintaining harmony and satisfaction in an organization.

a detail chart showing the department lineup. Try it and see what results you get. You will probably find several men will offer in explanation of their actions in the past regarding certain things, that they did not know that they were supposed to take such matters up in the manner outlined on the chart. You will probably be surprised to find out how little each man knows of what your intentions really are in your system of organization.

I recently read an article in a technical magazine on the subject of organization. Also, the editor's comments on the same. Both the author and the editor seemed to be offering a solution for a difficulty that had arisen in a certain organization, due to the difference in training between an operating and a construction engineer. They seemed to feel that there was such a great difference that they would not be able to agree on any subject pertaining to their work, and therefore should be kept entirely separate, and each one report separately to the president of the company.

This division often confronts the manager of a large organization. But, to the man who is in close touch with the situation, the solution is apparent. The men in the operating department have probably for several years been operating their plant more or less successfully. If any difficulty has developed they have solved the matter in a manner satisfactory to themselves. They know the weak spots in their equipment, and what should be done to strengthen them. Working along these lines the time comes to install new equipment. The engineering department is consulted, through the chief engineer. Plans should be worked up, in consultation with the operating department; the designing engineer, the operating engineer, and the construction engineer working together under the supervision of the company's chief engineer.

When the plans are out, material purchased and delivered, the actual work of installing the plant should be under the direction of the construction engineer, who has outlined his method of procedure by familiarizing himself, in consultation with the operating engineer, with the conditions in the operating department effected by the construction work. But, both of these men should be under the supervision of, and report to, the chief engineer, the chief engineer in turn making his report to the manager or president.

It is absolute folly to argue that operating engineering and construction engineering should be divided into two departments, each one working absolutely independent of the other, and each one reporting to the president, as suggested in the article mentioned above. In a great many cases the president or manager, as the case may be, is not familiar with the dividing line between the operating and construction engineering. Therefore, both these departments, along with all other engineering work of any nature, should be under the supervision of, or referred to the chief engineer.

If more managers would realize this fact there would be less costly experimental work and time lost by the head of some department trying to work out some pet scheme of his own, without the aid of the engineering department. I have known of various organizations where the foremen or superintendents of the various departments, for fear that they would not get credit for ideas originating with themselves, have worked out their own engineering problems, which is entirely wrong if the organization is maintaining an engineering department.

The engineering department, through the chief engineer, should be directly responsible to the manager of the organization, and all matters going into or out of the engineering department should pass through the manager, or his immediate assistants. I am familiar with one mining and milling organization whose policy it was to keep each division of engineering separate. That is, they had a chief mechanical engineer, a chief civil engineer, a chief electrical engineer, and a chief hydraulic engineer. Each branch had its own assistant engineers, draughtsmen, clerks, etc. Each chief engineer was reporting directly to the general manager. They were in a constant turmoil. If there was a piece of work to design affecting more or less all of them, each department would design a propo-

sition as they thought it should be. Each one holding out for his particular design, with a consequent delay, and a hybrid creation when the job was finished.

This condition prevailed in this organization, which was no small affair, until some one suggested the appointment of a general chief engineer, and while it was true that the chief engineer of each department felt that he should be appointed as the general chief engineer, yet when the company appointed a man from outside the organization, they all fell into line, the engineering departments were consolidated with each of the original chief engineers as the head of a department of their particular line of work.

Two other organizations that I am familiar with, one the American Smelting and Refining Company, and the other H. M. Byllesby & Company, one a mining, milling and smelting company, the other an engineering, holding and operating company are laid on very similar, and along the lines that outline each man's duties and I would like to see, in the interest of scientific organizations, a complete organization chart of each of these companies published, but I am satisfied that any one sufficiently interested could obtain a general outline of the plan of organization by communicating with these companies.

It is true, however, that in lining up your organization you will probably find some one in the organization whose ideas differ from your own, but that person should, as Hubbard says, "Get out or get in line," and unless every person in the organization will try to make the system as outlined a success, whether it is in accordance with their views on the subject or not, they are an undesirable factor in the organization, and should be eliminated.

*United Verde Copper Company,
Jerome, Arizona.*

Note on the Calibration of Optical Pyrometers

By Paul D. Foote

In the primary calibration of all forms of optical pyrometers it is necessary to sight upon a substance in which the temperature distribution is such as to obtain "black body" conditions. The Lummer and Kurlbaum electrically heated black body is in general use for this purpose. As modified by Wadner & Burgess¹ by the introduction of a supplementary heating coil at each end, very uniform temperatures may be maintained over a short length of the furnace. However, it is only after the most careful adjustment of diaphragms and the auxiliary heating circuit that platinum and carbon placed inside the "black body" become indistinguishable.

If the walls are very much hotter than the back diaphragm upon which the couples are mounted and the pyrometer sighted light will be reflected from the diaphragm, and an apparent temperature higher than the true temperature is observed. On the other hand, a large temperature gradient through the furnace may lower the emission so that the thermocouples indicate a temperature higher than that shown by an optical pyrometer.

A method has been suggested² which eliminates the rather complicated and expensive apparatus necessary for the Lummer-Kurlbaum "black body." The optical pyrometer is sighted on the bottom of a porcelain tube, preferably blackened inside, the tube being thrust in a crucible of molten metal at the freezing point.

Dr. C. W. Kanolt³ has applied this method to the calibration of the Morse pyrometer (and Holborn-Kurlbaum type of that pyrometer) by sighting upon a small carbon tube projecting into a pot containing about 100 grams of molten metal. This was placed in an Arsem vacuum furnace and the freeze or melt observed through the glass window of the furnace.

By slightly modifying the above methods, a simple and inexpensive "black body," which may be constructed without dif-

¹Bureau of Standards Reprint No. 55, p. 165.

²Burgess and LeChatelier. The Measurement of High Temperatures, 3rd ed., Wiley, 1912, p. 332.

³Bureau of Standards Technologic Papers, No. 10, p. 8.

facility in the shops of any industrial plant, has proved highly satisfactory for work at the Bureau of Standards.

A large graphite crucible E, Figure 1, was made, capable of holding 1000 to 1500 grams of metal, D. The graphite cover containing the cylindrical "black body" B, was turned on a lathe and a conical hole drilled through, making an opening of about 0.5 cm. at C. In this manner the radiation coming from the black body passed into the pyrometer and none was intercepted by the graphite walls, the dotted lines, H, representing the cone of rays required by the optical system of the Holborn-Kurlbaum pyrometer. Over the metal was placed a layer of powdered graphite, G, in order to prevent oxidation.

A stirring rod, A, could be used to insure a uniform temperature through the pot, although the high conductivity of the metal probably makes this unnecessary.

The pot was heated in an electric furnace constructed by winding nichrome or platinum ribbon upon a porcelain cylinder 8 x 35 cm. However, if due care is taken to pack the crucible in a bed of powdered graphite, and some protection is afforded against the stream of oxidizing air, a simple gas furnace will serve very satisfactorily. Pots which have been in use twenty or thirty hours as yet show little sign of crumbling.

As a practical test of blackness, it is interesting to note that the opening, C, was indistinguishable and the only means of superposing the lamp filament and the image of the opening was by allowing a few drops of water to fall into B when for a moment C would appear as a dark spot in the field. Melts or freezes running

from fifteen minutes to an hour are easily obtained with antimony, silver, copper and other metals. A typical freeze for antimony is shown in Fig. 2. The flat part of the curve at once shows the current through the pyrometer lamp corresponding to the freezing temperature of antimony (630 deg. C.). By the use of the melts of three metals, the parabolic empirical relation $i = a + bt + ct^2$ may be established.

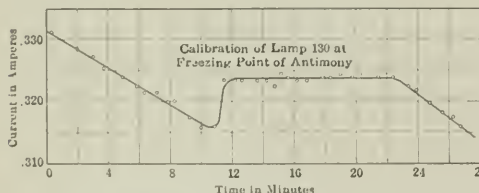


FIG. 2—TYPICAL FREEZING CURVE

If an observation at 1200 deg. C. is desired, inasmuch as there is no metal known to have a melt at this temperature, copper may be superheated and its temperature measured by means of thermocouples immersed in the liquid. However, when observations are made upon copper (1083 deg. C.), copper-silver eutectic (770 deg. C.), and antimony (630 deg. C.), the curve extrapolated as far as 1200 deg. C. reads identically with values obtained on the Lummer-Kurlbaum furnace at that temperature, and even as high as 1400 deg. C, the divergence between the extrapolated curve and the values actually observed is less than 1 deg., an amount of minor importance in industrial work. For higher temperatures an absorption glass should be used both on account of the inadvisability of

extrapolation over so great a range and for the safety of the pyrometer lamp, the filament of which begins to deteriorate, slightly, in this region.

Meyer¹ has suggested the use of an equation of the form $i = aT^n$ for the calibration of pyrometer lamps, where a and n are observed constants and T is the absolute temperature. This assumes² the filament to be of such a length that the thermal conduction along it is negligible. Both equations have been tried for several pyrometer lamps of the Bureau of Standards, and the parabolic form seems to fit the observations much more closely than the exponential equation. The latter form, however, possesses a distinct advantage in that only two points are required for calibration, but when used, it is necessary to know that the amount of heat dissipated by conduction along the leads is small in comparison with the total energy required by the lamp.

While the method of calibrating pyrometers by means of melts of certain metals as described above, is particularly intended for the Holborn-Kurlbaum pyrometer, it may be used just as satisfactorily with any form of optical pyrometer, care being taken that the cone of rays coming from the opening of the "black body" is not intercepted by the wall of the conical tube, and that the aperture is large enough to completely fill the optical system of the instrument.

Bureau of Standards.

Washington, D. C.

Effect of Heat on Amalgamating and Concentrating in Cyanide Solution

The use of warm solutions in cyaniding gold and silver ores, more particularly the latter, is no longer an innovation. Data on the practice are not as numerous as could be desired, but several instances are recorded in which warm solutions favorably affected extraction and settling. In some cases, perhaps, the results have shown little if any improvement over the use of solutions at normal temperature. This has been true of some silver cyanide mills of Mexico. Opposed to this, however, is the experience in silver cyaniding at Tonopah³, where solutions at 90 deg. F. materially increased extraction. In South Africa⁴ solutions are heated to temperatures ranging from 90 deg. to 120 deg. F., with beneficial effect on settling and a consequent increase of 20 per cent in tonnage treated. Other instances⁵ of the use of warm solutions can be found in the practice at the Hudson and Ophir mills, Colorado. Perhaps the most that can be said of the practice is that, in some instances, warm solutions are beneficial, whereas in others only negative results have been obtained.

Through the courtesy of the Liberty Bell Gold Mining Company, Telluride, Colo., we are able to present some interesting data regarding the effect of cold and warm solutions, particularly referring to amalgamation in cyanide solution. Metallurgical practice at the Liberty Bell mill⁶ includes stamping in cyanide solution, with subsequent amalgamation after both stamps and tube mills. Prior to April, 1912, cyanide solution was used at ordinary temperature, say 60 deg. F. Beginning with April, 1912, there was a marked increment in solution temperature, and a notable increase in percentage extraction of silver. The natural assumption was that the increased saving was due to better cyanide extraction. But upon tabulating the percentage recoveries from the different mill departments, some unexpected figures were obtained, which did not wholly confirm the first assumption. The following table gives typical percentages of recovery in the different departments over a period of one year, with an opportunity to compare the results obtained during the first half of that time, when cold solutions were used, with those of the last half when the solutions were warm.

¹Meyer, Diss. Greifswald, 1911.

²Pratt and Meyer, Zeit. fur wiss. phot., October, 1911, p. 135.

³Minn. & Sci. Press, Jan. 27, 1912, p. 176.

⁴Rand Metallurgical Practice, Vol. 1, p. 223.

⁵This journal, Jan. 1912, p. 27; Jan. 1913, p. 25.

⁶Notes on the Liberty Bell Mine, C. A. Chase, Bull. 60, A.I.M.E.

A marked gain is seen in total silver recovery, amounting to 10.57 per cent. Forty per cent of this gain, however, is by

Period.	Gold.				Silver.			
	Amal.	Cyan.	Conc.	Total.	Amal.	Cyan.	Conc.	Total.
First half.....	48.98	35.71	7.52	92.21	7.12	37.67	17.24	62.03
Second half.....	50.20	34.40	8.40	93.00	11.30	39.70	21.60	72.60
Gain second half.	1.22		0.88	0.79	4.18	2.03	4.36	10.57
Loss second half.	1.31							

amalgamation, and another 40 per cent by concentration, with only 20 per cent by cyanide.

Some laboratory tests were conducted to check the results obtained in the mill and to explain the distribution of the gain in silver recovery in the different departments. The following amalgamation tests with hot and cold pulp (barren solution) are suggestive.

Amalgamation Tests.

Test No.	Mg. Silver in Ore.	Mg. Silver Recovered in Amalgam.		Per Cent. Silver Extracted by Amalgamation.	
		Hot 85°.	Cold 60°.	Hot 85°.	Cold 60°.
1	61.8	2.18	2.90	3.5	4.7
2	"	3.41	1.65	5.7	2.7
3	"	5.50	2.95	8.9	4.8
4	"	2.68	1.58	4.3	2.5
5	"	5.93	3.46	9.6	5.6
6	"	4.22	3.94	6.8	6.4
7	"	6.86	2.90	11.2	4.7
8	"	9.62	3.16	15.5	5.1
9	"	4.80	2.60	7.8	4.2
10	"	7.96	4.47	12.8	7.2
11	"	8.20	4.50	13.2	7.3
12	"	8.16	3.70	13.2	6.0
13	"	5.42	4.35	8.7	7.0
14	"	5.59	5.62	9.0	9.0
Mean	61.8	5.86	3.42	9.45	5.52

The figures show nearly 90 per cent increase in amalgamation at a temperature of 85 deg. F. as compared with 60 deg. F. Similarly the following agitation tests in cyanide solution show an increased extraction of both gold and silver at the higher temperature, although the gain is not so marked in the case of gold as with silver.

Agitation Tests.

Ore assay: Au. 0.20; Ag. 2.06.		Temperature 85°		Temperature 60°				
Test No.	Tail Assay.		% Extraction.	Tail Assay.		% Extraction.		
	Au.	Ag.		Au.	Ag.			
1*	0.018	0.58	91.0	68.5	0.020	0.53	90.0	74.2
2	0.012	0.65	94.0	71.5	0.022	1.32	89.0	35.7
3	0.012	0.75	94.0	70.3	0.022	0.92	89.0	55.2
4	0.012	0.65	94.9	72.7	0.022	0.72	89.0	65.0

*Temperature on this cold test may have exceeded 60°.

It will be noted from the above tabulated amalgamation tests that the increase in silver extraction with warm pulp is fairly consistent. The gold increase was more erratic. This led to the suspicion that silver was being precipitated on and amalgamated with the mercury after first being taken into solution by cyanide. To confirm this view the following amalgamation tests were then made, using only mercury and zinc-box head-solution, with no ore. The same quantities of mercury solution were used as in the preceding tests with ore and barren solution, and the test was conducted for the same length of time, with the following results:

Test No.	Mg. Silver Precipitated on and Amalgamated with Mercury.	
	Hot 85°	Cold 60°
1	3.54	1.10
2	4.20	0.80
3	3.46	0.86

The increase in silver precipitated on and amalgamated with the mercury in the case of the warm solution was 8.1 per cent of the dissolved silver in the solution.

The average silver recovery in mill amalgamation for the period October, 1911, to March, 1912, inclusive, when cold solutions were in use, was 7.00 per cent. The same average for the following six months with warm solutions was 11.30 per cent, showing a gain of 4.21 per cent. The average silver content of the mill ore during the latter period was 2.45 oz. per ton. The apparent increased silver recovery by amalgamation during this period was, therefore, 0.703 oz. per ton of ore treated.

The average quantity of dissolved silver in battery solution during the latter period was 0.207 oz. per ton solution. Assum-

ing five tons solution used in the batteries per ton of ore treated, and 8.1 per cent more of its dissolved silver to have been precipitated and amalgamated on the plates than during the preceding six months, the increased quantity amalgamated amounts to 0.1202 oz. per ton of ore, or an apparent increased silver extraction of 4.91 per cent.

The indications are, therefore, that the apparent increase in silver extraction by amalgamation during the last six months is in reality an increased recovery by cyanide, probably due, as shown by the accompanying agitation tests, to the increase in solution temperature. The total increase in silver recovery should, then, more properly be credited to cyaniding and concentration, including in the former item that increase which is shown in amalgamation.

The foregoing results seem to indicate that, at least in this instance, heating is beneficial, and that the processes of amalgamation cyaniding and concentration are materially improved when performed in warm solutions. The reason for the gain in silver recovery by concentration probably is explained by the effect of heat on the settling process.

The Present Status of the Induction Furnace The New Two-Phase Combination Furnace and Details of Lining Construction

In the *London Electrician* of December 13, Mr John Hårdén gives an interesting review of the present status of the induction furnace and of some recent advances in design and construction. From this article we extract the following information:

"Development in the past has been somewhat handicapped by the fact that certain of the distinctive types of induction furnaces have been controlled by different interests, and a step in the right direction has been achieved by a recent combination of these various interests, by which the exchange of information relating to results obtained in the European countries and in America is made easier, while, in addition, it becomes possible to make combined use of the good points of the various patented types in designing future furnaces.

"In Great Britain the induction furnace interests are in the hands of the Gröndal Kjellin Co. (Ltd.), of London, who now control the patents covering the Kjellin, Röchling-Rodenhauser and Frick types of furnace. The distinction between these three types is sufficiently well shown for the present purpose by Figs. 1 to 4.

"Neglecting those furnaces which have been erected from time to time for experimental purposes, there are now working, or under construction, nine Kjellin furnaces, with a total capacity of about 17 tons; seventeen Röchling-Rodenhauser furnaces, with a total capacity of about 57 tons; and one Frick furnace, with a total capacity of 12 tons. At a low estimate these furnaces would be capable of producing some 100,000 tons of high-grade steels per annum.

"It was inevitable that some trouble should have been experienced in the development of these furnaces. The metallurgist who is to use it is confronted with a device which provides him with a receptacle in which to handle his metal, of an unaccustomed shape; he has available a means for heating his metal more rapidly and to a higher temperature than is possible in the usual types of metallurgical furnace, and, finally, he finds that his molten charge is automatically and very effectively stirred, due to electromagnetic effects. Being of a conservative nature, he quails before such a wealth of improvements, and is inclined to upbraid the 'electrician designer' for wasting his time with such a 'box of tricks.' His interest has been aroused, however, and, with the aid of his valuable suggestions, improvements are rapidly being made in those points relating to both construction and procedure which may truly be said to come within his province.

"These remarks are made to emphasize the fact that success can only be obtained by means of the closest co-operation between metallurgist and electrician. An abundance of results have been recorded which prove beyond question that

the induction furnace is a device by means of which all the metallurgical operations requiring the melting and refining of raw materials for the production of finished steels can be performed with at least equal, if not better, results than can be obtained by any of the methods generally adopted."

Mr. Hården then discusses the outlook in various countries, with special reference to Sweden and Norway, Germany and particularly England. Concerning the United States the au-

thor may have held over his foreign rival, due to his having exclusive knowledge of those intricate processes for the production of high-grade steels, will long remain with him, and, so far as skill in manufacture is concerned, he must inevitably meet his rival on even terms. In other words, the magic of a Sheffield brand will have departed unless it is backed by a price comparable with that of a corresponding foreign brand. So far as export trade is concerned, the British manufacturer has this advantage over his foreign rival, particularly the American: that a large percentage of his raw material is in close proximity to the manufacturing center, while the manufacturing center is near the seaport. This advantage, backed by a reduction in cost of manufacture, such as may be obtained by the employment of induction furnaces, should ensure that the British steel maker will retain the large share of the world's markets for high-grade steels which he at present enjoys.

Two-Phase Combination Furnace

"The most interesting advancement so far unrecorded is the design of a two-phase induction furnace on the lines of the Röchling-Rodenhauser furnace, by means of which it is hoped to effect considerable economy by reducing radiation losses and by improving the electromagnetic conditions.

"The distinctive features of this new type of furnace will be seen from the diagram in Fig. 5.

"A and B represent the cores of two identical single-phase transformers, each designed to carry about half the load. C and D are the two primary coils, separated by annular spaces E and F from the lining shell G. The front and back of the furnace hearth, which is indicated by I, are connected by the narrow annular ducts J and K, which will contain the metal forming the two secondaries of the transformers. Provision is made to prevent the slag from entering and obstructing these ducts. The shaded exterior part represents the lining surrounding the hearth I and ducts J and K.

"It will readily be seen by a comparison with the diagrams 3 and 4 that this form of furnace has several advantages over the single and three-phase arrangements, both of which types have been built. The complication of the three-phase furnace hearth and roof were soon found in practice to be so great that the type has now been abandoned, in spite of the fact that the electromagnetic conditions, as compared with those in the single-phase furnace, were much improved. In the case of the single-phase furnace the shape of the hearth and roof is extremely simple, and many furnaces of this type are giving excellent results in practice. The electromagnetic conditions are, however, less satisfactory, and an improvement in this respect is desirable.

"In the arrangement of the two-phase furnace shown above, it is hoped to combine the good points of both the three-phase and single-phase types. It will be seen that the shape of the hearth is identical with that of the single-phase furnace, while, owing to the fact that the cores surround the narrow heating channels, and not, as in the case of the single-phase furnace, the wide hearth containing the greater part of the charge, which must, for mechanical reasons, be separated from the core by a considerable space filled with lining material, the magnetic leakage is considerably reduced and a higher power factor results.

"A further advantage is gained by the fact that the lining of the hearth may be designed with a view to reducing the radiation losses to a minimum, since the space devoted to this part of the lining will not have such an appreciable effect upon the electromagnetic conditions as is the case in the single-phase type. The use of the two-phase type does away with the necessity for introducing an expensive motor-generator set, with single-phase low periodicity generator, between the furnace and any existing two or three-phase supply.

"In the case of the larger size of furnaces, and if the periodicity of the supply is normally high, the power factor may be regulated by the installation of a rotary condenser. For a three-phase supply the regulating transformer, for varying the voltage at the furnace terminals and consequently the temper-

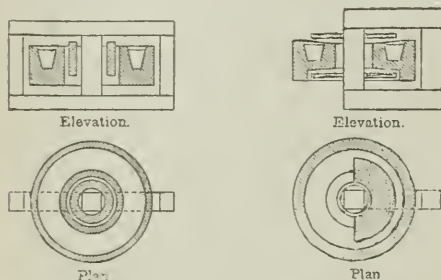


FIG. 1.—KJELLIN FURNACE.

FIG. 2.—FRICK FURNACE.

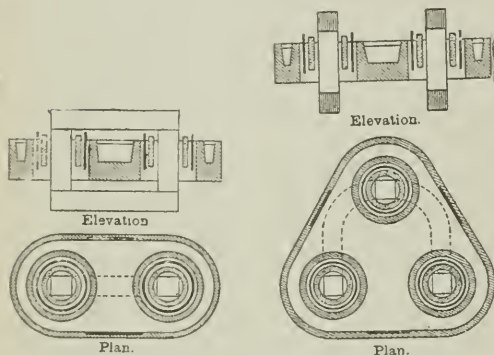


FIG. 3.—RÖCHLING-RODENHAUSER FURNACE. SINGLE-PHASE. FIG. 4.—RÖCHLING-RODENHAUSER FURNACE. THREE-PHASE.

FIGS. 1 TO 4—PRINCIPAL TYPES OF SIMPLE INDUCTION AND COMBINATION FURNACES

thor says: "So far as future development in the use of electric furnaces is concerned, the prospects are far brighter in the United States and Canada, more particularly in the States, than in any part of Europe. Here exists an enormous and ever-increasing demand for home consumption, which under the present tariff laws must be met largely by home production. This production on a large scale and at consequently lower rates, coupled with the low cost of Atlantic freights, will probably make it possible, as in other industries, to compete with the makers in Europe, in spite of the long and costly rail freights between the center of manufacture and the seaport.

Production on an enormous scale will be as much the maxim of the American manufacturer of high-grade steel as it is with the manufacturer of other commodities. Here, then is an almost ideal field for the induction furnace, since the demand for high-grade steels must be great and the facilities for producing cheap electric power, either from water-power, gas or oil are at least as favorable as in any other parts of the world. . . . Although the number of electric steel furnaces at present working in America is far behind the number in Europe, it requires no abnormal prophetic powers to foresee that in a very few years America will create a record in this as in so many other industries."

As to the outlook in Great Britain the author says:

"In these days of easy and rapid communication, it is idle to suppose that any advantage which the British steel maker

ature, may be designed on the Scott principle. By designing the furnace transformers for normal-periodicity current the iron section of the core becomes much less than is the case with the low periodicities, which are essential to large single-phase furnaces.

"As a matter of interest to the electrician, I give a diagram, Fig. 6, of the switching gear and connections which would be employed for a two-phase furnace supplied with power from two-phase mains. The voltage-regulating transformer would be arranged for hand operation, and is preferably in mechanical or electrical connection with the circuit-breaker, in order satisfactorily to break the large inductive load before switching over from one step to another on the regulating transformer.

"The furnace itself, being in reality merely a short-circuited transformer, will stand the sudden switching on or off of high currents without any ill effects."

In Fig. 6 *A* is a voltmeter; *B*, ammeter; *C*, wattmeter; *D*, synchroniser; *E*, power factor indicator; *F*, voltmeter; *G*, voltmeter; *H*, to synchronous motor; *I*, to induction motor for starting synchronous motor; *K*, furnace transformer coils; *L*, regulating transformer.

The electrical load of an induction furnace is not subject to large fluctuations. "It is, indeed, almost entirely under the control of the operator, the slight automatic change in the electrical conditions being due to the effect of the chemical and thermal changes in the bath, and, since these only take place slowly, no ill effect results from it. This is particularly the case with a furnace used for the treatment of hot metal, since less thermal change takes place in the charge during the time of its treatment."

Construction of Lining

In discussing the heat insulation of electric furnaces, the author refers to Dr. Carl Hering's articles in this journal on the subject. As to the unit "thermal ohm," he says: "This nomenclature, I fear, is somewhat apt to be misleading, and will, no doubt, be improved upon." But "those of us who are interested in the question of electrical furnaces cannot but feel indebted to Dr. Hering for raising this most important question in so able a manner.

"In addition to its heat-insulating value, a lining must, besides being of sufficient mechanical strength to carry the charge, be capable of withstanding the internal strains set up by expansion, while at the same time it must be dense enough to prevent excessive percolation of molten metal through the interstices between the particles of which it is composed. Its chemical effect upon the composition of the charges is, of course, also of great importance.

"In any furnace all these qualities are very desirable, but in

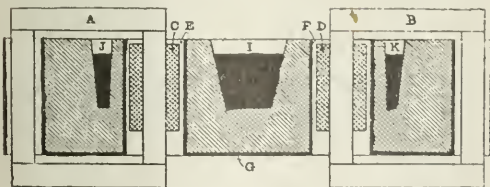


FIG. 5—TWO-PHASE COMBINATION FURNACE

none so much as in the induction furnace, and in none are they more difficult to attain. The high temperature possible, the automatic motion of the molten metal, both most desirable characteristics when under control, and the peculiar shape of bath, make the design and construction of the lining one of prime importance.

"In the first 1½-ton induction furnace built by Dr. Kjellin, at Gysinge, an acid lining consisting of silica brick was used. It was found, however, that too much silica was absorbed by the steel, probably due to the eroding action of the circulating metal, and it was soon abandoned in favor of a basic lining, which was constructed as follows: 10 kg of finely ground

burnt magnesite were mixed with every 500 kg. of sintered magnesite, to which were added 40 kg. of Holland clay, reduced to a paste with water and some boric acid; when it was ready for ramming in the furnace round a suitable template. For a complete lining up to the level of the molten metal some 2700 kg. of this mixture was used. The lining above the level of the metal was finished with a course of magnesite brick. On this lining 28½ tons of steel from cold material, representing about 210 complete charges, could be produced.

"The use of water in bringing the lining mixture to a proper

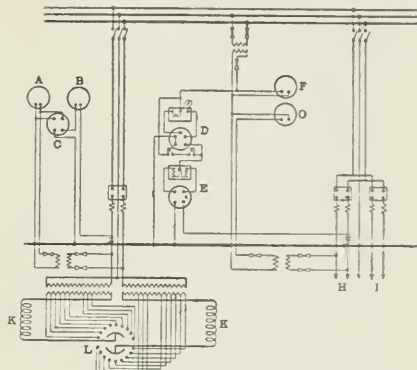


FIG. 6—DIAGRAM OF CONNECTIONS

degree of plasticity, to enable it to be rammed into place, was improved upon later by the use of hot tar, with which it is possible to obtain a more homogeneous and less porous lining, better calculated to withstand the tendency of the molten metal to percolate through the pores.

"The Röchling-Rodenhauser furnaces at the Röchlingsche Eisen & Stahlwerke, at Völklingen, in Germany and at Vereinigte Hüttenwerke Burbach-Eich-Dölnelmeier Akt-Ges. at Luxembourg, are all lined in the following manner. The furnace castings are first protected by means of layers of firebrick, which also serves to a large extent as the principal protection against radiation losses and rarely require replacement. On the bottom surface of the firebrick is first rammed, by means of automatic stamps, to the level of the hearth bottom, a layer of a mixture of carefully roasted dolomite and tar, ground and intimately mixed in suitable mills. Templates of cast iron for shaping the hearth and the heating channel's are then placed in position, and more of the lining mixture is rammed round them, after which they are removed and the top of the lining finished off with dolomite bricks.

"The roof of the hearth and the heating-channel covers are formed of silica bricks carried in suitable frames. Before placing the roof in position the lining is dried by placing cast-iron rings in the channels, the temperature of which is raised gradually to a point just below the melting point of the cast-iron rings by increasing the current in the transformer winding. These rings may be well coated with graphite and oil to retard oxidation. As soon as vapors from the burning tar have ceased to be evolved, the roof and channel covers are placed in position, a charge of molten pig is run in and the temperature further increased to the maximum required to complete the final sintering of the lining.

"In a 4-ton furnace the time required to replace a worn-out lining may be analysed as follows:

Tearing out old lining and ramming in new 124 hours

Burning out tar with hot rings 6 "

Completing the sintering of the lining 6 "

"During the burning and sintering processes some 230 units would be consumed.

"Such a lining as described above will stand from 90 to 125 charges of hot metal, or from 360 to 500 tons, according to the class of steel produced, which determines the time required.

Lining Construction at Poldihütte

"I have already indicated some of the difficulties which have had to be faced in the design and construction of linings, and it will be of interest to give some description of a lining which has been evolved by the Poldihütte Co., a well-known Austrian firm, after much experimental work with a 4-ton tilting Kjellin furnace. This firm, who installed their furnace in January, 1908, early realized the importance of the lining question, and set themselves, with conspicuous success, to solve the difficulties involved.

"The problem was complicated by the fact that the furnace, in addition to being fitted with tilting gear, was mounted upon a carriage, and provision was made for rotating it round a vertical axis to facilitate teeming operations. Under these drastic conditions, with the usual type of rammed magnesite lining, it was found impossible to obtain more than forty-nine charges without a renewal of the lining becoming necessary, on account of the appearance of cracks.

"After much experiment, it became evident that no simple form of rammed lining was suitable for the conditions under which the furnace was handled, and tests were then made with a lining compounded of several layers of materials having different compositions, with a view to localizing the cracks which so rapidly appeared. Working on these lines, so much improvement was made that in 1909 the average number of charges obtained from a lining amounted to about 200.

"From this date improvement was rapid, as the following table of results showing the average number of charges obtained during a period of three months indicates:

First quarter of 1910.....	227	charges.
Second " 1910.....	250	"
Third " 1910.....	262	"
Fourth " 1910.....	288	"
First " 1911.....	386	"
Second " 1911.....	392	"
Third " 1911.....	401	"

"The final lining evolved, which is fully patented, is indicated in Fig. 7, representing a vertical section of the lining.

"The lining is constructed in the following manner: First the bottom of the furnace shell *M* is covered with a single layer of firebrick *A*; upon this a layer of magnesite brick *B* is laid, using as cement a mixture of magnesite powder, lime and water. The outside of the shell *M* is then laid with a single course of firebrick *C*. Round the inside of this firebrick, and round the outside of the cooling jacket *N*, are fitted spacing pieces of wrought iron, which are fitted into place in somewhat the same manner as the staves of a barrel, and form the templates for the asbestos-wool channels *D* and *E*. The spacing pieces have holes punched in the top ends to facilitate removal, and in the case of those round the cooling jacket *N* are held in place by a steel band, and in the case of those round the firebrick lining *C* by means of wood staging.

"Cylinders built up of three or four sheets of cartridge paper and tracing cloth, with the smooth surface outside, are then placed in position at *G*, and strengthened temporarily by means of well-oiled sheet-iron cylinders. This prepares a space into which a mixture of well-roasted dolomite and tar carefully prepared is rammed. The remainder of the magnesite bricks *H* are now laid upon a sloping layer of magnesite sand *I*, using the same cement as before, while gradually, as the number of courses are increased, the sheet-iron cylinders holding the paper-covered rammed dolomite *F* in place, and which for ease of working are built upon parts, are withdrawn.

"In preparing the mixture for the lining *J* of the heating channel *L* much care is taken. It is made up of an equal proportion of three materials: First, high-grade 'Veit' or 'Stryan' magnesite, ranging in size from $\frac{3}{8}$ in. to $1\frac{1}{2}$ in.; second, similar material varying in size from $\frac{1}{4}$ in. to $\frac{3}{8}$ in.; third, fine crushed magnesite bricks from old linings, with which is mixed 1 per cent of finely ground 'Thomas' slag. Care is taken to remove all impurities, particularly metallic constituents as far as possible.

"Equal parts of these constituents are intimately mixed

with hot tar until a homogeneous substance is produced, which when squeezed in the hand forms into firm balls. Too much tar will cause the mixture to flow when rammed; with too little it will not bind well. Having prepared this mixture, it is rammed in while still hot on top of the magnesite bricks *H* up to the level of the heating-channel bottom. The channel template of well-oiled cast iron is then placed in position and the ramming completed.

"After drawing the spacing pieces forming the channels *D* and *E* and filling the spaces with asbestos wool, which may be used repeatedly for many linings, the channel template is removed and the lining is ready to be dried in the usual way with cast-iron rings electrically heated.

"To the first charge which is worked about 12 lb. of aluminium are added, with sufficient scale or rusty iron to oxidize it. The alumina formed fluxes the rammed magnesite, forming a surface which resists the percolation of the iron.

"The advantages of this lining are that the troubles due to vibration and expansion in all directions have been to a very large extent overcome. As regards its thermal insulating value, unfortunately there are at the present time insufficient figures available to determine the extent to which improvement in this respect over the more simple types of lining has been effected.

"The conclusions to be drawn from this brief survey of the induction-furnace position may be summarized in very few words. As in all things, there is room for further improvement, and I have tried to indicate upon what lines development is proceeding.

"Up to the present time the scope for the economic use, particularly in England, has not been wide, owing to the fact that they are still somewhat ahead of their time. All evidence, however, points to the fact that the growth of the demand, for which they, as instruments, will provide the supply, is fast reaching those dimensions required to demonstrate that induction furnaces are a necessity to the steel industry.

"Their inherent characteristics are of such value that it is safe to say that they will not be relegated to the limbo of forgotten inventions without leaving an abiding mark upon the history of this age of steel."

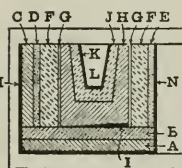


FIG. 7—SECTION OF POL-DIHÜTTE LINING

Silicon Castings for the Chemical Industries

Pure silicon is a metal of remarkable acid-resisting properties. This has long been a matter of common knowledge, but no use was ever made of it in the technical arts because silicon metal was not available in commercial quantities and also because no method was known of making silicon castings.

As stated in an article on the works of the Carborundum Company in Niagara Falls, published in our issue of May, 1909 (vol. VII, page 192), the Carborundum Company has been the first to produce pure silicon commercially, the process for its manufacture having been developed by its works manager, Mr. F. J. Tone, U. S. Patents 745,122, 833,427, 842,273 and 860,276 (see our vol. II, p. 3; IV, 464; V, 141).

That article contained some notes on the electrical furnace used for the production of silicon, some typical analyses of commercial silicon and a discussion of the various applications of metallic silicon. Among the uses it was mentioned that silicon was produced in the form of small castings, such as rods for electric resistances, but the difficulty of casting perfect forms of metallic silicon was noticed.

Since then great advances have been made in the development of a successful method of casting silicon as the result of long investigation which was attended with the greatest difficulties. All these have now been entirely overcome and it is possible by some recently developed processes to make castings of all ordinary shapes and sizes. Cast silicon chemical ware is thus placed within the reach of all chemical industries.

Properties of Cast Silicon

Density—2.5 to 2.6.

Electrical resistivity—0.15 ohms per centimeter cube at 18 deg. C., decreasing rapidly with rise in temperature.

Thermal conductivity—good (no absolute value determined).

Specific heat—0.176 (17 deg.-100 deg. C.).

Melting point—1430 deg. C.

Boiling point—2800 deg. (calculated).

Hardness—6 on Mohs scale.

Compressibility— 0.16×10^{-6} (least compressible of any element).

Coefficient of expansion— 5.39×10^{-6} at 100-200 deg. (less than glass or platinum).

Tensile strength—approximately 1000 lbs. per sq. in. (exceeds that of stoneware).

Resistivity of Silicon to Various Chemical Reagents

Sulphuric Acid: Sulphuric acid in any concentration under gentle evaporation has no action on silicon.

When vigorously boiling the loss in grams per square foot in 24 hours is as follows:

Concentration	Grams loss
10 per cent	.0592
20 "	.0829
30 "	.0799
40 "	.0681
50 "	.3197
60 "	.2634
70 "	.1657
80 "	.1391
90 "	.1273
95 "	after initial solubility of .0033 gm. action ceases

Fuming sulphuric acid readily attacks silicon.

Nitric Acid: Nitric acid either dilute or concentrated has little action on silicon. The loss in grams per square foot in 24 hours is as follows:

Concentration	Vigorous Boiling	Gentle Evaporation
5	.0000	.0089
10	.0237	.0207
20	.1391	.0355
30	.2102	.0296
40	.1539	.0888
50	.2072	.1509
60	.2486	.1569
70	ceases after .0352	ceases after .0891

Hydrochloric Acid: This has a slight but continuous solvent action on silicon.

Hydrofluoric Acid: This acid attacks silicon vigorously.

Chlorine and Chlorine Liquors: Silicon metal appears to be highly resistant to these reagents in practice.

Phosphoric Acid: This shows a continuous but slight attack which does not altogether disappear. This results in a slight turbidity in the product treated.

Alkalies: All alkalies readily attack silicon.

Uses of Cast Silicon

Silicon is used in the form of pipes for the conveyance of acid gases at a high temperature from stills to condensers, also in the construction of the condensing batteries themselves, proving much more efficient than stoneware owing to its high thermal conductivity and ability to withstand sudden changes in temperature. Silicon pipes are also used for the transportation of hot liquid sulphuric and nitric acids. It cannot be used for hydrochloric acid unless the discoloration resulting from the slight solubility is immaterial.

As a lining it is used in centrifugal pumps, acid valves and pipes for the elevation of corrosive liquids by compressed air, and for the construction of plows in ore roasters it has proven efficient.

Cast silicon is also being used in the form of shallow pans and pots for the concentration of zinc chloride solutions, sup-

planting the enameled stoneware vessels which are rapidly destroyed.

Cast silicon ware is produced in practically all shapes required by the chemical industry, such as pipes, evaporating vessels, receivers, tile, alembics crucibles and pump parts. Castings are preferably made at least $\frac{1}{2}$ inch in thickness, excepting pipes less than 2 inches in diameter, which can be made with a $\frac{3}{4}$ -inch or $\frac{1}{4}$ -inch wall. In the designs of silicon apparatus sharp corners and sharp bends must be avoided so far as possible.

In comparing the cost per pound of silicon castings with the cost of special iron castings used for acid work, attention is called to the low specific gravity of silicon. In silicon castings there is required only one-third the weight of an iron casting of the same form.

Notes on Chemistry and Metallurgy in Great Britain

Institution of Electrical Engineers

The Research Work Committee's report, which the council adopted at a meeting on the 12th of December, advises that the scope of the research work undertaken by the Institution should be such as would avoid over-lapping the work of other societies, and should be confined as far as possible to (a) the organization and correlation of research work in electrical engineering, and (b) the organization and supervision of specific researches in subjects connected with the electrical industry, and the allocation of grants in aid of the same.

The former should include (1) the establishment of a general committee for research to which investigators would be invited to send particulars as to the subject and character of the researches they are undertaking. Members of the institution would also be invited to communicate to this committee any special difficulties they had encountered which called for investigation and to make suggestions for subjects for research; (2) the collection and publication of particulars of plant available for special testing and investigation work in the various laboratories, together with a list of the subjects in which researches have been carried out; (3) the compilation and publication of bibliographies of specific fields of research.

With regard to the heading (b) the committee consider that it is desirable that the institution should originate and direct researches on certain subjects, and suggests that the General Research Committee should have power to appoint sub-committees of experts to deal with researches in selected subjects. Firstly, a series of investigations might be initiated on "the electrical properties of materials," which would deal with such matters as magnet-steel, copper, carbon, paper, rubber, mica, porcelain, oils and varnishes, etc., for which fuller data are required.

The method of co-ordination in research is essentially German in its origin, and has been productive of immense benefit to German science. It may confidently be anticipated that if the recommendations of the Committee of the Institution are loyally carried out very valuable results will be attained.

The Electrolysis of Nitric Acid Solutions of Copper

At the November meeting of the Faraday Society J. H. Stansbie read a paper dealing with this subject. Under ordinary conditions the deposition of the whole of the copper from nitric solutions is practically impossible. In the electrolysis of such solutions nitrous acid is formed, and the fact that this acid has a material action in the solution of copper by ordinary nitric acid suggested an explanation and a partial remedy. Investigation demonstrated that the amount of copper remaining in solution was greatest when the quantity of nitrous acid formed was greatest.

The remedy proposed by the author consists in the addition of sulphuric acid, and the employment of a rotating cathode. He suggested that the effect of adding sulphuric acid is partly that it affords preferential conductance and so reduces the production of nitrous acid; and also that it forms with nitrous acid a nitro-compound having much less solvent action on cop-

per than nitrous acid has. The rotating cathode almost entirely prevents the redissolving of the deposited copper; and thus, with due care, only a slight trace of copper remains undeposited.

Electrically Refined Nickel

Experiments with an electric furnace for refining nickel, at the Technical College, Trondheim, which have extended over a considerable period, have given very good results; and it is claimed that an electric furnace for the commercial production of metallic nickel has been successfully devised. The Kristiansand Nickel Refining Works will shortly be enlarged to turn out twice their present output; and it is probable that the new furnace will be tried there. The production of nickel in this furnace is stated to be quite satisfactory in metallurgical respects.

The Institute of Metals

Hitherto the annual general meeting of the Institute of Metals has been held in January; but the forthcoming meeting will be held instead on the 11th and 12th of March at the Institution of Mechanical Engineers. Dr. G. D. Bengough will present his second report to the Corrosion Committee which will embody the results of the condenser corrosion tests which have been in progress during the past nine months or so. A committee has been appointed to deal with the nomenclature of alloys; and another committee has been instituted to assist the Dominion's Royal Commission in inquiries as to the supply of non-ferrous metals and their ores.

The nomenclature committee will consist of nine members of the Institute of Metals and one member each of the Institution of Electrical Engineers, the Institution of Mechanical Engineers, the Institution of Naval Architects, the Institution of Engineers and Shipbuilders in Scotland, the Northeast Coast Institution of Engineers and Shipbuilders, and the Society of Chemical Industry.

The Physical Society

The annual exhibition of scientific apparatus, which was held at the Imperial College of Science on the 17th of December, was well attended, and the exhibits fully maintained the standard of former displays. Messrs. Evershed and Vignolles showed a new type of testing set reading up to 5000 megohms; and Messrs. Elliott Brothers' instruments also deserve mention. Messrs. A. Gallenkamp & Company exhibited a laboratory electric furnace in which windings of high resistance wire having a melting point of 1537 deg. C. are substituted for platinum.

Messrs. Gambrell Brothers had on view their new Jordan convection-radiometer and thermo-galvanometer for measuring small and fairly constant thermal changes. The Cambridge Scientific Instrument Company's exhibit included a double range indicator designed for taking steel-hardening temperatures with a thermocouple, and the Whipple temperature indicator. The Foster Instrument Company had their graphic recorder for giving records of temperature with a fixed-focus radiation pyrometer, a thermocouple, or a resistance thermometer, or, alternatively, for recording current or voltage. Mr. R. W. Paul showed several fine single-pivot instruments, an Irwin astatic dynamometer which is not influenced by external fields, and high frequency capacity and inductance measuring apparatus.

Messrs. H. Tinsley & Company exhibited a new type of Drysdale phase-shifting transformer, giving constant voltage at all angles and correct to 12 minutes of phase-angle. The Weston Company had a good assortment of instruments; and the degree of accuracy claimed for some of the direct-current examples is such that they could, for many purposes, be substituted for potentiometers. Messrs. J. I. Griffin and Son gave prominence to gyrostatic apparatus, including the Gray-Burnside motor gyrostat for demonstrating practical applications of the gyroscope and a gyrostatic compound pendulum which has a motor-gyrostat for its bob.

Messrs. Alex. Wright & Company showed improved apparatus for the determination of the amount of fire-damp in

air and for the accurate estimation of fire-damp, carbon dioxide and oxygen in the atmosphere in mines. The Marconi company exhibited the radiogoniometer, a new instrument for locating the direction of an emitting source or for regulating the direction of propagation; the British Wireless Telegraph Company showed a vibration proof crystal detector; the Dubilier Electrical Syndicate's main feature was high-frequency apparatus; and Messrs. Nalder Brothers and Thompson had a good exhibit of electrical instruments.

Excellent optical instruments were shown by Messrs. E. Leitz, Messrs. R. & J. Beck, Messrs. J. H. Daelmeyer, Messrs. Carl Zeiss, and the Westminster Engineering Company, but particulars of these and other meritorious exhibits would occupy much more space than is available. Mr. S. G. Brown gave a useful lecture on "Some Methods of Magnifying Feeble Signalling Currents."

The Greenwich Generating Station

Advice gratis is not always worthless, and had the London County Council condescended to give due consideration to the criticisms which appeared in certain technical journals and elsewhere on their decision to put down large low-speed reciprocating engines at the Greenwich tramways generating station, although the turbine had been proved to be unquestionably far superior in all respects for driving generators of large output, some hundreds of thousands of pounds would have been saved to the ratepayers. Three 3500-kw reciprocating sets have been running only about 6½ years, and during the greater part of this period—since the second part of the station, with its equipment of turbine plant, was completed—they have only been called on to supply a very moderate proportion of the output.

Now two of them will soon cease from their stately and ponderous 94 r.p.m. to be replaced by two 8000 kw turbine sets, in the same space; for the Highways Committee has arrived at the conclusion that the turbines will give the 9000 extra kilowatts with a net annual reduction of the coal bill to the extent of some £14,000 or more; and it is proposed to discard the remaining reciprocating engines and replace them with turbo sets of 8000 kw each in the not far distant future.

The aggregate generating capacity of the station will then be 52,000 kw against 34,000 kw at present; while the saving in coal will be something like £25,000 per year at a moderate estimate. Of course, the L. C. C. will want to make the best of a bad bargain and dispose of their magnificent engines with double H.P. and L.P. Corliss cylinders for a good round sum; but the report of the Highways Committee is such a splendid testimonial to their coal-consuming capabilities, as compared with turbines, that the prospect of their fetching more than a tithe of their cost may be much more remote than that of consignment to the scrap heap.

Clyde Shipbuilding

During the eleven months ended on the 30th of November, 258 vessels, having a total of 588,806 tons were launched from Clyde yards. The tonnage for November alone was 71,658 for twenty-nine vessels, and takes second place in the monthly tonnages of 1912. Exact figures for December are not yet obtainable; but a thoroughly reliable estimate puts the Clyde total for the year at 650,000 tons, or 20,000 tons more than in 1911, which was the highest hitherto. This means that more than 50 per cent of the tonnage for the whole of England this year has been put afloat on the Clyde. The warships built on the Clyde this year include the Ajax, 23,000 tons, super-dreadnought, three cruisers, two depot ships, one survey ship, and six destroyers. The liners numbered sixteen, of which four were for the P. & O. Company and five for the British India line.

Engineering Imports and Exports

The returns issued by the Board of Trade for the eleven months ended the 30th of November, show material increase in all descriptions of engineering imports and exports, except imports of ships, as compared with the corresponding period

of last year. Imports of iron and steel, including manufactures, amounted to £11,713,216, and the exports to £44,279,332, the respective increases being £1,578,416 and £4,248,897. The imports of other metals, including manufactures, reached £28,358,042, an increase of £3,294,028; and exports were £11,223,953, an increase of £1,117,744. Electrical goods were imported to the value of £1,320,182, an increase of £23,334; and were exported to the value of £4,049,758, an increase of £1,442,706. Imports of machinery totalled £6,206,125, an increase of £890,569; and exports went up to £30,418,438, an increase of £2,091,938. Imports of new ships are valued at £32,172, a decrease of £30,321; but exports touched £6,237,295, an improvement of £1,340,263. Taking the month of November alone, imports of iron and steel increased £197,664 and exports were £529,125 better; in other metals imports and exports rose £711,533 and £173,450 respectively; imports of machinery were £87,275 higher, and exports increased £34,060; imports of ships increased £4,020, while exports went up £88,303; but electrical goods showed decreases of £8,973 in the imports and £15,775 in the exports.

Market Prices—December

Tin has been on the whole, quiet, considering the usual character of its fluctuations. Opening at £226 it was up to £227 3/4 on the 2d, and dropped away to 225 on the 4th, and £224 on the 9th. On the 10th it was again rising and reached £227 1/2 on the 18th, and closes at £228-10.

Copper has had a downward tendency, opening at £76 1/2 and dropping to £74-5 by the 6th and £73-5 by the 17th. It then recovered in tone and was £75 1/2 on the 20th and finally closed at 76.

Cleveland has been fairly steady with downward tendency. Opening at 67/4 it had fallen to 66/8 by the 7th, but finished at 67/9.

Scotch Pig has been in sympathy with Cleveland, opening at 73/10 and dropping off after the 11th, it ultimately recovered to 73/9.

Haematite has been steady, opening at 82/-, rising temporarily on the 10th, but dropping again, it closes at 82/3.

Lead. Very steady along £18-10-0 with tendency to rise the latter part of the month; closes, however, at £18-10-0.

	£ s. d.
Aluminium, per ton.....	88. 0. 0
Alum, lump, loose, per ton.....	5. 15. 0
Antimony, black sulphide powder, per ton.....	21. 0. 0
Borax, British Refined Crystal, per ton.....	17. 0. 0
Copper, 10 to 25 per cent, per unit.....	13/6 to 14. 0. 0
Copper Sulphate, per ton.....	25. 5. 0
Carbolic Acid, liquid 97/99 per cent, per gal.....	0. 1. 5
Creosote, ordinary good liquid, per gal.....	3
Caustic Soda, Ash 48 per cent, ordinary per ton.....	5. 10. 0
Hydrochloric Acid, per cwt.....	5. 0
India rubber, Para, fine, per lb.....	4. 6 3/4
Naphtha, solvent, 90 per cent, 160 deg. C., per gal.....	1. 2
Petroleum, Russian Spot.....	8
Quicksilver, per bottle.....	7. 8. 6
Sal Ammoniac lump, firsts, delivered U.K., per ton.....	44. 0. 0
Sulphate of Ammonia, f.o.b., Liverpool, per ton.....	13. 18. 0
Sulphur, recovered, per ton.....	5. 5. 0
Shellac, standard T.N. Orange Spots, per cwt.....	4. 4. 0
Tin Ore, 70 per cent per ton.....	148 to 150. 0. 0
Zinc, Veille Montagne, per ton.....	30. 5. 0
Platinum, nominal per ounce.....	9. 5. 0

The differences on the month are thus:

Higher.	£ s. d.	Lower.	£ s. d.
Aluminium	3. 0. 0	Copper Ore	6
Copper Sulphate	2. 6	Quicksilver	4. 0
Sulph. of Ammonia. .	1. 3	Tin Ore	2. 0. 0
Tin	2. 10. 0	Copper	10. 0
Cleveland	7		
Scotch Pig	1 1/2		
Haematite	3		

London, England.

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver

Grinding-Pans vs. Tube Mills at the Homestake.—The experiments in the regrinding of sand at the Homestake mine, South Dakota, have resulted in the adoption of tube mills in preference to grinding pans. The details of the practice with both machines are given by Messrs. A. J. CLARK and W. J. S. ARWOOD, in *Bulletin* No. 98, Inst. Min. & Met.

The regrinding machinery consists of seven pans each 5 ft. in diameter, and a tube mill 5 ft. by 14 ft.

The pans are a modification of the Wheeler type, arranged to maintain a continuous discharge at the periphery, the depth of discharge being adjusted by the use of overflow rings of varying height, which are readily mounted upon the permanent lower portion of the shell. This shell extends 20 in. above the base-plate and determines the minimum depth of discharge.

The pulp fed to the pans is introduced in a distributing hopper, cast about the central shaft and forming part of the miller carriage. This hopper is tapped with pieces of pipe through which the pulp is led to the bottom of the pan and discharged directly in the path of the millers.

The total weight of moving parts when the pan has been equipped with new millers, is nearly 2200 lb., to which total the millers contribute 860 lb. The pans operate at 38 r.p.m. grind a trifle less than 20 tons per day, and consume \$4.70 each.

Mercury is fed in small quantities to the pans and the pulp, after overflowing the pans, passes over amalgamating plates, one plate, 4 ft. 6 in. wide and 6 ft. long, being provided for each pan.

These plates are dressed twice daily, and the amalgam is removed on alternate days; the pan is cleaned up when new plates and millers are to be installed, usually about once in three weeks.

The tube mill, driven by gearing from an electric motor, and provided with a scoop feeder, is lined with siliceous blocks, formerly 3 in. and now 5 in. thick. Flint pebbles are used for grinders, experiments with ore having proved unsatisfactory.

The pulp fed to the tube mill is diluted with sufficient water to carry it freely through the pipes into which the concentrating cones discharge, and is delivered into a dewatering cone 3 ft. 6 in. by 70 deg. slope, which is mounted above the feed hopper of the tube mill. The thickened pulp discharged from this cone carries fully 62 per cent solids, but it is interesting to note that the cone remains clear of any accumulation of sand and a rod introduced at the top meeting with no resistance until it encounters the bottom of the cone immediately above the bushing.

As this adjustment depends entirely upon the discharge area, it is necessary to frequently renew the bushing, and at this point the life of a bushing is not more than two weeks.

The overflow of the dewatering cone is nearly free from solids, containing but a small quantity of sand, over 99 per cent of which will pass a 200-mesh screen. This is led around the tube mill and unites with the ground sand at the discharge where it dilutes the pulp sufficiently to carry it over an amalgam table, 12 ft. by 18 ft. of silvered copper.

The mill grinds 73 tons per day, and consumes 37.5 hp when driven at 27 r.p.m.

All pulp discharged from the grinding machines unites with the main stream of pulp before arriving at the second battery of classifying cones.

The pans show a slightly greater recovery of amalgam, but the excessive wear of iron more than offsets all other considerations, and on this account we are preparing to install tube mills for all future work in this department.

The loss of mercury at this plant amounts to 0.061 oz. per ton ground; the amalgam retorts about 20 per cent and has a gold fineness of 630.

The two types of grinding machine have been operated on similar material with a view to determining which is the better

suited for the work, and the conclusion arrived at is in favor of the tube mill. A comparison of the work done shows the pan to be more economical of power, but we consider, and experiments have shown, that if 30 per cent to 40 per cent more sand were fed to the tube mill, and the discharge elevated and returned to classification system, the tube mill would be at least equal to the pan in this respect.

The daily performance of the tube mill and pans is determined by timing the filling of a 5-gal. bucket ($\frac{3}{4}$ cu. ft.) and weighing the catch. Wherever possible the tippie is used for diverting the flow to the sampler. Suitables have been drawn up so that the tons per day and the water ratio can be read at a glance from these data. The readings are made twice a shift and averaged.

Assay samples of the feed to the pans and tube mill are taken with slotted-top hand samplers, and are used for assay and sizing tests. Similar samples of the ground product are taken from the tippie discharge, which prevents contamination from amalgam plates. Three samples of 1.5 A. T. each are taken for assay.

Tin

Cornish Tin Dressing and the Vanning Assay.—New light is thrown on Cornish tin dressing practice and the use of the vanning shovel, in a paper by W. FISCHER WILKINSON published in *Bulletin* No. 99, Inst. Min. & Met. The author presents data obtained during two years' residence in Cornwall as principal of the School of Mines, during which time he was enabled to see what was being done in the mills of the district, as well as to investigate certain problems in the mill which is run for the benefit of the students.

Cornish mines and mills are weak as to statistics. The units adopted, namely, "black tin," "pounds of concentrate per long ton," etc., are indefinite, unreliable and ambiguous. The use of the vanning assay further complicates the situation and clouds the actual conditions. It is only since the introduction of the chemical assay that the serious losses in concentration have been made apparent. It has been generally stated that mill recovery was in the neighborhood of 90 per cent, with additional saving in the stream works, but the author comes to the conclusion that the actual extraction of tin throughout Cornwall today is between 50 per cent and 60 per cent. This great reduction is due to the abandonment of the vanning shovel and the substitution of the chemical assay.

The author gives comparative tests between the vanning shovel and the chemical assay on five different lots of ore and concentrate of varying grades. The results by the former method indicate from 22 per cent to 95 per cent of the tin shown by the chemical assay, and the highest figure was obtained by vanning high-grade, coarse cassiterite. The shovel seems to be least reliable on low-grade material. The figures show that on Cornish tin ores averaging from 1 per cent to 2 per cent tin, the vanning shovel will be about $\frac{1}{3}$ below the real value, and that mill extractions calculated on this assay will be in error to that extent. Thus a mill that shows a recovery of 85 per cent by the vanning assay will in reality be making a recovery of about 56.6 per cent.

Examples are given of the work of two companies that give publicity to their technical results. The Cornish Tailings Company, retreating an old dump containing 17 lb. tin per long ton, saves 40 per cent in a modern plant. The Carn Brea & Tincroft Mining Company recovers 95.81 per cent by vanning assay, equivalent to 73.7 per cent by the chemical assay. This is considerably higher than the other mines appear to recover.

The author believes that the first step in improving the recovery is to keep accurate statistics of the value of the ore; to abandon the vanning assay as well as the practice of reporting the yield in "black tin." Proper attention to statistics would show where the greatest loss was being made, and would indicate the direction for improvement.

The practice at the King Edward mill of the School of Mines was to crush on stamps through 4-mesh battery screen, running the pulp over a Buss table where 42 per cent of the metal

was recovered in the form of a concentrate containing 32 per cent tin. The first-class middling from this table was screened, the oversize being reground and combined with the undersize for classification. The second-class middling was screened, the oversize being thrown away and the undersize sent to the classifier. The slime and fine sand also were classified. The practice of tabling removed a valuable concentrate and relieved the classifier of 15 per cent of the original weight of ore and 42 per cent of its valuable metal. The coarse product from the classifier was treated on a Frue vanner and the fine on a round table. The tailings of both these machines were retreated.

The author believes that the use of screens such as the Cal-low of Bunker Hill would permit the immediate rejection of much worthless material, as experiments showed that the undersize of such screening contained a high percentage of the metal. The mill returns compiled from the battery sample and the tin sold during the period 1910-12, gave a recovery of only 50 per cent. He shows from the test figures that there is much room for improvement.

Recent Metallurgical Patents

Iron and Steel

Unmagnetizable Steel.—A patent for an unmagnetizable manganese-titanium steel of FRIEDRICH KOHLHAAS covers about the same composition as his German Patent 231,499 of Nov. 13, 1909, viz., a steel containing from 9.8 to 10.3 per cent of manganese, 0.9 to 1 per cent carbon, 0.2 to 1.4 per cent titanium, 0.8 per cent silicon, and maximum 0.03 per cent sulphur and 0.015 per cent phosphorus. The German patent gives the silicon as 0.4 to 0.7 per cent. Steel of from 10 per cent on of manganese has a very low magnetic permeability: it can be magnetized only with extraordinary difficulty, being practically unmagnetizable. Such steel, however, is known to be too hard for being worked and its toughness is too low, the elastic limit being situated pretty near the breaking point. It has been found that this same steel can be successfully produced by adding to it an admixture of titanium in the above named proportions. The new product can be treated with comparative ease; its elastic limit is 47 per cent of its breaking strength, it is less magnetizable than pure nickel and cobalt, and it will lend itself more readily to the various uses of unmagnetic material. (13,440, reissued July 9, 1912; original 981,575 of Jan. 10, 1911.)

Gas Washer.—MR. MARK W. JOHNSON, JR., of Birmingham, Ala., has designed a new gas washer based on the idea of mixing the gas with a tangentially introduced spray of water and leading this whirling mixture of gas and vapor under and in close contact with a thin sheet of water streaming down the outer surface of a flaring nozzle. Water and gas then receive a strong centrifugal action in a circular ring chamber by means of which the outer stratum of water containing the concentrated impurities, such as dust, cinder, etc., can be diverted onto a hopper and from here to a separator. The cleaned gases bubble up through the center of the device and can be carried off. The separator has a suitable drain opening for blowing off mud and a port through which another part of the cleaned gas flows into the pipe line for use. The first named water spray is produced by a needle valve, which is described, together with the other details of the invention, in six special drawings. (1,034,403, Aug. 6, 1912.)

Various Furnace Constructions

Crucible Furnace.—The following arrangement of burner for heating a cylindrical crucible furnace with either gas or a liquid hydrocarbon, is claimed by DAVID REISINGER STEELE, of Curtis Bay, Md. Near the bottom of the cylinder is a housing projecting laterally from the cylinder to be used as a combustion chamber. It forms a passageway which communicates tangentially with the furnace chamber. A burner is arranged so as to direct the burning gases or the atomized liquid fuel crosswise or transversely into this passageway, the air entering through another pipe at right angles to the flame issuing

from the burner. The air thus delivered will mingle with the burning gases, divert them from their course and carry them into the furnace chamber. Owing to the tangential arrangement, they will be caused to whirl around the crucible on their way to the top of the furnace. In the case of liquid fuel a branch of the air pipe is connected to the burner in order to produce a spray of the combustible. The furnace can be mounted on trunnions and the pipes connected through a swivel joint to the stationary pipes. (1,020,090, June 11, 1912.)

Air-Cooled Open-Hearth Furnace.—Mr. HENRY KNUTH of Monterey, Mexico, designs a series of spaced air passages which extend transversely and in a straight line through the port arch of open-hearth furnaces. The ends of the passages are left open to have them visible from the furnace floor. The circulation of air through these passages is controlled independently, and in a number of claims it is described how these passages can be connected with each other and with inclined passages leading in various ways, as, for instance, from the front and rear walls upwardly to the side walls of the gas port. A metallic lining is inserted into the inclined portion of lower passages and the circulation of a cooling medium forced through. (1,044,788, Nov. 19, 1912.)

Eyesight Plate for Twyers.—The eyesight glasses in the peepholes of blast furnaces mostly become dirty and fouled and are exposed to the high heat of the furnace. Mr. JAMES B. ROGERS of Ashland, Ky., overcomes this trouble by fastening to the cap of a twyer a rotatably mounted plate provided with a number of holes of the same or larger sizes as the peephole in the cap. Glasses can be inserted into the holes of the plate. By rotating this plate or shutter, the peep sights or glasses, or the larger opening can be brought into or out of register with the opening in the cap or furnace member. If necessary, the same holes can be used for introducing through same a pyrometer or pipes for gas, steam, etc. The invention is equally suitable for all kinds of furnaces in which it is necessary or desirable to have a peephole. (1,041,189, of Nov. 26, 1912.)

Hydrocarbon Furnace.—Mr. JOHN C. SCRIVNER of Madison, Ill., claims the construction of an oil heating furnace into which the burner is introduced from one side near the bottom. While other furnaces mostly have a fire bridge built on the bottom of the furnace, the peculiarity of this invention consists in the roof and side walls of the furnace being slotted, the slots being disposed downwardly and forwardly toward the burner. A removable bridge wall extends through these slots from the top toward the burner, and ends in close proximity thereof, somewhat above the body portion of the furnace bottom. This bridge wall can be a metallic one. The arrangement of the blast wall under an angle is supposed to distribute the heat more evenly. (1,003,522, Sept. 10, 1911.)

Reversing Valve.—Mr. HARRY BURNLEY ROSE of Johnstown, Pa., claims the invention of a new style of something similar to the Forber valve. One of the advantages of his swinging, oscillating type of valve is that it possesses means whereby it may rest and be shifted from one position to another by mechanism which will be submerged in the water-seal when the valve is seated, and thus be protected, while the operating mechanism is entirely accessible and exposed to view on the outside of the hood. This facilitates ease and economy of maintenance and operation. Furthermore the valve can be easily withdrawn and another one substituted without delay, thereby obviating and shortening in time the strenuous labor under the influence of intense heat and noxious gases. (1,045,749, Nov. 26, 1912.)

Mr. E. A. Grönwall, of Ludvika, Sweden, who is well known also in this country, was married on January 27 to Miss Märta Rydberg of Stockholm. Mr. Grönwall, jointly with Messrs. A. R. Dindblad and O. Stallhane, started the experimental work on the smelting of iron ore in the electric furnace, which was later taken up by the Jernkontoret and led to a new electrometallurgical industry in Sweden and Norway.

Producer Gas as a Substitute for Oil for Furnace Work.

The constantly increasing price of fuel oil, coupled with the assurance of the oil companies that such oil will never again be an economical proposition for commercial fuel purposes has brought both producer gas and the use of coal dust prominently to the front as economical and efficient methods of manufacturing a substitute for oil for all operations where high and low heats are essential.

The use of coal dust is noted in a separate article. In this article some interesting figures are given on the use of producer gas for furnace work as made by the Loomis-Pettibone gas producer, built by the Power & Mining Machinery Company, of Milwaukee.

The Loomis-Pettibone system manufactures a clean, fixed gas from bituminous or anthracite coal and coke, and, in addition, wood and other refuse containing carbon may also be gasified.

In the Loomis-Pettibone system a water gas containing 250 to 275 B.t.u. per cubic foot is manufactured intermittently with the production of a lean producer gas, containing from 90 to 105 B.t.u. per cubic foot. The water-gas can be used for all purposes where high temperatures, without regeneration, are necessary, as in manufactories carrying on a large variety of brazing, small forge work, etc., where the furnaces are small, and distributed over a large area. Temperatures ranging from 2500 to 2900 deg. Fahr. are easily obtainable with this gas, and with properly constructed furnaces, it is possible to contain an added efficiency in operation so that the quantity of B.t.u.'s in the gas used will be only 66 to 80 per cent of the B.t.u.'s furnished in oil as used in approved oil furnaces for the same purposes.

Water-gas does not cause the metal forged to scale like oil, and with it it is possible to get much closer regulation of furnace temperatures.

By mixing a small portion of the water-gas produced with the lean producer gas, a mixed gas of from 125 to 135 B.t.u. is obtained, which gas may readily be used for operations such as annealing, tempering, case-hardening, babbiting, and all such purposes, where temperatures to be obtained do not exceed 1700 deg.

One long ton of good bituminous fuel when gasified by this system will generate 40,000 cu. ft. of water-gas and 120,000 cu. ft. of producer gas, as described above, and there are installations in daily operation which exceed these figures as to output.

If conditions are such that it is possible to install gas engines, part of the lean producer gas can be used to furnish power.

In cases where it is impossible to make use of the producer or mixed gases of the Loomis-Pettibone system, where the heats required are all at high temperatures the Power & Mining Machinery Company recommend their blue water-gas apparatus, operating on anthracite coal and coke. This apparatus makes a water-gas containing 330 B.t.u. per cubic foot from the fuels specified above, and, when equipped according to this system with high-pressure blower, steam drying and distributing apparatus, quick-operating valves, etc., will produce 50,000 to 60,000 cu. ft. of such gas per ton of fuel gasified when operated properly according to instructions. This gas possesses all of the qualities of the water-gas made in the Loomis-Pettibone apparatus, and, in addition, has a higher heating value per cubic foot.

In the present note it was merely intended to give figures of the results obtained in commercial practice in order to provide a basis for estimating the possibilities of gas in competition with crude oil. As to the description of the construction of the Loomis-Pettibone Gas Producer and Blue Water-Gas Plants reference may be had to Bulletin 100 of the Power & Mining Machinery Company, of Milwaukee, which is entitled "Loomis-Pettibone Gas Generating System, Gas for Furnace Work."

Pulverized Coal as a Fuel

By A. W. Raymond

The recent increased cost of crude oil has turned the attention of fuel oil users on pulverized coal as a substitute, instead of reverting to the old producer gas plant, and experiments which have been carried on along these lines have demonstrated not only its adaptability and efficiency, but also its economy. When properly pulverized coal will yield higher temperatures and more B.t.u. per ton than can be obtained from a like amount of coal used in a producer gas plant while the equipment is about one-third the cost.

The method of burning the coal dust is very simple. A pipe conducts a blast of air into the furnace and the coal dust is automatically introduced into the air at a point about one foot from the nozzle of the pipe. Any method of feeding the coal, such as a screw conveyor, which will insure a regular and uniform feed will be found satisfactory. The temperature in the furnace and the length of the flame is very easily controlled by the velocity of the air blast and the quantity of coal dust mixed with it. In order to get complete combustion of the coal to carbon dioxide the mixture of air and coal dust must contain at least four times the calculated amount of oxygen required.

For the benefit of those readers who are in doubt as to the adaptability of coal dust as a fuel, the writer would like to say that to his personal knowledge pulverized coal has been burned successfully for the last ten years in a number of large plants. Among these plants are the Canadian Copper Co., at Copper Cliff, Ont., Canada, where powdered coal is burned in the smelting furnaces, and the Chicago and Hamilton, Ont., plants of the International Harvester Co., who use it as the source of heat for their annealing ovens. These plants pulverize the coal to such a degree of fineness that 98.5% of it will pass through a 100-mesh sieve.

The writer has also recently had occasion to come in personal contact with some experiments on a regenerative open-hearth furnace. Two heats of open-hearth steel have been made with powdered coal as a fuel and very even and high temperatures were maintained with coal pulverized so that 91% would pass a 200-mesh test sieve. The checkers showed no sign of plugging and the steel was as free from sulphur as when producer gas or fuel oil is used. For these tests the coal was ground on a pulverizer depending on screens to get a fine product and it was clearly demonstrated that the cost of grinding the coal to the necessary degree of fineness on this type of machine was prohibitive, so the reader can deduce that all that is needed to make the use of pulverized coal a regenerative type of open-hearth furnace a success is a method of pulverization which will yield a product as fine, or, better still, even finer than 91% through a 200-mesh test sieve.

The knowledge which has been gained from investigation, experiences, and comparative tests shows the Raymond four-roller mill, fitted with what the Raymond people call their vacuum air separator, to be an excellent method of pulverization which will answer the purpose. Averaging these tests which cover a period of over a year's time on fourteen pulverizers, the cost per ton of pulverizing the coal so that 99.5% will pass a 100-mesh test sieve amounted to \$0.325. This figure includes the power used at a cost of 1.5 cents a kilowatt hour and shows a saving of \$0.211 per ton over any other method with which it was compared. Though the above tests only proved the necessity of a finely pulverized coal on a regenerative type of furnace, the writer holds to the belief and feels confident that the reader will agree with him that the highest efficiency in any or all cases can only be obtained when the coal is ground extremely fine and uniform. Let us now take up the process of properly preparing the coal for the burners step by step.

The coal must first be crushed so that it will all pass through a ring $1\frac{1}{4}$ " in diameter, as there is no machinery made which will reduce the ordinary mine run of coal to the required fine-

ness in one operation. Though not absolutely essential, it is generally conceded that the crushed coal should be dried thoroughly before pulverizing or burning, as dried coal can be pulverized with 15% less horsepower than the wet, and it is almost axiomatic to say that dried coal is more easily ignited and consequently gives higher temperatures and less ash. Actual experience has proven that drying the coal before pulverizing or burning it is a money saving practice. Excellent drying apparatus is manufactured by the Ruggles-Coles Engineering Co. After the coal is thoroughly dried it is ready to be fed into the pulverizer and the necessity of using the right kind of pulverizing machinery cannot be too strongly emphasized.

The pulverization of coal entails no danger from spontaneous combustion, providing the necessary precautions are taken. The pulverizer used must be one which does not heat up the coal; that is, one in which the friction is reduced to a minimum, and the danger of sparking against any iron in the coal is eliminated. A ball mill is impractical because the pusher plates behind the balls get very hot and will invariably cause the coal to take fire.

The first step is to select a pulverizer which delivers a cool product so that it can be safely stored for a day or two. Investigation will show the Raymond mill, before referred to in this article, to be such a pulverizer. This mill has another very important advantage in that the coal is never allowed to settle in the grinding chamber and for this reason the danger of an explosion in the pulverizer is entirely eliminated. These mills have been grinding coal in the United States and Canada for the last ten years and there is no record of any explosion or fires in, or resulting from, them. There is also another point to consider in the selection of a pulverizer.

The efficiency of combustion depends entirely upon the degree of pulverization. In other words, the finer the degree of pulverization the more economical the operation, the higher the temperature obtained and consequently a low percentage of a finely divided ash. In a regenerative type of furnace a low percentage of ash in a very finely divided state is absolutely essential, as anything else would clog up the checker work. Therefore, the coal must be pulverized very fine, the degree of fineness being determined by the cost of grinding as this cost must be kept as low as possible in order to make a powdered coal burning installation more economical than a producer gas plant. Here again we find the Raymond mill peculiarly adapted to coal grinding. The capacity is not limited, as in the case of a screen pulverizer, to the capacity of the screens, but every atom of power put into the mill is used in grinding the oversized coal, as the pulverized coal is removed immediately by the vacuum air separator. This principle is the very essence of economy as there is no waste of power.

Not only must the coal be very fine, but it must also be very uniform. Nothing but a very uniformly pulverized coal will maintain the steady and even temperatures which are required in almost every case. Particles of coal too large to pass an 80-mesh mean a resultant ash which contains a great many unused British thermal units, consequently, economical operation requires a very uniform degree of pulverization, which is impossible to obtain in a screen machine. A test made in the coal house of the Huron Portland Cement Co.'s plant at Alpena, Mich., and covering a period of six days showed that the coal ground on Raymond mills did not vary more than 0.2 in the percentage which passed a 100-mesh test sieve; in other words, the coal was ground so that never less than 98.4% or more than 98.6% would pass a 100-mesh test sieve.

The next question is, what is the cheapest method of pulverizing? In the writer's opinion the following figures obtained from the Omega Portland Cement Co. speak for themselves. These figures are averaged for a year's time and the coal was ground so that 98.5% would pass through a 100-mesh test sieve. In the one case the coal was ground in a Raymond mill and in the other it was pulverized in a hammer cage mill and finished on a tube mill. The cost of operation and repair, not including power, on the hammer cage and tube mill com-

bined was \$0.336 per ton, while on the Raymond mill only \$0.125 per ton, and the power required on the Raymond mill per ton of coal ground was a little less than half that required for the other method. Other tests comparing the Raymond mill with a ball mill used in the place of the hammer cage have substantiated the above figures.

The use of the Raymond mills in other large cement plants has resulted in a saving of two-thirds the labor and an increase of 12% in efficiency of combustion due to the high percentages of impalpable powder which is obtained by the Raymond system. One cement plant was able to make 13,500 tons of coal ground on Raymond mills, burning the same amount of clinker, as 15,120 tons ground by a different method would burn.

In brief, then, the installation for properly preparing the coal most economically should consist, in the writer's opinion, of gyratory crushers, Ruggles-Coles dryers and Raymond roller mills with vacuum air separators.

A New Electric Speed Indicator and Recorder

The Brown Instrument Company and the Keystone Electrical Instrument Company of Philadelphia, Pa., are placing on the market a new electric tachometer or speed indicator to accurately indicate, or record on a chart, the revolutions per minutes of a rotating shaft or wheel.

There are many operations where an accurate speed indicator is desired to be permanently connected to an engine or machine, and to show at a glance the speed at which it is running, or



FIG. 1.—ELECTRIC SPEED INDICATOR

to keep a constant record of this speed on a chart for the whole 24 hours.

The electric tachometer has the advantage of not only being exceedingly accurate, but that the indicating or recording instrument can be located at any desired distance from the shaft, the speed of which is to be measured. The instrument consists of a small generator, particularly designed for use with this tachometer. It generates 25 volts at 1000 revolutions per minute. On account of the voltage generated, an instrument, absolutely dead beat in operation, can be supplied. Instruments have been particularly designed by the above companies of unusually high resistance, exceeding 3000 ohms in each instrument, so that the length of leads connecting the magnet or generator and the indicating instrument have no effect on the

readings. For the same reason one, two or three instruments can be connected to the same generator without affecting the indications. This high resistance feature is exceedingly advantageous, for oftentimes after the generator and one indicating instrument have been installed, it is desirable to place a recording instrument in the office, or at another convenient point, and this can be connected to the generator without recalibration of the instrument.

In Fig. 1 is shown the standard Brown electric tachometer.

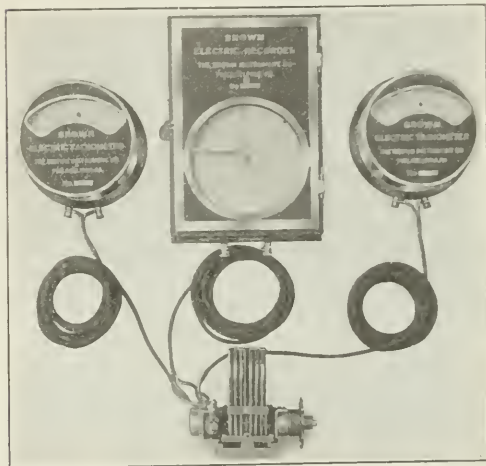


FIG. 2.—GENERATOR WITH TWO INDICATING AND ONE RECORDING INSTRUMENTS

generator, leads and indicating instrument as usually supplied while in Fig. 2 is shown a group of instruments connected to a generator driven by sprocket and chain, two indicating instruments and one recording instrument being connected to the same generator to both indicate and record the speed at three different points.

With many machines it is desirable to operate them at a certain speed to produce the best results, as for instance, in large machine tools it is desirable to know the cutting speed, and by use of this electric tachometer mounted beside the machine the exact speed can be noted at all times, and maintained as desired.

At all blast furnace plants where gas or steam blowing engines are in use it is necessary to run these engines at a constant speed so that a certain number of cubic feet of air per minute can be blown into the furnace. The revolutions per minute in cases such as this are of exceedingly great importance, and with the Brown electric tachometer with one indicator mounted near the blowing engines, on the engine itself, or on a switchboard nearby, and with a recording instrument if desired in the Master Mechanics' office, a perfect regulation and record of the speed can be secured.

There are many tests which constantly have to be made where an electric tachometer of this character will prove particularly advantageous. As an instance, we might cite the locomotive tests which are run by several universities and larger railroads which have testing departments in which the speed of the driving wheels can be readily determined.

Another use of this electric tachometer is on war vessels and merchant ships where it is particularly desirable not only to indicate the speed of the propeller shaft in the engine room, but also frequently at the same time in the chart house and on the bridge.

A bulletin on this new electric tachometer which has some radically improved features may be obtained from the manufacturers.

A New Temperature Recorder

At the New York Automobile Show which closed last week we had the pleasure of making the acquaintance of a temperature recorder of striking novelty. It is built by the Leeds & Northrup Company, of Philadelphia.

While it is new in so far as it has not been described before, yet it has been in use in various plants under quite different conditions in order to test it thoroughly in practical use. The development of the instrument covers more than five years. A large number of models was made and rigorously tested. In

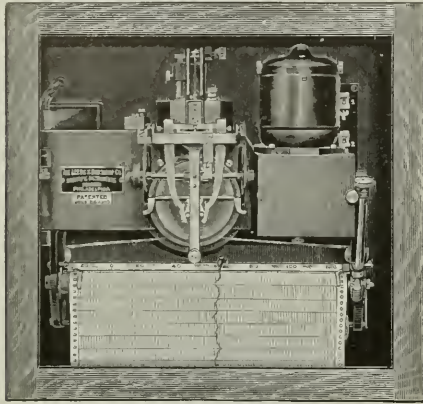


FIG. 1.—TEMPERATURE RECORDER

addition to tests of complete instruments, every important part was subjected to prolonged tests for wear and other objectionable features. In addition to tests in their own laboratory the Leeds & Northrup Company had a number of instruments of each type in operation in the shops of large steel mills and in other works where they could be watched under such exacting conditions as are associated with temperature measurements. The National Bureau of Standards used one of these recorders on shipboard of the scout cruiser *Salem*. The instrument in question was sensitive to 0.05 deg. C. and proved perfectly satisfactory. If memory serves us right, the boat in question is considered one of the shakiest in the navy.

One striking novelty of the design is shown in the outside appearance of the apparatus in Fig. 1. It is the fact that the whole mechanism of the recorder is exposed to view and inspection. The whole apparatus consists of standard parts. These parts are in general of dimensions and strength comparable to the parts of a sewing machine. The design is such that each part may be removed and replaced easily. If necessary, a recorder could be entirely dismantled and reassembled with the aid of two screwdrivers and two standard wrenches. If a part is damaged a duplicate standard part can be inserted. These recorders have

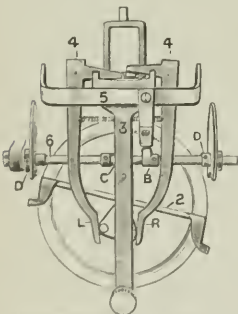


FIG. 2—DETAILS OF MECHANISM

been developed to take down a single curve or several curves at the same time. In this article we deal with the single-curve recorder only.

These recorders can be used either with resistance thermometers or with thermocouples. If the recorders are used with a resistance thermometer the connections are those of a Wheat-

stone bridge. If the recorder is used with a thermocouple the connections are those of a potentiometer.

In these recorders the automatic balancing of the galvanometer of Wheatstone bridge is accomplished in the manner shown in Figs. 2 and 3. The motor drives the shaft (6) through a worm and gear at a speed of about 30 revolutions per minute. The shaft carries four cams (6B), (6C), (6D), (6D'), so located on the shaft that at the same time that (6B) raises the rocker arm (5), (6C) carries the clutch arm (2) away from the clutch disk (1).

The galvanometer, except at periodic intervals, swings freely about the location shown in Fig. 3. When the galvanometer is balanced the pointer lies at the position shown in Fig. 3, and as the rocker arm (5) is raised it lifts the pointer into the space between the two levers (4R) and (4L). If, however, the galvanometer is unbalanced, the resultant movement is shown in Fig. 2, the one lever or the other being lifted due to the interposition of the pointer between the lever and the rising rocker arm. The wings on the clutch arm can clear the cams 6DD' only when the clutch arm is horizontal, consequently, after being set as in Fig. 1, subsequent rotation of the shaft (6) returns the mechanism to the position shown in Fig. 3. When the

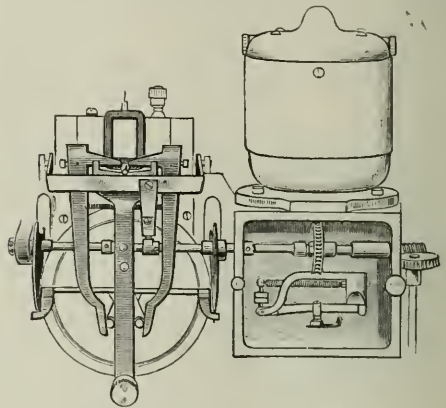


FIG. 3.—DETAILS OF MECHANISM

clutch arm is being tilted it is free from the disk; when it is returned it is in engagement with the disk and the two turn together.

Mounted on the disk shaft is the circular slide resistance. Also there is present on the shaft a wheel for communicating all motions of the shaft to the pen or print-wheel making the record. The movement of the disk, as described above, is always in such direction as will tend to restore a balance in the potentiometer or bridge circuit and the amount of movement is approximately proportional to the deflection of the galvanometer.

A pamphlet giving further details of the instrument may be had from the manufacturers.

International Commission on Annual Tables of Constants

The following gentlemen have agreed to serve on the American Advisory Committee of the International Commission on Annual Tables of Constants:

Dr. W. D. Bancroft, Dr. Carl Barns, Dr. A. L. Day, Dr. A. A. Michelson, Dr. W. A. Noyes, Dr. Ira Remsen, Dr. Jos. W. Richards, Dr. Theo. W. Richards, Dr. Alexander Smith, Dr. S. W. Stratton, and Dr. W. R. Whitney.

The first volume of the Annual Tables of Constants appeared last year and a review of it was published on page 572 of our volume X. The second volume will be out in the near

future.

Special Mixing and Grinding Machinery

The adjoining four illustrations show several mixing and grinding machines with various novel features.

Fig. 1 is a view of a special mixing machine designed for handling a pulpy mass. The machine illustrated had an extra special feature, as it was one of several designed and built for



FIG. 1—SPECIAL MIXING MACHINE FOR A PULPY MASS

handling a material which should not come in contact with iron, copper, steel, etc. These machines were constructed very heavy, water-tight in every respect, and all parts that could come in contact with the product were very heavily coated with tin.

The machines were equipped with special stuffing boxes, had bronze bushings in the bearings, and were equipped with quick-discharge valves of a special type. All surfaces were free from projections, so that they could be kept perfectly clean. These machines were approximately 30 in. in diameter by 7 in. long. They were sold with a guarantee of 1500 gal. capacity per hour, but actually produce 3000 gal. per hour each, with the use of 50 per cent less horsepower than guaranteed.

A Special Built Ball Grinding Mill

Figs. 2 and 3 show the first ball-grinding mill built with a seamless steel shell. This machine is really the regular type

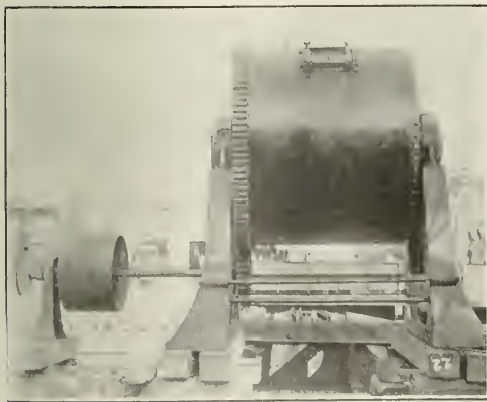


FIG. 2—BALL GRINDING MILL

of pebble mill with the exception that instead of using pebbles for producing the grinding and mixing action, metal balls were used as the material to be handled is of a fluffy nature, and at the same time has an inclination to become sticky. For this

reason it was necessary to have the machine absolutely smooth inside, without obstruction, so that there would be no possibility of the material adhering to any point.

Fig. 3 shows the machine surrounded by a dust-proof discharge casing, which was made practically air-tight, so that either cold air could be circulated between the shell of the mill

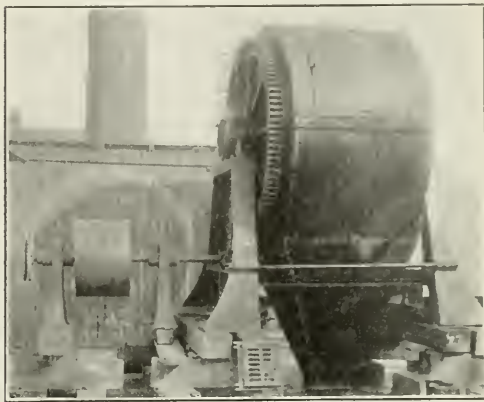


FIG. 3—BALL GRINDING MILL WITH DUSTPROOF DISCHARGE CASING

and the casing or cold water could be run through the pipes surrounding the machine proper in order to keep it cool. The cylindrical part of the machine is 6 ft. diameter by 5 ft. long. The shipping weight was 18,000 lb. Its relative size can be seen by comparing it with the freight car standing behind it.

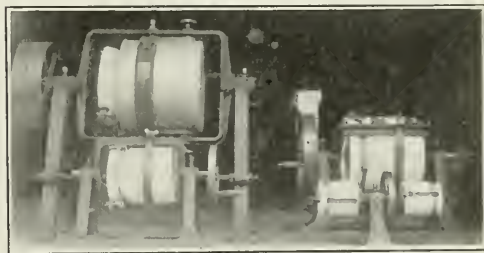


FIG. 4—PORCELAIN JAR MILLS

Fig. 4 shows the relative size of four different types of porcelain jar mills. It also illustrates part of a shipment of this class of machinery made to South Africa several months ago.

All these machines were personally designed by Mr. Paul O. Abbe, of New York City, to meet the special requirements for which they were intended.

The Basket Type Evaporator

For many purposes in chemical engineering the use of the basket type evaporator, developed and patented by the Swenson Evaporator Company, of Chicago, is found to be particularly suitable. The principal feature of this patented type is the inner shell located 12 in. outside the nest of tubes and providing additional heating surface which results in an increase of 8 or 10 per cent in the evaporative efficiency of the outfit.

The main advantage of this construction is not so much the increased heating area thus made available as the influence of this shell upon the direction and vigor of the circulation. Upon starting up an evaporator of this type, the evaporation from the space between the inner shell and the tubes starts almost instantly, resulting in a shooting up of a ring of liquor from this space long before evaporation begins in the tubes. With a

space for return current of about 8 in. between the inner shell and the outer shell of the evaporator, this type of apparatus has a splendid downward circulation rigidly directed into this space.

Sluggish circulation would be very troublesome in handling heavy or salty liquors, as it would result in incrustation of the pipes, resulting in slowing up the work by diminishing the value of the heating surface and not permitting the same concentration with a given temperature difference.

The matter of steam or vapor circulation around the tubes is very important. In the Swenson basket type about 8 per cent of the tubes are omitted in order to provide more generous circulating spaces. In this way the steam reaches all of the tubes as soon as it gets into the basket and both the operation and the wear are much more uniform. This is noticed by the equal evaporation over the entire boiling surface. As particular attention is paid to the removal of non-condensable gases the wearing qualities of the upper parts of the tubes are much increased.

An Interesting Agitator Test in Cyanide Practice.

To Dr. Wilbur A. Hendryx, who recently returned from South Africa, we are obliged for some interesting notes on an official test of the Hendryx agitator on the Rand.

One of the 18-ft. agitators, constructed by Head, Wrightson

then leaching the sands six to seven days, and treatment of slimes by decantation. A few of the more recent plants have installed agitation and filtration for their slimes. The average extraction as near as Dr. Hendryx was able to ascertain by the above methods, including amalgamation, is from 92 to 93 per cent.

In view of this situation, the results of the official test of the Hendryx agitator shown in the adjoining chart are interesting. They were so satisfactory that the Hendryx agitator was installed at the Wolluter Gold Mine, which has a capacity of 1,000 tons per day.

It will be noticed from the report that only one charge of slimes, 95½ per cent minus 120 was treated in two hours with 98 per cent extraction. The other four tests were made on the mixed product of sands and slimes, and it will be noticed by the solution assays that six hours was the longest period to get the extraction, and in charges 3, 4, and 5, that it was practically completed in 3 hours.

Taking these facts into consideration and the results on the slimes it appears quite reasonable to conclude that if these ores were ground to minus 90 mesh, 3 hours would be sufficient in which to treat this product. Most of the mills on the Rand to-day are grinding at least 98 per cent to minus 90.

It will also be seen that the percentage in cyanide loss is extremely small and this would substantiate the claim that

RESULTS OF ACTUAL TESTS MADE BY THE MINES TRIALS' COMMITTEE AT THE WOLLUTER G.M. CO.

TEST NO. AND DATE.	1.—MARCH 11, 1912.	2.—MARCH 20, 1912.	3.—MARCH 26, 1912.	4.—APRIL 9, 1912.	5.—MAY 2, 1912.
PRODUCT...	Sands and Slimes.	Slimes only.	Sands and Slimes.	Sands and Slimes.	Sands and Slimes.
QUANTITY AND RATIO	Ore: 35 tons Solu.: 70 „ } 1:2	Ore: 35-03 tons Solu.: 70-00 „ } 1:2	Ore: 35 tons } Solu.: 71 „ } 1:2	Ore: 35 tons } Solu.: 72-4 „ } 1:2-063	Ore: 35-09 tons } Solu.: 73-69 „ } 1:2-1
GRADING	+ 60 9-4% + 90 19-3% + 120 11-3% — 120 60-0%	— -1% -4% 99-5%	13-75% 17-74% 10-70% 57-81%	5-93% 20-52% 12-0% 61-45%	9-18% 21-10% 11-39% 57-33%
AVERAGE ASSAY OF CHARGE	2-55 dwts.	1-65 dwts.	2-2 dwts.	1-95 dwts.	2-3 dwts.
SOLUTION STRIP PER TON OF ORE—					
Start	-068 × 2 = -136	-021 × 2 = -042	-063 × 2 = -126	-052 × 2 = -104	-0525 × 2 = -105
Finish	-066 × 2 = -132	020 × 2 = -040	-060 × 2 = -120	-050 × 2 = -100	-050 × 2 = -100
Loss	-002 × 2 = -004	-001 × 2 = -002	-003 × 2 = -006	-002 × 2 = -004	-0025 × 2 = -005
EXTRACTION — BY SOLUTION.	Assay × Ratio, dwts.	dwts.	dwts.	dwts.	dwts.
1 hr.	.76 × 2 = 1-52	1 hr. .5 × 2 = 1-0	1 hr. .67 × 2 = 1-34	1 hr. .6 × 2-063 = 1-24	.57 × 2-1 = 1-218
2 hrs.	1-00 × 2 = 2-00	1 hr. .58 × 2 = 1-16	2 hrs. .83 × 2 = 1-66	2 hrs. .67 × 2-063 = 1-385	.65 × 2-1 = 1-365
3 hrs.	1-10 × 2 = 2-20	1 hr. .69 × 2 = 1-38	3 hrs. .84 × 2 = 1-68	3 hrs. .68 × 2-063 = 1-406	.79 × 2-1 = 1-659
4 hrs.	1-15 × 2 = 2-30	2 hrs. .76 × 2 = 1-52	4 hrs. .9 × 2 = 1-80	4 hrs. .79 × 2-063 = 1-633	.85 × 2-1 = 1-785
5 hrs.	1-15 × 2 = 2-30	3 hrs. .76 × 2 = 1-52	5 hrs. .92 × 2 = 1-84	5 hrs. .84 × 2-063 = 1-737	.89 × 2-1 = 1-869
6 hrs.	1-20 × 2 = 2-40		6 hrs. .94 × 2 = 1-88	6 hrs. .84 × 2-063 = 1-737	.98 × 2-1 = 2-058
7 hrs.	1-20 × 2 = 2-40				
% EXTRACTION BY KCY. SOLUTION	2-4 dwts. from 2-55 = 94-1%	1-52 dwts. from 1-65 = 92%	1-88 dwts. from 2-2 = 85-4%	1-737 dwts. from 1-95 = 89-3%	2-058 dwts. from 2-3 = 89-47%
WASHED RESIDUE ASSAY	.281 dwts.	.05 dwts.	.35 dwts.	.25 dwts.	.55 dwts.
COMPARATIVE % EXTRACTION FROM—					
Residue Assay	93-06%	97%	84%	87-99%	76-3%
Solution Assay	94-1%	95%	85-4%	89-3%	89-47%

The above figures represent the exact results obtained by the Mines Trials' Committee (Johannesburg Transvaal) in the Hendryx Agitator upon residues after Amalgamation.

THE FOLLOWING FIGURES SHOW THE TOTAL EXTRACTION FROM THE ORIGINAL ORE, INCLUDING AMALGAMATION IN THE MILL AND WERE FURNISHED BY THE COURTESY OF THE WOLLUTER G.M. CO.

TOTAL EXTRACTION, BY AMALGAMATION AND KCY SOLUTION				
FIGURED FROM RESIDUES	97-6%	99-0%	94-5%	95-5%
FIGURED FROM SOLUTIONS	97-9%	98-0%	94-6%	96-2%
				91-6%
				96-31%

& Company, Ltd., of London, Eng., who are manufacturers and sales agents of the Hendryx apparatus in foreign countries, was sent down to Johannesburg, and turned over to the Mines Trial Committee of the Chamber of Mines of Johannesburg. This committee is composed of high-class mechanical and metallurgical engineers who represent all the groups of mines on the Rand.

The present general method of treatment on the Rand consists of stamping, amalgamating, tube milling, and classifying,

Dr. Hendryx has always made that the loss of chemicals was far less by the use of mechanical agitators than where air agitators were resorted to, as well as the time of extraction being much shorter, and that it also substantiates the fact that it is practical to treat sands and slimes together.

If the continuous process is adopted, the time for filling and discharging is eliminated, and this method of treatment is receiving serious consideration by the leading engineers on the Rand.

The First Combination Induction Furnace in Operation in the United States

By C. H. Von Baur

It was the desire of the Crucible Steel Casting Co. of Lansdowne, Pa., to be up-to-date in making steel for small and intricate castings, which prompted them to adopt an electric furnace.

The type selected is the Roechling-Rodenhauser induction furnace and the eighteenth to be placed in operation. Since then four additional R.-R. furnaces have been ordered by other steel mills. This one, the first to be placed in operation in this country, has a pouring capacity of 2 gross tons and

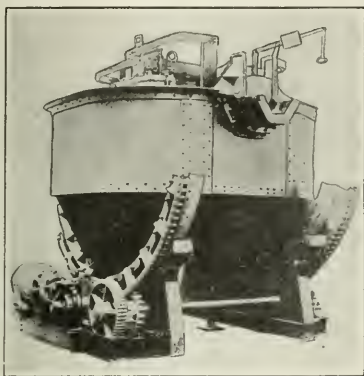


FIG. 1.—2-TON, 300-KW. ROECHLING-RODENHAUSER ELECTRIC INDUCTION FURNACE (FRONT VIEW)

operates at 300 kw from a 25-cycle, single-phase, 480-volt, 75 per cent power factor alternator, driven by a two-cylinder De La Vergne oil engine. This prime mover was decided upon, as it burns not only the heavy Mexican oils 16 to 19° Beaumé, but also the less expensive water gas tar. When running to capacity on single turn (12 hours), the furnace receives its electricity at not over 0.7 cent per kw-hour. The local lighting companies' price for current was more than twice as high as the above rate.

The furnace is 200 feet from the alternator. In order that the heat regulation might be accomplished within sight of and reasonably near the furnace, a remote control switchboard was erected about 30 ft. in front of and slightly higher than the spout. From this platform the furnace heat may be raised or lowered by pressing a button or shut off altogether by similar means. There is also here the controller which operates the motor for tilting the furnace. Electric instruments at both switchboards show at all times the degree of heat in the furnace. The blower motor supplying air for cooling the furnace transformer coils consumes 4.95 kw.

The bottom consists of magnesite and tar tamped in with pneumatic rammers. The roof is made of magnesite brick and lasts indefinitely.

The first steel was made in November 18, 1912. Since then a second lining has been put in. This was baked in more carefully than the first, taking about 24 hours at 50 kw. At the end of the fifth day's pour absolutely no sign of wear could be seen in the channels and only a slight scarification in the central hearth. The side walls usually last 3 weeks before thorough repairs are made.

At present it is the object to charge only cold metal, melt it and refine it. Room has been left for a preliminary melting furnace, to be installed when the capacity of the electric with cold charging on single turn has been reached. The furnace has a capacity of 1200 tons annually with cold charging,

and 2850 tons with hot charging, both on single turn.

One gratifying point about this type of furnace is that good steel can be made and was made with old open-hearth melters, right from the start, the reason being that the melter could keep his whole mind on making steel, without being distracted by an irregular heat supply, spot heating, frozen corners, or broken electrodes.

One important point in connection with the use of this electric furnace is the possibility of avoiding defective castings. In one run made there were no defective castings which had to be welded with the oxy-acetylene or the electric arc. Several castings have been sawed through and in every case they were found to be free from blowholes. Disregarding for the moment the fact that electric steel in the ladle costs less than crucible, small converter, or even steel made in the open hearth, either acid or basic, there is no doubt a large additional saving effected on account of the defective castings being reduced to a minimum.

Assuming that the molds are always in good condition, there are several causes which enable good castings to be made. *First*, the steel is thoroughly deoxidized and pours as well if not better than the best crucible steel. *Second*, while the steel has any desired temperature it is not "blowy," as converted steel is apt to be, nor do the castings suffer from "heat" or sulphur cracks. *Third*, a thoroughly reducing action is easily obtained toward the end of the heat without reducing the temperature as in the open-hearth.

The results obtained applied to a concrete case will show how great this saving is. A well-known foundry making miscellaneous small and medium size steel castings, bench work, etc., acknowledged the loss of 6% due to some of the above causes.

This foundry poured about 2400 tons annually, of which about 65% resulted in castings. The greater portion being small work, the average price was slightly over 10c. a pound. Electric steel in foundries has been known to reduce the "defects" to one per cent. The possibility of such a foundry adopting electric steel would result in sending 78 tons of castings more from the shipping room with the same output poured, and increase the profit \$17,500—a large additional interest on the investment.

An interesting feature when melting scrap in the electric induction furnace, is that the time required for a heat is reduced as the electric energy at the furnace terminal is increased; that is, the shortened melting time raises the attain-



FIG. 2.—VIEW OF FURNACE AFTER ERECTION

able efficiency of the furnace. There is another case of an induction furnace—a one-and-a-half-ton Kjellm—where a mixture of pig and scrap was melted with 060 kw hours per ton over a six-hour melting period, having 160 kw at the terminals, whereas 800 kw hours only were consumed during a four-hour period having 200 kw at the furnace. This gives a total efficiency of 50% for the six-hour melting time, and 60% for the four-hour melt.

In the accompanying figure the upper dotted line shows these conditions and proves the following formula:

$$\text{Tons melted} = \frac{\text{kw at furnace} \times \text{hrs.}}{\text{kw hours per ton.}}$$

The power should be kept at the furnace at a uniform rate.

This brings us to the case under discussion, where with a two-ton furnace a mixture of 1.38 tons consisting almost exclusively of scrap was melted and brought to a temperature as before indicated with 844 kw hours per ton in four hours and fifty minutes, or—

$$1.38 T = \frac{241 \text{ kw} \times 4.83 \text{ hrs.}}{844 \text{ kw hours per ton.}}$$

Drawing a parallel line to the foregoing case we reach the point the two-ton furnace has been designed for, namely, to produce 1.5 tons in 3.75 hrs. with 300 kw at the furnace terminals, or—

$$1.5 T = \frac{300 \times 3.75}{750}$$

Later on when a preliminary melting furnace of one form or another is installed, the output will be greatly increased, as 275 kw hours per ton will be sufficient to refine and degasify ordinary mixtures of pig iron and scrap. This would give a 10-ton output in 12 hours, greatly decrease the cost for electricity, and for the bottom, per ton of output, and decrease the overhead charges to 45% compared to cold charging.

The quality of electric steel made is the equal of crucible or

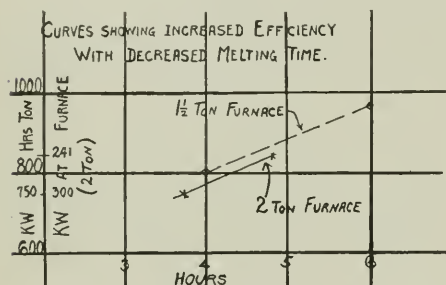


FIG. 3.—MELTING TIME AND EFFICIENCY

better. This high grade of deoxidized steel was made with about the following cost per ton:

Raw material (scrap).....	\$14.50	
Oxidation loss about 2%.....	.29	
		\$14.79
Current, 844 kw hours, at 7c.....	\$5.91	
Fluxes	.76	
Ore	.05	
Labor	1.50	
Tools, repairs and bottom.....	.76	
Cooling air for transformers.....	.12	
		9.10
Conversion cost		
Interest 5%, depreciation 10% per ton, about		1.50
Cost of one gross ton electric steel ready to pour	\$25.39	

When larger amounts of scrap are charged at a time, thus lessening the radiating losses due to the doors being open, and when 280 to 300 kw. is at the furnace instead of 250, the time for melting and deoxidizing will be shortened considerably, and the kw hours per ton reduced to about 750, which is near the European record for similar work in this size and type of furnace, exclusive of the heat required for baking in the bottom, and heating at night on single turn. It required 125 to 130 kw continuously to keep a full charge at temperature.

According to various calculations and tests, the amount of theoretical electrical energy necessary for melting down one gross ton of 2/3 scrap and 1/3 pig iron is given at 480 kw hours,

at a tapping temperature. This figure has also been given as 430 kw hours for mild steel when tapping at 1650° C.

The total electrical and thermal efficiency based on the observed radiating losses, with this furnace, is

$$100 - \left(\frac{130 \times 100}{300} \right) = 56.6\%$$

This figure checks up very well with conditions of this heat, which was hot enough, as already mentioned, to melt a 1 1/4-in. rod in 15 seconds, indicating a temperature well above 1650° C. If 430 kw hours per ton for melting is taken as the correct figure, no doubt 50 kw hours should be added to this for the additional superheating, which was estimated to be between 100 and 125° C. It takes about 40 kw hours to raise the temperature 100° C. Consequently

$$\frac{480}{844} = \text{approx. } 56\% +$$

The operation of this electric induction furnace, both economically and practically, is eminently satisfactory. For the past 12 years a Kjellin induction furnace has been in operation making tool steel, where its field in displacing the crucible is limited. For four years past it has been used in regular foundry operation, where a greater tonnage awaits it, but its largest field is in finishing open hearth and especially Bessemer steel for rails, as these have been found unbreakable in service.

The above furnace installation, including the generator, etc., was made by the American Electric Furnace Co., New York, the plate work and rolling portion by the Wellman, Seaver, Morgan Co., Cleveland, O., and the electrical parts by the General Electric Co., Schenectady, N. Y.

Obituary

Professor George A. Koenig, who was formerly a member of the University of Pennsylvania faculty, died on Tuesday, January 14, shortly before midnight, at the home of his son, Dr. Augustus Koenig, in Philadelphia. Professor Koenig was the oldest member of the faculty of the Michigan School of Mines, with which he had been connected as Professor of Chemistry since 1892; on January 7 he arrived in Philadelphia with his wife just a few hours before the death of his daughter, Miss Hilda Koenig.

Dr. Koenig is survived by his widow and two children, Dr. Augustus Koenig, a physician and metallurgist, and Mrs. George E. Nitzsche, an artist and portrait painter.

Dr. Koenig was assistant professor of chemistry and mineralogy at the University of Pennsylvania from 1872 to 1879, and professor of mineralogy and geology from 1879 to 1892, and professor of chemistry at the Michigan School of Mines since 1892. At the University of Pennsylvania he gave the first course in mining that was ever given in any educational institution in the United States. He took out a number of valuable patents. In 1897 he invented a new method of assaying without muffle, and later he invented an assay furnace to carry out these methods. In 1881 he patented a method for chlorination of low-grade silver and gold ores, and in 1911 a method of extracting vanadium from ores containing vanadium by which 95% of the vanadium could be recovered. He has prepared artificial crystals of arsenides, of which no natural crystals have ever been found. In 1885 he published his chromometric methods, a development of blow-pipe analysis, and in 1905 an elementary book of chemistry.

Dr. Koenig had a large and valuable collection of minerals which he presented to the Michigan School of Mines. Among the new minerals discovered and named by him are: Hydrotitanite, Randite, Leydyite, Alaskaite, Begerite, Rementite, Footeite, Paramelaconite, Mezapilite, Mohawkite, Keeweenawite, Stibiodomykite, Melanochalcite. He has also re-examined many other minerals and discovered diamond in meteoric iron.

Professor Koenig was born at Willstatt, Germany, on May 12, 1844. He was educated at the gymnasium at Kork, the School of Moravian Brothers, Lausanne, Switzerland, and at

the Polytechnic School, Karlsruhe, Germany, where he received his degree of Mechanical Engineer. From 1863 to 1865 he was a student at Heidelberg, and from 1865 to 1867 at the University of Berlin, where he received his degrees of Master of Arts and Doctor of Philosophy. Subsequently he attended the Mining Academy at Freiberg, Saxony, for a year.

He came to the United States in October, 1868, and manufactured sodium stannate from tin scraps in Philadelphia. He subsequently became the chief chemist of the Tacony Chemical Works of Philadelphia, and during the winter of 1870 he examined for the Lennig Company several old silver mines in Mexico, traveling much of the distance by stage coach, and enduring many hardships, and having several exciting encounters with the Apache Indians.

Dr. Koenig was held in high esteem, not only for his brilliant scholarship, but for his character and personal worth. He had those genial and magnetic qualities which won for him life-long friendships, not only among thousands of students who came under his charge during the forty years of his career as a professor, but also by his associates and co-workers in scientific fields. He was noted for his fairness and impartiality.

When he left Houghton, only a week before his death, the *Houghton Mining Gazette* printed the following editorial, in which is expressed the sentiment of those with whom he had lived for twenty years:

"The sympathy of the entire community goes out to Dr. and Mrs. Koenig in their bereavement. When the doctor left the other day to find relief and recuperation from his own physical ailments it was with sadness and sorrow that the local public learned that a leave of absence was necessary. And when this wholly unexpected blow, the death of his charming daughter, comes to add to his weight of sorrow the burden seems all the more difficult to carry.

"It is with no feeling of disrespect to the other members of the faculty of the Michigan College of Mines when we say that no man connected with that institution has such a place of love and friendship in the hearts and minds of the graduates and students and townspeople of Houghton as that held by Dr. Koenig.

"Standing high in his rating as a professional man, he stands still higher in the estimate of those qualities of personal charm that make a gentleman known and respected and admired by his fellows.

"We say these few words to let Dr. Koenig know that the sympathy and kindly regard of the whole people is with him in his present bereavement and to express the hope and belief that his wonderful mental strength will give him power to bear up and return to us in the spring renewed in vigor and physical health."

Personal

Mr. Kosaku Asano, metallurgist for the Ashio smelter, Japan, has been visiting American metallurgical works.

Mr. Gelasio Caetani, metallurgical engineer of San Francisco, will deliver a course of lectures at Harvard University in February, after which he expects to go to Europe for a short time.

Mr. A. R. Campbell has been appointed assistant manager of the Dearing, Kan., plant of the American Zinc, Lead & Smelting Company. He was formerly superintendent of the Needles Smelting & Refining Company, Needles, Cal.

Mr. Franz Cazin, metallurgical engineer of Denver, has been engaged in professional work in Mexico.

Mr. F. L. Clerc, of Boulder, Colo., is at Rockport, Texas, where he will stay until about March.

Mr. William Brown Dickson, recently elected president of the International Steam Pump Company, was born in Pittsburgh, Pa., on Nov. 6, 1865, son of John and Mary A. (McConnell) Dickson. He was educated in the public schools of Pittsburgh, Pa., commenced work at an early age and, in 1881,

secured employment at the Homestead steel works, which had just been built and placed in operation. After several years passed at manual labor in the mills, he was transferred to the office, acting at first in a clerical capacity, but subsequently received rapid promotion through various grades up to that of assistant to president and managing director of the Carnegie Steel Company. He became a junior partner in the Carnegie Steel Company, Ltd., in 1899. On the formation of the United States Steel Corporation in 1901 he accompanied Mr. C. M. Schwab to New York as assistant to the president of the United States Steel Corporation and later was elected a vice-president of that corporation, which position he held until May 1, 1911, when he resigned in order to retire from active business. Mr. Dickson was elected president of the International Steam Pump Company on Nov. 19, 1912. He succeeded the late Benjamin Guggenheim, who went down on the *Titanic*. Those who are familiar with Mr. Dickson's record in the steel industry do not hesitate to say that the International Steam Pump Company made a wise selection.

Mr. John V. N. Dorr, metallurgical engineer of Denver, spent the month of January in a trip to Central America, taking the West Indian cruise and visiting Panama.

Mr. John Howe Hall, formerly assistant superintendent of the Crucible Steel Department of the Bethlehem Steel Company, South Bethlehem, Pa., and for the last six years metallographer and metallurgist in charge of the research department of what is now the Taylor-Wharton Iron & Steel Company, having completed the installation of metallographic and testing equipment for the latter company, under the direction of Professor Henry F. Howe, vice-president and consulting engineer, and having perfected the operation of this department in investigating, improving, and controlling the quality of the output, has severed his active connection with them (retaining, however, a position as consulting engineer, and has opened an office at 165 Broadway, New York City, to engage in the general practice of consulting metallurgical engineering, making a specialty of steel-making processes, heat treatment and annealing, metallographic and testing equipment for the investigation, improvement and control of product, special steels, etc.

Mr. Charles W. Henderson, of Denver, recently completed an examination of the zinc resources and treatment methods of the Butte district in the interest of the United States Geological Survey.

Dr. Carl Hering, of Philadelphia, Pa., delivered a lecture on January 13 before the engineering students of the University of Virginia, on electrochemistry and electrometallurgy, in the course of which he sketched some of the latest advances in these branches and dwelt on the great possibilities of the future, mentioning many problems awaiting solution.

Mr. L. F. S. Holland, superintendent of the Smuggler-Union Company at Telluride, Colo., has returned to his work after an extended absence in Nova Scotia.

Mr. Milton M. Kohn, well known as the inventor of the replaceable-unit electric furnace, was one of the passengers on the steamship *Turrialba* of the United Fruit Company which was almost wrecked on Christmas Day. Mr. Kohn, who was returning from a trip to the Panama Canal, is none the worse for the experience.

Mr. C. B. Lakenan, general manager of the Nevada Consolidated Copper Company, was married recently to Mrs. Floribel White, of Los Angeles.

Prof. H. K. Richardson, of the electrochemical department of the Pennsylvania State College, has been chosen by the International Committee of the Y. M. C. A. to take charge of their scientific work in Chengtu, Province of Szechuan, China. The work is both scientific and religious. The scientific work is divided into two parts, one the training of science teachers for the new demand of the educated class of China, the other the taking care of a science museum where working models of inventions that have contributed to the progress of our civilization can be demonstrated.

Mr. Henry E. Wood, of Denver, visited New York City and other Eastern cities in January.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

653,538, July 10, 1900, Nathaniel Lewis Turner, of Salmon City, Idaho.

Relates to an apparatus for electrolytically depositing metals from solutions, such as gold-potassium cyanid, from the treatment of slimes, etc., and consists of a tank in which are suspended several concentric anodes, with means to detachably suspend separate cathodes between the anodes. Several rotating stirrers are provided which serve to keep the solution circulating. The anodes may be of iron or copper, and are covered with canvas; the cathodes are of lead or copper, and may also be covered with canvas. Provision is made for the examination and replacement of cathodes during the electrodeposition.

654,437, July 24, 1900, William Arthur Caldecott, of Johannesburg, South African Republic.

Relates to a method of precipitating gold from their cyanid solutions. Consists in passing the cyanid solution through precipitating boxes filled with mechanical mixtures of zinc shavings and lead shavings, as distinguished from zinc-lead couples, and shavings of zinc-lead alloys. The shavings are made by winding layers of zinc and lead, separated by paper, on a mandrel, forming a compact cylinder, and making the shavings as usual. The zinc and lead together form an electric couple, the paper forming a separator between them. It is stated that the mechanical mixture of shavings retains its efficiency longer than other forms of shavings.

656,305, August 21, 1900, Wilhelm Strzoda, of Falenze, Germany.

Relates to the solution of zinc compounds and ores in alkaline solutions, and the deposition of the zinc from the solution. Within a metal-lined tank are suspended anodes and cathodes, the metal lining also being connected as a cathode. Zinc ore or waste products are placed on the bottom, and an alkaline solution added to the tank. The solution may contain caustic soda, potash, ammonia, or any ammoniacal salt, either alone or with the addition of a salt of sodium, or of potassium, or the like. The continual evolution of hydrogen from the bottom of the tank keeps the ore, etc., stirred up in the solution. With sodium hydrate the soluble zinc compounds form sodium zincate, which is decomposed by the current depositing metallic zinc and reforming sodium hydrate. Other metals present, such as gold, silver, lead, tin, and cadmium, will be reduced to the metallic state, and if their compounds are also soluble, the metal will be deposited on the cathode with the zinc. The deposited metals may be separated in any usual manner. The solution can be separated from the residue by a filter-press and re-used.

657,032, August 28, 1900, Albion M. Rouse, of Denver, Colorado, assignor of one-half to William G. Shedd and Frank Brooks, of same place.

Relates to apparatus for dissolving the metallic values in ores, and for electroplating the dissolved metal. It consists of a tank in which ore is placed, and containing a central tube communicating with the outer space through perforations at the bottom. Within the tube, and just above the perforations is a rotary stirrer, or propeller which serves to pump the mixed solution and finely-divided ore pulp up through the central tube and down through the outer tank, causing continuous agitation and circulation of the pulp. By means of an anode and cathode located on opposite sides of the outer tank the dissolved metal is separated from the solution.

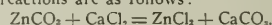
662,861, November 20, 1900, Emanuel Motz, of Jefferson, South Carolina.

Relates to an apparatus for electroamalgamating and electro-

dissolving and depositing metal from ores and solutions. It consists of a plurality of sections of a flume, connected so as to constitute a continuous channel for a flow of pulp and solution. On the bottom of each section, transversely to the flume, is a cathode plate of copper amalgamated, and parallel thereto and above it is an anode of lead suitably supported. Between two adjacent anodes is a vertically adjustable amalgamated metallic cathode plate the width of the flume, and which serves to agitate the pulp in the solution as the latter passes beneath the plate. The several sections are arranged in cascade so as to afford a continuous flow to the pulp. During the operation, any free gold is caught on the amalgamated cathode plates; the finely-divided gold will be dissolved by the solution with the aid of electrolysis and the dissolved gold redeposited on the amalgamated cathodes.

664,269, December 18, 1900, Carl Hoepfner, of Frankfort-on-the-Main, Germany.

Relates to a process of recovering zinc from ores and other materials. The zinc should be as carbonate or oxid, and is digested while under pressure and above normal temperature, with a concentrated solution of calcium chlorid and with carbon dioxide; or a mixture of this gas and air with zinc carbonate. The reactions are as follows:



The solution of zinc chlorid may then be enriched, if desired, and then electrolyzed. Any zinc chlorid remaining in the electrolyte is precipitated with hydrated lime. In the extraction of zinc from calamine (native carbonate) or roasted blende in contact with the carbon dioxide, or artificial zinc carbonate, solutions of calcium chlorid, solutions obtained in the ammonia-soda process, chlorate of potash, etc., may be used, or solutions of carnallite to which hydrated lime is added, may be used; the potassium chlorid in the carnallite may be used in the manufacture of chlorate of potash. The process is preferably carried out in a steam-jacketed digester containing a rotating stirrer.

664,537, Dec. 25, 1900, James Douglas, of New York, N. Y.

Relates to the extraction of copper from copper ore and matte by the chlorin or chlorin compounds resulting from the electrolysis of cuprous chlorid, and refers to his earlier patent No. 563,144. Broken pieces of ore or matte are placed in a suitable column, such as earthenware tubes, and the chlorin or gaseous chlorin compounds from the electrolysis of cuprous chlorid is passed through the ore or matte. The copper in the ore combines with the chlorin or chlorin compounds to form cupric and cuprous chlorids. If the acid electrolyte resulting from the electrolysis of the cuprous chlorid is used to leach the ore, the solution obtained is run back into the electrolytic vat.

668,842, Feb. 26, 1901, Albion M. Rouse, of Denver, Colorado, assignor of one-half to William G. Shedd and Frank Brooks, of same place.

Appears to be an improvement on his earlier patent No. 657,032, and relates to an apparatus for recovering gold and silver from ores. It consists of a plurality of tubs, having conical bottoms, with a valve and conduit connections to a vertical tube in which is a rotary stirrer to circulate the ore upwards through the tube and into the tubs through distributors. The ore is reduced to a fine pulp and mixed with about an equal weight of water, to which is added the solvent chemical, such as potassium cyanid, and supplied to the tubs, its temperature being maintained at from 90 deg. F. to 120 deg. F. by a steam pipe running through the tub. The pulp flows through a valve at the bottom of the tub, into a conduit leading to the upright tube, thence to the distributor consisting of a channel along the interior wall at the upper edge of the tub; the underpart of the channel being perforated, and provided with inclined chutes which give the pulp a swirling motion as it runs into the tub. This swirling motion of the ore causes the particles to rub against and clean each other, as well as thoroughly mix the ore and solvent. Anodes and cathodes are suspended in the circulating pulp, the dissolved metals being electrodeposited; the deposit of one operation serving as the cathode in a succeeding operation.

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More Aspects of Conservation

We have often puzzled over the meaning of the stock apology of those publicists who advocate the restriction of water-power development on the ground that such powers "should be reserved for the people." Naturally enough, they do not as a rule trouble to explain exactly what the phrase means, if anything. Our thanks are therefore due to Secretary Stimson, who, at a hearing before the Foreign Affairs Committee in regard to the restriction of water-power development at Niagara Falls, made a dangerous descent to the definite. Advocating that the water should be developed to its fullest efficiency, he suggested at the same time that provision be made to give preference in the power permits to cities and to lighting and other public service corporations.

We could wish that it had been somebody with a lesser reputation for clear-headedness than Secretary Stimson who had made this suggestion. From an economic point of view, it verges on unsoundness, since it is doubtful whether in unrestricted competition for hydro-electric power the consumptive outlets favored by Mr. Stimson could, in the long run, hold out against outlets which, on account of their much superior load factor, form the natural market for such power. As is well known, the annual load factor (that is, the ratio of the average load to peak load during the year) of municipal lighting, and other similar public-service load, is usually in the neighborhood of 30 per cent. The much higher investment per h.p. capacity in an hydro-electric plant, as compared, for instance with a steam generating plant, results in a diminution in the difference of cost between hydro-electric power and steam-generated power as the load factor diminishes, and it is safe to say that in most localities the difference in cost on service of low load factor is not such as would be appreciable to the average consumer of such power.

What we wish particularly to call attention to, however, are the really remarkable aspects of Secretary Stimson's suggestion from the straight and sincere point of view of conservation. We are not going to touch, on this occasion, on the monumental waste of power at Niagara Falls enforced (for the present) by the treaty between the United States and Great Britain, but confine ourselves to the consideration of the 20,000 cubic feet per second permitted for American use by the treaty. This represents a definite amount of power. Even Secretary Stimson appears to appreciate this, and is sufficient of a conservationist to desire that it be developed "to its fullest efficiency," whatever that may be. It never seems, however, to have occurred to him that the principles of conservation demand, in the first place, that the power be utilized to its fullest efficiency.

Seventy per cent of all hydro-electric power which is devoted to municipal purposes is wasted; wasted, because not

utilized. And it is wasted irretrievably; it is not a resource which, like the natural fuels, if not utilized today is saved for tomorrow. It *might* have been utilized, and by its utilization have saved a corresponding amount of exhaustible fuel. This makes the crime a double one, one of omission and one of commission. The only possible excuse for the consumption of hydro-electric power in municipal and similar service is the lack of a market with better load factor. Such a market is furnished in an ideal form by the electrochemical industries, which can be so run as to have a load factor of from 95 to 100 per cent.

In view of the power famine in the Niagara district, Secretary Stimson will hardly offer this excuse, however, for his astonishing proposal. Ignorance of the elementary engineering proposition which we have pointed out above is almost inconceivable in one who has at his disposal the advice of the Engineering Corps, unless, perhaps, the natural activities of the Department of War, dealing as they do with destruction and, in a minor degree, with construction, are unfavorable to considerations of mere conservation. We are reluctantly forced to the conclusion that, in making the suggestion, Secretary Stimson was influenced, probably against his better judgment, by the unreasoning clamor of ill-informed persons whose prejudiced vociferations have done so much in recent years to retard, against every principle of conservation, the development of this great branch of the country's resources.

The conservation of our exhaustible supplies of natural fuels demands the development of the water powers of America, and every attention should be paid to their efficient utilization, not merely in relation to efficiency of power production, but particularly in relation to the even more important matter of efficient consumption in respect of load factor. It is becoming increasingly evident that the political and professional advocates of conservation cannot be trusted, either because of their ignorance or because of their prejudice, with questions of water-power policy; and the engineering professions and technical press will fail in their duty if they do not supply this deficiency by furnishing persistent instruction and criticism.

Pumping Slime with the Air-Lift

The growing popularity of the air-lift for transferring slime in connection with the cyanide process, calls to mind the fact that data and information pertaining to that operation are comparatively meagre. As a water pump the air-lift has been the subject of several investigations, with the result that definite conclusions have been announced as to its advantages and disadvantages, efficiency and essential features of design. The adaptation of the air-lift to slime pumping has been based largely on the data derived from water pumping, and in this respect it is probable that no serious error has been committed. Nevertheless, the idea suggests itself that a mechanical investigation of the air-lift as a slime pump would be valuable to the metallurgical engineer.

It is true that such data of operation as air and power consumption can be obtained from individual sources, but usually they refer to the performance of particular pumps which may or may not be well designed or operating under

favorable conditions. In any event they are not comprehensive enough to form a basis for conclusions. An independent experimental investigation ought to develop such data as are not likely to be brought out in practice, and determine relations having a direct bearing on correct design and operation. There are five variables which may affect a particular size and type of air-lift: submergence, lift, discharge, volume and pressure of air. To test various combinations of these variables involves a vast amount of work, but some interesting and valuable data would result.

It is unlikely that the efficient percentage of submergence in pumping slime would vary greatly from that determined for pumping water, namely, 63 per cent; but there are other relations in which the denser medium would make a difference. The coefficient of pipe friction and slip; the quantity of air for maximum discharge; the ratio of volume of air to volume of slime; maximum output; efficiency and input, and other points which could be determined experimentally under varying conditions of submergence and lift, would form a basis for intelligent design and operation of air-lifts for slime. No harm would result if several such investigations should be undertaken at the same time by different parties; on the contrary, the possibility of having several sets of data for comparison would be rather attractive.

Requirements for Successful Ore-Flotation

Signs of the times in metallurgy point toward a wider adoption of flotation in copper and zinc-lead concentration in the United States. This form of concentration, developed to its highest state of perfection in Australia, and widely used there in the treatment of zinc-lead ores, has been the subject of widespread experimentation during the past year. Two notable installations have been made in the western hemisphere; one at the Braden copper mines in South America, and the other at the Butte & Superior zinc mines in Montana. Experimental plants have been built and used at several places, and many ores have been tested. The success which has attended these first efforts evidently has influenced the management of the Miami and Inspiration copper companies in Arizona to delay completion of plans for remodeling the first-named mill and constructing the second, until tests can be made to determine the possible value of flotation in the treatment of their ores. The Federal company in Idaho also is reported to be experimenting with the process on zinc-lead ores.

The requirements for successful ore-flotation have been pretty well determined, and those who contemplate experimenting with the process should inform themselves on these points before proceeding further. Decomposed, oxidized ore, in general, seems to exert a deleterious effect, forming no "froth" and yielding poor results. Clean sulphide minerals are most favorable and give good results. Close sizing seems to be a prerequisite, for large particles of mineral in an unsized feed show a tendency to fall through the scum of floating concentrate, and in so doing they not only pass into the tailing, but carry finer mineral with them. This fact suggests a maximum size of particle, which should be determined by experiment.

The process further requires uniform conditions in the feed, and in the speed of the flotation machine. Irregular

quantity of feed disturbs the operation and causes loss of mineral. Fluctuation in the speed likewise has a bad effect and should be overcome by providing an independent source of power. The supply of oil should be constant. Not the least important factor is the consistency of the feed. Australian practice has developed a ratio of water to solids of 3 or 4 to 1. In some cases a thicker pulp has been used, but more dilute pulps are not advantageous.

The excellent paper by Mr. H. Hardy Smith, presented before the Australasian Institute of Mining Engineers and published in this issue, may be recommended to the careful attention of all interested engineers as one of the most valuable contributions to the literature on flotation.

Nichols-Hesse Dinner

Those who attended the International Congress of Applied Chemistry last year in New York City will certainly be delighted to hear of a movement to show the appreciation of their fellow chemists to the two men who bore the brunt of the battle—the President of the Congress, Dr. William H. Nichols, and the Secretary, Dr. B. C. Hesse. This mark of the appreciation of their work will take the form of a complimentary dinner, to be held at the Chemists' Club on April, 1912. It is particularly pleasant that this decision should have been reached by the joint unanimous action of the executive officials of the various chemical societies. Without doubt the rank and file of American chemists will join to make the dinner a brilliant success.

Can the Bessemer Steel Industry Be Saved?

It is an interesting fact that there has been no important improvement in the quality of Bessemer steel since the early years of the industry, now more than half a century old. Improvements in processes have been made, but they have conduced to lower costs rather than improved quality. Nothing could be much more beautiful, from an economic standpoint, than the method by which molten pig iron at a red heat is converted in a few minutes into molten steel at a white heat, yet all the trends of the times were apparently in recent years dooming this industry to eventual extinction. The United States saw its maximum production of Bessemer steel as far back as in 1906, whereas the iron industry as a whole has so grown meanwhile that pig iron is being produced at a rate 30 per cent greater than in 1906. In one product after another Bessemer steel has become unacceptable. Producers assert that in many cases the preference expressed by buyers for open-hearth steel is unreasonable, but they do not take steps to correct what they denominate a misconception.

The failure of Bessemer steel to equal open-hearth steel in quality is a failure under unfortunate circumstances, for the raw material of the Bessemer process is by far the purer. Two facts plead that someone should save the Bessemer process, one being that the United States possesses reserves of hundreds of millions of tons of relatively pure iron ore, within the Bessemer limit as to phosphorus, while the other is that in many cases, particularly the rolling of sheets, the drawing of wire and the threading of pipe, Bessemer steel yields itself more readily to the mechanical operation than does open hearth steel.

Two losses confront the industry. One is, perhaps, only a paper loss, covering the greater value which a few years ago Bessemer ores lying in the ground were held to possess over non-Bessemer ores, apart from the difference in average iron content, while the other lies in actual equipment which is being rendered less useful. The loss is not merely in Bessemer vessel capacity, and indeed that has been saved in some instances by the adoption of the duplex process, but covers also the complete rearrangement of works, if open-hearth is to be substituted for Bessemer, by reason of the much greater ground space required by open-hearth furnaces of equal tonnage output.

There is no doubt that Bessemer steel possesses certain valuable qualities as compared with open-hearth, even though they cannot all be measured quantitatively, and Bessemer steel, merely shorn of certain objectionable qualities, would undoubtedly prove superior to open-hearth for many uses involving an enormous tonnage.

We have expressed the opinion before that the savior of the Bessemer converter industry will be the electric refining furnace, and nothing has happened that would induce us to change this opinion. The refining of molten Bessemer metal in the electric furnace will undoubtedly solve all technical problems, in as much as it will yield a metal that should satisfy all requirements. The reason why the industrial development of the addition of electric refining furnaces to existing Bessemer plants has been so very slow is a commercial one and a perfectly natural one. It concerns both the steel consumer and producer and has to do in both cases with the question of cost.

In the case of the consumer, the railroad companies naturally hesitate to pay a higher price for rails made from electrically refined steel, until the evidence of the superiority of the "electric steel rail" is complete. Now this complete evidence can only come from the experience with electric steel rails in the track for a series of years. That is a safe and sane, but naturally very slow development. Last May Mr. William R. Walker stated officially for the U. S. Steel Corporation before the American Iron and Steel Institute that at that time approximately 5600 tons of standard electric steel rails were in service in the United States and had then been in the track about two years, being exposed at times to severest weather conditions and to all the possible conditions of severe service, and that up to that time they had not heard of any basic electric rails in use in this country being broken in service. It is a pleasure now to say here that what was true then holds good now.

As to the producer, the attempts are being continued to reduce the additional cost of electric refining as much as possible. And this again is essentially a slow process, as it involves a lot of experimental work both as to the best division of the metallurgical reactions between converter and electric furnace and as to administrative measures and arrangement of plant. The goal is to get the cost of combined converter and electric furnace process down to the cost of the open-hearth. There is every reason to expect that this goal will be reached. When this is accomplished, it is evident that the Bessemer converter industry has been saved.

Readers' Views and Comments

The Tungsten-Molybdenum Couple

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have read with great interest Dr. Northrup's article on the tungsten-molybdenum thermocouple in your January issue, page 45, and also the discussions of Drs. Hering and Thwing on pages 66 and 67 of the February issue. It may interest these gentlemen to hear that the tungsten-molybdenum couple was announced three months before the meeting of the Philadelphia branch of the A. I. E. E. referred to by Dr. Hering.

My Congress paper on the "Applications of Ductile Tungsten" was reported in abstract in the September 12, 1912, number of this journal, page 580. It is interesting to note that our value for the maximum point is 540 deg., compared to 530 deg. reported by Dr. Northrup.

COLIN G. FINK,

*Harrison Laboratories, General Electric Company,
Harrison, N. J.*

* * *

Conservation and the Wastage at Niagara

To the Editor of Metallurgical and Chemical Engineering:

Sir:—With reference to recent articles in your journal on conservation and the wastage at Niagara, I send you the following verses, which were submitted to the *Outlook* five years ago.

It was my idea that the *Outlook's* views on conservation and on industrial democracy would naturally make the lines acceptable.

They were promptly declined.

Niagara Captive

Niagara captive! And by ribbons led!
His mighty force with that of toiling head
And hand to join. So changed since ancient days
When red men chanted hymns of praise;
In flower-laden white canoe
Each spring their fairest maiden sent into
The Thunder of the Waters.

Niagara an adult and to Effort bred—
No more to play the livelong day,
But proudly share the sweat and grime
Of stalwart manhood's laboring prime.
The evergrowing purpose runs:—
Earth's wealth is measured, not the sun's;
The stewards of great treasure may
Not waste Tomorrow's dire need
For Pleasure's or for Profit's greed.

Oh, Hercules, still at thy labors keep!
Canst take the raging current from the flood
And swiftly, silent 'round a cable sweep?
Ye Seven Wonders of the ancient world,
Long since into oblivion hurled,
Your kings and gods born to commemorate—
'Tis to the people do we dedicate
The Wonders of Today

Buffalo, N. Y.

* * *

EDWARD ZAREMBA

Early Mining Schools in America

To the Editor of Metallurgical and Chemical Engineering:

Sir On p. 114 of your February issue occurs the following misstatement: "George A. Koenig * * * at the University of Pennsylvania gave the first course of mining that was ever given in any educational institution in the United States."

Professor Koenig joined the faculty of the University of

Pennsylvania in 1872, and by 1871 six educational institutions had already graduated classes in mining. The first classes in mining graduated at Columbia in 1867, and at the University of Michigan the same year, Massachusetts Institute of Technology in 1868, Washington and Lee in 1869, and Lehigh and Lafayette each in 1871. Among these early graduates were some of the most distinguished men in the mining profession.

A course in mining was also started at Rensselaer Polytechnic Institute in 1867, but it was abandoned in 1871 for lack of funds.

Who first taught a course in mining in the United States would probably be difficult to determine, for undoubtedly rudimentary courses in the art were given long before schools of mining engineering were organized, but certainly it long antedated Professor Koenig's day.

EX-PROFESSOR.

San Francisco, Calif.

* * *

Corrosion and Immersion Tests on Metals

To the Editor of Metallurgical and Chemical Engineering:

Sir:—It has been customary for years and there seems to be no inclination to depart from the custom of giving the results on corrosion and immersion tests in terms of percentage loss by weight.

This is obviously not good practice as two results are not at all comparable unless the ratio of the weight to the surface exposed is the same in both cases.

If a 1-inch solid cube of steel is immersed in a 10 per cent. solution of sulphuric acid for 10 hours and a 1-inch hollow cube of the same steel is immersed for the same length of time the actual loss in weight should be presumably the same and on calculating the percentage loss by weight as is customary, the results would be entirely different and erroneous.

The foregoing is given as an illustration and is perhaps exaggerated but illustrates the point.

The writer has adopted the plan of giving the loss in grams per square inch of surface exposed.

If it is desirable to adhere to the metric system grams per square centimeter can be given.

There is, of course, the factor of time and to compare two results the time must be equal. Unfortunately the results of several hours cannot be reduced to a unit, say the hour, as corrosion or dissolution does not proceed at a uniform rate. It would be well for investigations to adopt some standard as to time and I would suggest one day (24 hours), 7 days and 28 days as being reasonable.

Other factors of importance are:

(a) The metal should always be tested with the same acid or substance for which the metal is intended.

(b) The ratio of the acid to the test piece should be the same in comparative tests as where this is different one becomes neutralized more quickly than the other. A rule could be established whereby a definite number of cubic centimeters of the acid could be taken for each square inch of surface exposed on the test piece.

(c) A separate receptacle should be had for each test piece. Should two test pieces of different metals be placed in the same bath electrolytic action may set up and cause dissolution to proceed more rapidly than otherwise.

(d) The temperature of the bath should be equal on comparative tests.

(e) In reporting results it should be stated whether the test piece was cast and machined, cast without machining, rolled or forged, etc.

E. I. du Pont de Nemours Powder Company,

Henry Clay, Del.

CHARLES E. ARNOLD.

The Solubility of Alumina in a Bath of Fused Fluorides

To the Editor of Metallurgical and Chemical Engineering:

Sir:—As the following data were unknown to several men most prominently connected with the development of the aluminium industry when told them after the symposium on alumina held at the recent meeting of the New York section of the American Electrochemical Society and were found only recently by me in some casual reading, their reproduction may be of interest.

Déville¹, in a Memoir on Silicium, after giving the results of other methods of isolating that element says:

"Finally silicium may also be extracted from silica by means of the battery—an exceedingly simple process which may also be utilized for the electrolysis of all substances which dissolve under heat in the fluorides of the alkalis—and these substances are very numerous. A mixture is made of sodium and potassium fluorides in about equal proportions which are melted together above a double current burner fed by alcohol to which turpentine is added and aided by a blow-pipe. When the material is fully fused, calcined silica is fed in which quickly dissolves. Then, putting the platinum and carbon poles of a Bunsen battery of four cells in the crucible, silicium is seen to deposit on the negative pole and oxygen is evolved at the positive pole. I have most frequently been satisfied to verify the decomposition of silica by the battery by this experiment, in using, as negative electrode, a platinum wire which was transformed with great ease into a fusible silicide.

"I desired to generalize this method in applying it to very different materials which are soluble in the fluorides, first of all to isolate the elements by the battery, and then to determine their position in the electrochemical series. Thus one can deduce from the experiment which I have just described that silica separates into its elements before soda and potash, and in consequence that silica should be decomposed by the alkali metals, which is indeed the fact. But if for silica alumina is substituted, the decomposition by the battery, in the midst of the alkali fluorides, would be effected in quite different order. One obtains as a matter of fact, at the negative pole, sodium which burns with a yellow flame, and at the positive pole, fluorine, which is immediately converted into hydrofluoric acid readily recognized by its odor. Alumina, therefore behaves quite differently from silica, and one must admit that it is not decomposed by the alkali metals, which all the attempts that have been made to obtain this reduction directly seem to have already shown."

In the light of subsequent events it is curious to note how close Déville, to whom we owe the first industrial isolation of aluminium, was, in 1857 to the winning of aluminium by means of electrolysis.

I had been successful there was no cheap current available. It is also interesting to note that this was not an haphazard experiment but one undertaken with a view to generalizing. It is true that soda and potash should be decomposed more readily by the current than alumina, but the alumina should have been decomposed more readily than the fluorides, and no doubt would have been if the proportions had been propitious.

New York City.

CHARLES A. DOREMUS.

* * *

Nitrogen, Its Influence in Steel

To the Editor of Metallurgical and Chemical Engineering:

Sir:—As several articles have recently appeared in the various technical journals relative to the momentous influence that small amounts of nitrogen may have in steel, and as various papers have been read on this subject before the engineering and chemical societies, the writer is led to relate the following from his experience and investigations dealing with the influence of this element in open-hearth and electric-furnace steels met with in every day practice, and of the grades used for small shafts, spindles, pneumatic tools, forming dies, etc.

¹Memoir on Silicium, *Annales de Chimie et de Physique*, 3, XLIX, p. 69, 1857.

In the main it may be conceded that nitrogen produces brittleness in all grades of steel though whether this element is itself the cause or whether its influence on other injurious elements, such as sulphur and phosphorus, in conjunction with it is the cause of brittleness, has not been satisfactorily determined.

It will require numerous tests, in view of the fact that practically similar conditions give inconsistent results, in which careful and complete analysis of a great many steels are compared with their physical properties to arrive at definite conclusions. It must be admitted, in general, that the results already obtained show practically no difference in the merits or demerits of steels that have been examined unless the phosphorus and nitrogen contents were abnormally high.

From the fact that metallurgists in common are at variance as to the proper methods for measuring the quality, or merits, of steel and from the uniformity of the merit formulae and figures, it is almost impossible to adopt a rule covering all conditions of comparison.

Mr. C. E. Stromeyer, in a paper read before the Institute of Naval Architects, London, England, gives the following rule for determining the quality, or merit, of steels from the phosphorus and nitrogen contents:

"The sum of the percentage of phosphorus plus five times the percentage of nitrogen should not exceed 0.080 per cent in good quality steel."

He further states, "that nitrogen has a ten fold greater effect than phosphorus in raising the tenacity of steel."

The formula most commonly used in England for determining the quality or merit of steels from their physical properties, elongation and tensile strength alone being considered as elastic limit and reduction of area are not factors, is as follows:

"M" = "T" + "E",

"M" = merit,

"T" = tensile strength in gross tons

"E" = elongation in per cent.

If tensile strength and elongation are directly or indirectly a measure of toughness, or the combination of ductility and toughness and ductility or toughness are the opposite qualities of brittleness, then a comparison of results determined from formulae combining tensile strength and elongation, and formulae showing the effects of nitrogen, should give conclusive evidence relative to the influence of the element nitrogen.

The following table gives the complete analyses, including nitrogen, of a number of steels that were investigated by the writer to determine the influence of the nitrogen present:

TABLE No. 1. COMPLETE ANALYSIS OF SAMPLES

Test Num- ber	Carb- on	Sili- con	Man- gan- ese	Sul- phur	Phos- phorus	Titan- ium	Chro- mium	Van- adium	Nitro- gen	Dupli- cate Anal- yses
1	.84	17	29	.009	.011				.12	0.10
2	.18	20	.40	.022	.013	.080			.08	.08
3	1.01	10	.32	.009	.014	Trace			.13	.12
4	1.28	18	.37	.009	.013				.08	.060
5	.64	12	.25	.022	.018	.010			.10	.08
6	.48	20	.40	.012	.08			.20	.08	.14
7										

Note: Determined by

Summary.

No. 1—Average of nine Electric-furnace heats.

No. 2—One Electric-furnace heat to which .50 per cent Ferr. Testum was added.

No. 3—One Electric-furnace heat to which .30 per cent Rutile (1.5% of Titanium) was added.

No. 4—One Electric-furnace heat to which .60 per cent Rutile was added.

No. 5—Average of nine open-hearth furnace heats.

No. 6—One heat of Chrome-Yanawara steel.

No. 7—Drillings from Bessemer steel (20% of Nitrogen determined).

The method used for determining the nitrogen in these steels is as follows:

An Erlenmeyer flask of about 750 c.c. capacity is provided with a rubber stopper which carries a funnel and tube connected to a Bunsen condenser. The funnel is provided with a suitable stop cock.

About 500 c.c. of water, to which has been added .30 c.c. of

a solution of caustic soda equal in strength to hydrochloric acid of specific gravity 1.12 are put in the flask and boiled. During this preliminary boiling one gramme of the sample steel is dissolved in 20 c.c. of hydrochloric acid and this solution is then put in the funnel of the Erlenmeyer flask from which it is allowed to fall, drop by drop, by means of the stop cock in the boiling alkali solution, and distillation is continued until all the ammonia is driven over. The distillate is collected in graduated glass cylinders after which it is treated with Nessler's reagent and the coloration produced matched with standard solution of ammonium chloride.

The standard solution is made up of a strength equal to 0.03815 gram ammonium chloride per liter so that one c.c. is equivalent to 0.01 milligram of nitrogen.

It is, of course absolutely necessary in determinations of this kind to take extreme care that the water, hydrochloric acid, etc., used are perfectly free from nitrogen compounds.

The table below shows the physical properties of these steels together with a column of merits obtained from the English formulas given above.

TABLE NO. 2—PHYSICAL PROPERTIES OF SAMPLES

Test Number	Elastic Limit, Pounds	Tensile Strength, Pounds	Elongation, Per Cent	Reduction of Area, Per Cent	Merit
1	68,000	105,500	18.5	46	65.5
2	58,400	98,400	19.5	52	63.5
3	69,500	105,000	21.5	42	68.5
4	70,400	108,900	17.5	41	66.5
5	53,700	95,200	17	34.5	60
6	82,000	123,500	19	45.5	74
7			Not determined		

The steels containing titanium are especially given as the addition of this element is considered to retard the segregation of sulphur, phosphorus and carbon present; also, by its action in reducing a part of the nitrogen, it decreases brittleness and augments toughness and ductility. Its influence is said to be similar to manganese in reducing oxygen and in addition combines directly with nitrogen.

A lengthy paper on this subject by P. H. Dudley will be found in the *Journal of Ind. and Eng. Chemistry*, volume 3, pages 299 to 304.

In the heats referred to, the titanium was added as the steel was being poured into the ladle. In sample No. 2, 0.50 per cent, by weight, of ferro-titanium was added; this heat shows 0.080 per cent of titanium in the steel. Sample No. 3 about 0.30 per cent of titanium, by weight, was added in the form of rutile (red oxide of titanium ore); it will be observed in this case that the steel only shows a trace of titanium. It may be possible in this and, also, in sample No. 4, that the rutile added, being in fine powder and of light weight, floated on the surface of the molten steel and combined directly with the slag. In sample No. 4 about 0.60 per cent titanium was added in the form of rutile.

If we now apply Mr. Stromeyer's rule for quality judged from the phosphorus and nitrogen contents and assume the factor of P plus 5 (nitrogen) should not exceed 0.080 per cent, we get figures as follows:

Sample.	P + 5 (Nit.).	Merit.
1	.060	18.75
2	.053	33.75
3	.079	1.25
4	.068	15.00
5	.033	58.75
6	.038	52.50
7	Not determined.	

The merits obtained from the physical properties indicate sample No. 3 as being the next to the best steel, while from the P plus 5 (Nit.) formulae it is placed as last in rank, in fact it is very close to the danger point. This is only another case of the inconsistency or disagreement of merit figures. Sample No. 4 is third highest from its physical properties and is next to the last merit from its phosphorus and nitrogen contents.

Sample No. 6, which is an extremely good steel as indicated by its physical properties, also excels in merit from its nitrogen and phosphorus contents; it will be observed that it is very low in both these elements.

The average of the open-hearth heats, sample No. 5, are extremely low in nitrogen; the physical properties are, also, good for this class of steel. These steels were made in a basic furnace and are far superior to the rail steel given as sample No. 7 if the elimination of nitrogen is a criterion. The functions leading to superiority of basic open-hearth steels are the possibilities for the elimination of sulphur and phosphorus and the reduction of the oxides. In this case extreme care in operation was taken; the quality of the steel produced shows the effects of this careful handling.

The writer trusts the merits and demerits of nitrogen in steel will be discussed further or until a definite and clear understanding of its effects on the quality of all grades of steel shall be firmly established.

E. D. A.

The Western Metallurgical Field

Proposed Merger at Cripple Creek

Considerable interest attaches to the recent examination of the Golden Cycle mine in the Cripple Creek district by Mr. Richard A. Parker on behalf of the El Paso company. It is currently reported that Mr. Allen L. Burris, president of the El Paso, has been making an effort to consolidate some Cripple Creek mines, together with the district railroads, and that Golden Cycle mines and cyanide mill are part of the merger. A \$25,000,000 holding corporation is part of the announced plan, and the properties to be acquired are the following: Golden Cycle mines and cyanide mill, El Paso mines, Colorado Springs & Cripple Creek District Railway, Midland Terminal Railway, Florence & Cripple Creek Railway, Gold Belt Railway, and the High and Low electric lines. The holding company will engage in general mining business, confining its activities to the Cripple Creek district for the present.

Copper Smelting in Michigan

Smelting practice in the Michigan copper district does not vary materially at the different smelters. The process is practically one of melting and refining, as the concentrates are mostly metal. At the Quincy smelter the concentrates are separated into three grades: No. 0, mass, containing from 65 to 70 per cent copper; No. 1, coarse concentrate and some mortar discharge, averaging about 75 per cent copper; No. 2, fine concentrate, containing from 10 to 25 per cent copper. The poorest grade, No. 2, is agglomerated in a reverberatory furnace and later smelted in a blast furnace. This practice has superseded the former method of briquetting the fine copper for blast furnace reduction. No. 0 and 1 grades are melted in reverberatory furnaces having a capacity of about 50,000 lb, refined copper. The operation is intermittent, covering the following operations: The furnace is charged at 4 p. m., and the charge is skimmed every two hours after slag begins to form. This melting and skimming is continued for 15 hours, after which the charge is refined with air, oxidizing arsenic and other impurities, and forming some copper oxide. The molten bath at this stage contains about 95 per cent metal and 5 per cent oxide. The latter is then reduced by poling and with charcoal. Casting is commenced about 1 p. m. and finished about 3 p. m. The furnace is repaired if necessary and made ready for the next charge at 4 p. m.

Casting at the Quincy smelter is done by hand from ladles that are carried on an overhead track that runs above the line of molds. At the Michigan smelter a Walker casting wheel is in use, and at the Calumet & Hecla similar wheels are used in connection with two new reverberatory furnaces. The latter have capacities of 125,000 lb. of refined copper per day each, and are equipped with Stirling boilers for generating steam from waste heat, after the manner of similar equipment on reverberatory furnaces in the western copper smelters. There are some objections to the use of casting wheels; the cast

copper cools quickly, but if the wheel is moved before the surface is set, the ingots have a rough, wavy surface which is objectionable in subsequent rolling for industrial uses.

Reverberatory slags are high in copper, but if lower-grade slags were made it would be at the expense of the grade of metal produced; and as the slags would have to be resmelted in any event, there is no object in trying to get them as low as possible. Such slags will carry from 10 to 12 per cent copper, and are richer in metal toward the end of the skimming process. They are smelted in a low-pressure blast furnace, the product of which is copper of 95 per cent purity, which is later refined in a reverberatory. Slag runs continuously from the blast furnace, carrying about 0.8 per cent copper, which is about twice the grade of slags from western matte furnaces. Lower-grade slags involve reduction of iron which contaminates the copper. The principal flux used in the blast furnace is limestone, although iron in the form of pyrites cinder is used on occasion. Anthracite coal is preferred to coke, the latter being used when the furnace shows signs of running cold, or when accretions build up in the shaft.

Company Reports

The *Shannon Copper Company* has just issued a report covering the operations during the last four months of 1912. The company treated 95,235 tons of its own ore, producing 5,149,805 lb. copper, 1012 oz. gold and 54,394 oz. silver. The cost per lb. copper during the four months was as follows:

September	13.858 cents.
October	14.837 "
November	12.714 "
December	12.548 "

The high costs during this period were due to charging off certain increases in taxes and liability insurance which the legislature enacted last summer, and which could not be distributed over the costs for the year. Another cause contributing to high cost was the chilling of the large furnace during October, reducing production for several days without reducing expenses. The net profits for the period were as follows:

	Mines.	Railroad.	Total.
September	\$41,000	\$1,415	\$42,415
October	29,000	1,039	30,039
November	66,200	1,850	68,050
December	65,695	1,773	67,468

The report states that the mines are in better condition than for years, and the lower levels hold out better promise than ever before. Dividends amounting to \$300,000 were paid during the last eight months of the year.

The *Nevada Consolidated Copper Company's* report for the last quarter of 1912 shows a marked decrease in production compared with the preceding quarters, owing to labor difficulties which caused a complete suspension of operations from October 2 to October 24. The grade of ore milled, 1.44 per cent copper, was lower than usual. The production of copper was only 8,986,905 lb., less than half the usual production.

The reports of the *Utah Copper Company*, covering the fourth quarter of 1912, shows a monthly production of 4,302,194 lb. copper as compared with an average of twice that figure for the preceding months of the year. The decrease is ascribed to labor troubles, referred to in the last quarterly report, and to the shortage of labor caused by the emigration of foreign laborers for service in the Balkan War. The average assay of ore treated during the quarter was 1.104 per cent copper, as compared with 1.41 per cent for the preceding quarter. The average cost of copper, after making allowances for smelter deductions was 14.83 cents per pound, compared with 7.57 cents for the preceding quarter. The high cost was due to deficiency in tonnage, low grade of ore and extraordinary expenses. On the date of the report, Feb. 1, the mills were treating 18,000 tons per day, and it was expected that within thirty days the full capacity of 20,000 tons daily would again be reached.

The nineteenth annual report of the *Portland Gold Mining Company*, Cripple Creek district, Colorado, was issued on Feb-

ruary 3, and shows the company to be in excellent condition, with greatly improved metallurgical facilities. The Colorado Springs mill has been reconstructed into an up-to-date cyanide mill, and has been operating at its full capacity of 400 tons per day since November 15, 1912. The new mill at Victor has continued to demonstrate its success in treating low grade ore which formerly was thrown on the dump as waste. Its capacity has been increased from 12,285 tons treated in December, 1911, to 14,791 tons in December, 1912. Additions have been made to increase the concentrating capacity, consisting of twelve concentrating tables. Further additions to plant consisted of four Pachuca tanks, one Dorr thickener and two classifiers. These changes have raised the original capacity of the mill from 300 tons per day to 500 tons. It is expected that the net profits from this mill will be equal to the dividend requirements of the company, leaving the profits from the old mill to be applied toward building up a substantial reserve fund. The average value per ton of ore treated in the new mill was \$3.15, yielding a net profit of \$1.17 per ton. The profits earned by this mill in 1911 and 1912 have equaled 90 per cent of its cost, and only a beginning has been made toward the treatment of the vast tonnage of low-grade ore available. The Cripple Creek drainage tunnel has lowered the water in the Portland mines 207 ft., and mining operations are now conducted on the sixteenth level, where the ore bodies are larger than above. All pending litigation has been compromised. The total net profits for the year were \$325,507.84; dividends for the year, \$240,000, and for the entire history of the company, \$9,157,080.

Rochester, Nevada

The newest mining camp in Nevada which is attracting more than usual attention, is Rochester, in the Humboldt mountains, Humboldt county, being east of Oriana and Lovelock, which are on the Southern Pacific railroad. An incipient boom is now under way, and the prospect is that by late spring or early summer a new mining district will be well organized. New discoveries of high-grade silver ore have revived interest in a district which was active in the early sixties, when rich refractory ore was shipped from there to Swansea, Wales, for reduction. The ore now being mined resembles that of Tonopah, being high grade in silver and low in gold. At present two mines are shipping about 20 tons of ore daily to the Mason Valley smelter at Thompson, Nevada. A great deal of development work is being done, and some sales of property are being made. Approximately 2000 people are in camp, and more are coming in daily. The winter weather is reported to be favorable to prospecting. Owners of prospects for sale are much more conservative in their demands than they have been in other Nevada camps, and indications are that outside capital will be encouraged to come in. An element of permanency is found in the construction of a steam-electric power plant at Oriana, 10 miles distant, to transmit power to the mines. At present all mining is done by hand-drilling, and as the rock is extremely hard, little progress can be made.

Organizations of Mining Men

Matters of state and national legislation affecting the mining industry are resulting in the organization of mining men in different parts of the country. The question of mine taxation is the foremost issue in the different states, while the tariff on lead and zinc is the national issue around which much discussion is being waged.

The subject of taxation is before the legislatures of Utah and Colorado, and new organizations have been effected in both states to see that no unwise measures are passed. The principles of mine taxation unquestionably are not yet settled to the satisfaction of mine owners, nor is there any uniformity in the laws in different mining states. This state of affairs has received the attention of the American Mining Congress which, at its last session in Spokane, passed a resolution appointing a committee on mine taxation to report at the next session of the congress. In the meantime Colorado proposes a law that shall assess mining property at its full cash value, directing assessors

to take into consideration any matters that will enable them to arrive at a fair and equitable cash valuation. Manifestly this means nothing, and points the way to the great need of a scientific method of arriving at the value of mining property. Such a method as Colorado proposes will result in confusion and dissatisfaction. It has been remarked before by competent engineers that foreign countries have solved this question more satisfactorily than we have; that distinction is made between idle and active owners; that the former are subject to more than nominal taxation, and that output is taken as a basis for the taxation of active companies. It is to be hoped that our own states will accomplish something constructive and sound along this line, for many instances might be cited in which the question of unscientific taxation determines the difference between idleness and activity of some companies.

The proposed reductions of the tariff on lead and zinc are being vigorously opposed by the American Mining Congress under the active direction of the secretary, James F. Callbreath, who is at Washington. He has been well supported by representatives from Missouri, but assistance from other states has not been as vigorous as is desired. If the question has vital interest to producers of these metals, there seems to be a strange apathy among many of them.

Iron and Steel Market

The pressure upon steel mills for delivery of material showed no abatement in February. Demand against existing contracts continued strong, with specifications substantially equal to shipments, and efforts in many cases on the part of buyers to have shipments expedited against specifications already filed.

There have been no important advances or declines in the regular base prices of important finished steel products, and these prices are well maintained. There has been a very considerable reduction in the premiums paid for early delivery of certain products, also a diminution in the tonnage of material sold at premiums, so that the premium market is now relatively unimportant. Probably this change can properly be ascribed to the fact that we have been passing through the winter period.

In pig iron, which in recent years has presented a market altogether distinct from that of steel products, the slight weakening which occurred in January was carried farther. Pig iron prices no longer control steel prices, only a very small portion of the total merchant pig iron output being sold by direct barter with steel producers. Steel prices are affected only sentimentally, and then but very slightly.

Connellsville coke, which began to weaken late in January, slumped in the fore part of February, so that prompt furnace coke, which ruled at \$4.00 a ton or higher from the beginning of November until late in January, sold at as low as \$2.25, firming up slightly after considerable accumulations were liquidated, and making a market of \$2.40 to \$2.50 in the closing week of February. There being no important negotiations, a definite market for contract coke was not developed, but early in January it was impossible to contract for standard brands at less than about \$3.50, while by the middle of February it became evident that if buyers and sellers got together it would hardly be on a basis outside the limits of \$2.25 and \$2.50. The slump in coke was due to several influences. The market had become unreasonably high, both in relation to cost of manufacture and in relation to selling prices of pig iron and steel products. There were fears in November and early December of an actual famine later on, production being usually curtailed around the holidays through too strenuous celebration by the workmen, and also being usually subject to curtailment through bad weather. Stocks were accumulated against these probable contingencies, but the winter proving exceptionally open production was not curtailed through natural conditions and in January furnaces began to use up stocks. At the same time more than half a dozen steel works furnaces began to work badly and took reduced coke shipments. Production increased slightly, and the combination of circumstances, increased production and reduced demand, although

amounting to but a very small percentage of the total movement, was sufficient to change the complexion of the market completely. There is no permanent decrease in consumption, whereas with the freer supply furnaces will refuse to take the poor coke they willingly accepted in December and January, forcing restriction of output at the plants making poor coke, and this readjustment will probably hold coke at a moderately high level, though much below the unreasonably high level which recently prevailed.

Reports of business and financial activity outside the steel industry proper have been uniformly unfavorable, and it has been a surprise that the steel industry has not been affected. There are many who think it has been, evidently believing that unfavorable developments in the steel industry have been concealed. There is no reason to believe that their attitude is correct. The steel industry is in strong position, with a very large volume of sound business on books, and strictly new buying naturally light in consequence. The divergence between conditions in steel and in other branches of activity is explainable on the ground that the country has continued to develop steel requirements in recent years, while steel making and finishing capacity has increased very little in the past three or four years, in comparison with the rapid increase in demand in all previous time. This fact can readily be illustrated quantitatively.

In the decade ended with 1897 pig iron production increased 94 per cent. over the previous decade, and the next decade showed a further increase of 120 per cent., the average of these increases being 107 per cent. An increase of 107 per cent. over the 181,470,000 tons produced in the ten years ended with 1907 would require an output of 376,000,000 tons in the ten years to end with 1917. The five years of that period already elapsed have produced but 122,412,000 tons, leaving 254,000,000 tons for the five years 1913 to 1917 inclusive, whereas it is straining the industry's capacity at present to maintain a rate of a trifle over 33,000,000 tons. There are a few idle furnaces, but it would be impossible to swell the production rate to 35,000,000 tons a year and avoid loss to poorly positioned operations, unless finished products should advance more than raw materials and the alignment is such that this could not occur.

There is reason to conclude, therefore, that the steel industry remains active and in strong position when general business is not particularly good simply because, from a variety of influences, development of producing capacity has not kept pace with development of consuming capacity.

Pig Iron

Since its high point towards the close of December pig iron has declined an average of almost 50 cents a ton, the declines being practically uniform in foundry pig iron in all markets, but being very slight in steel making iron in the Pittsburgh valley district and in eastern Pennsylvania. The declines have occurred very quietly, in a dull and stagnant market, and it is likely that the development of a moderate demand would develop still lower, rather than higher prices. The practice of buyers of foundry iron is well known: on a rising market they buy farther and farther ahead, while given a weak market they refuse to buy until delivery has been completed on all their purchases. The furnaces are fully sold up for perhaps two or three months to come, and deliveries are being very well taken, so that iron is not accumulating, but it is fairly safe to predict that prices will continue soft until April or perhaps a later month, when the real test of the market will come. In the circumstances, it would not require an exceptionally heavy demand to stiffen prices again. The market now stands quotable as follows: Bessemer, \$17.25; basic, \$16.35; No. 2 foundry and malleable, \$17; forge, \$16.50, f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh; No. 2X, delivered Philadelphia, \$18.25; No. 2, foundry, f.o.b. Buffalo furnaces, \$17; No. 2 foundry, f.o.b. Cleveland furnaces, \$16.50 to \$17; No. 2 foundry, f.o.b. Chicago furnaces, \$17.25; No. 2 foundry, Birmingham, \$13.50, with some resale iron offered at \$13.

Steel

Never, perhaps, have transactions in billets and sheet bars been so light. There has been practically no steel available for early shipment, and for such later shipment as the mills could compass they simply refuse to quote. As representing the general temper of the market we quote billets at \$29, sheet bars at \$30 and rods at \$30, f.o.b. maker's mill. Pittsburgh or Youngstown.

Finished Steel

Effective February 1, the National Tube Company advanced steel boiler tubes one point, or about \$2 a ton, other makers immediately concurring.

Regular prices at Pittsburgh, unless otherwise stated, are given below, subject in some cases to premiums for early delivery:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.50 cents.

Steel bars, 1.40 cents.

Steel hoops, 1.60 cents.

Common iron bars, 1.65 cents, Pittsburgh; 1.65 cents, Philadelphia; 1.60 cents, Chicago.

Sheets, blue annealed, 10 gage, 1.75 cents; black, 28 gage, 2.35 cents; galvanized, 28 gage, 3.50 cents; painted corrugated, 28 gage, 2.55 cents; galvanized corrugated, 3.55 cents.

Tin plates, \$3.60 for 100-pound cokes.

Merchant steel pipe, $\frac{3}{4}$ to 3-in., 80 per cent. off list.

Steel boiler tubes, $\frac{3}{4}$ to 4 $\frac{1}{2}$ -in., 70 per cent. off list.

Standard railroad spikes, 1.85 to 1.90 cents.

Button head structural rivets, 2.20 cents; cone head boiler rivets, 2.30 cents.

The price of 1.45 cents now quoted on plates and shapes is the price which the steel corporation has regularly maintained; for a time independent manufacturers endeavored to obtain 1.50 cents and as the steel corporation accepted only selected business at its quotation the market was quoted in general at 1.50 cents. As it is now recognized that the independents can book little attractive business at higher than 1.45 cents the general market is now quoted at that figure.

John Fritz

In John Fritz the engineering profession and the steel industry has lost a master mind, the United States one of its greatest citizens. A self-made man and a self-taught engineer he was a peer of the most eminent steel men of three generations and the engineering societies honored themselves when on the occasion of his 80th birthday they established the John Fritz medal to perpetuate the memory of his life work.

In view of the fact that he left for us to read his delightfully interesting autobiography there is no necessity here to go into details. Only a few dates and a few attainments shall be mentioned.

John Fritz was born Aug. 21, 1822, in Londonderry Township, Chester County, Pa. His early days were spent on a farm. At the age of fifteen he left school and became a blacksmith's apprentice. He became interested in foundry work, obtained employment with the Norristown Iron Works in 1845, and within a few months was superintendent of the plant. In 1849 he left this position and accepted one as mechanical engineer to put up the machinery for a rolling mill at Safe Harbor, Pa.

His health soon failed at Safe Harbor, and he was forced to leave. Then for several years he struggled with fever, working only a part of the time. His services were much sought, both at Safe Harbor and at Norristown.

In 1854 one of his former employers became interested in the Cambria Iron Works, at Johnstown, Pa., and induced Mr. Fritz to go there as general superintendent.

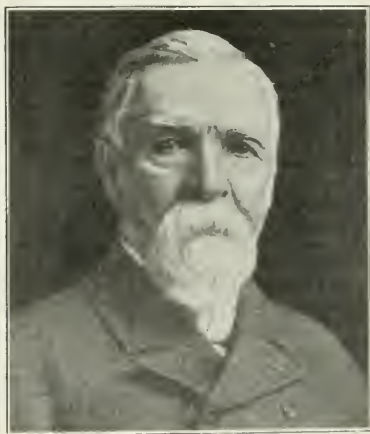
One of his innovations there was the replacing of the two-high rolls with three-high rolls, to effect greater economy and lessen the likelihood of accidents. This was made in 1857.

In 1860 he left the Cambria Iron Works and became general superintendent and chief engineer of the Bethlehem Iron Company, which was then about to build a plant at Bethlehem, Pa.

In 1864, the Bessemer process was introduced into the United States. Mr. Fritz had been studying the matter of steel-headed rails, and although the first steel rails in this country were failures, he induced his employers to build a Bessemer plant. Believing the failures were due to an unnecessary amount of phosphorus. The plant was started in 1868. About the same time the manufacture of acid open-hearth steel began, using the Siemens regenerating furnace. Three years later Mr. Fritz had a Siemens furnace installed at the Bethlehem plant, with important modifications of his own that proved successful.

The following year he added a blooming mill. He then turned his attention to the making of plates and structural shapes, but his company would not take up their manufacture. He later succeeded, however, in getting the company to install a forge and armor-plate plant, although there was at that time neither the demand for the thick steel plate nor proof that it could succeed. It did succeed in tests, and the demand was created.

Mr. Fritz retired from active work several years ago. Pre-



THE LATE JOHN FRITZ

vious to his death he had been in feeble health for more than a year. He died at his home in Bethlehem on February 13.

The funeral services were held at his late residence, which, though spacious, could not contain his many hundreds of friends, including eminent men of all callings and the delegations from the national societies. A special car brought engineers from New York, and there were present a large number of Lehigh University students. Thousands of the townspeople and steel plant employees waited patiently in the winter wind to follow "Uncle John Fritz" reverently a few hundred feet to his last resting place above the valley where the vast steel plant which is one of his enduring monuments extended for miles in unvoiced silence.

George W. Maynard

George W. Maynard, prominent as mining engineer and metallurgist and highly esteemed as a man by all who had the pleasure to know him, died in Boston on February 13, after an illness of several weeks which had started while he was on a professional mining trip to the West.

Mr. Maynard was born in Brooklyn, N. Y., on June 12, 1830. He graduated from Columbia as one of the famous class of 1860—the first class of the department of Chemistry at Columbia.

Last September (Vol. X, p. 517, 1912) it was the privilege of this journal to publish from Mr. Maynard's own pen rem-

iniscences of those earliest days of chemistry in New York City, together with the picture of the class of 1860 of the department of chemistry. Mr. Maynard wrote these reminiscences just before he started on his last trip to the West. They were his last contribution to literature and were characteristic of the man and his true big heart.

In 1860 he went to Germany and for a year studied chemistry, physics, and mineralogy in the University of Goettingen, under Woehler. While there he was a fellow student with J. Pierpont Morgan. Leaving Goettingen he spent a year at Clausthal in the Hartz Mountains with Bruno Kerl, studying mining and metallurgy in the School of Mines there. In 1862 he received the degree of Master of Arts from Columbia.

Mr. Maynard's professional practice started with an engagement at the Connoree mines in County Wicklow, Ireland, where he remained a year occupied in chemical and metallurgical work. In 1864 he returned to New York to establish an engineering office and chemical laboratory, and in the spring of the same year he made his first professional examination of a mining property in Colorado. He liked the West and in December, 1864, he returned to Colorado to establish an engineering office and assay laboratory in Gilpin County.

Mr. Maynard returned to New York in the winter of 1867, taking charge of a sulphuric-acid plant on Staten Island. In 1868 he was appointed Professor of mining and metallurgy at the Rensselaer Polytechnic Institute in Troy, N. Y., and occupied that chair until 1872. From 1873 to 1879 he was in Europe. His headquarters were in London, but during this time he spent six months in eastern Russia, erecting a copper reduction works for a British company at Voskresenky.

His stay in London became important for his future, as he was consulting engineer for several iron and steel works in England and Wales, and in such capacity made at Middlesbrough, England, the first demonstration in a large way of the basic Bessemer process of steel making. He was a very intimate friend of Sydney Gilchrist Thomas.

Mr. Maynard returned to New York in 1879 for the purpose of introducing the Thomas-Gilchrist process into the United States, and sold the patents to the Bessemer company, afterward known as the Steel Patents Company. Subsequent to that time Mr. Maynard always resided in this country, though he made various brief professional trips abroad.

Mr. Maynard continued his consulting practice until the time of his death. He died in the saddle. His many friends will remember him as a mining and metallurgical engineer of high attainments and as a man of lovable character.

Raphael H. Wolff

Raphael H. Wolff, widely known through his long connection with the iron and steel industries of both the United States and Germany and prominent through his great activity in introducing the electric steel furnace in this country, died on January 23 in a sanitarium in the Grunewald, near Berlin, Germany. Mr. Wolff had gone to Europe last June on one of his trips in which he used to combine rest and recreation with study of commercial and industrial conditions and none of the many friends who saw him off on that fine Saturday morning on the dock in Hoboken, could have imagined that it was a farewell forever.

Mr. Wolff came of an old family of wire manufacturers, with a mill at Neheim on the Ruhr in the Westphalian district. He came to the United States in 1870 and first found employment with the Pratt and Whitney Co., of Hartford.

Mr. Wolff was then for many years an importer of iron and steel in New York City, chiefly of wire rods and steel billets. Later he established a wire mill at 17th Street and the Harlem River, New York City, and conducted it as the head of the firm of R. H. Wolff & Company, Ltd. There he manufactured for many years cold rolled strip steel and high-grade wires, including piano wire and watch-spring steel ribbon. During the period of the bicycle craze he also took up the building of wheels.

Mr. Wolff sold his plant in 1902 to the Washburn Wire Company. He then became connected with the Crucible Steel Company of America. He became their European representative and established their European department with headquarters at Hamburg, Germany, and was soon successful in building up an important business.

Some ten years ago Mr. Wolff investigated for an American company the Heroult electric steel refining process and reported favorably on it, but his recommendations were not followed. It is a pity on account of the private character of this report that extracts from it cannot be given here, as that report was a model of business sagacity, proving that Mr. Wolff was one of the earliest practical iron and steel men who grasped the potentialities of electric refining.

As a result of his firm conviction in the great feature of the electric steel furnace Mr. Wolff became connected with Dr. Paul Heroult as his American representative and as such he developed an untiring activity in interesting the big men in the American iron and steel industry in the electric furnace. By word and by writing he conducted a persistent campaign for the Heroult electric steel furnace. His articles on the use of electric furnace for making alloy steels, on commercial electric steel and gas power and the possibilities of the electric



THE LATE R. H. WOLFF

furnace in steel foundry practice, on the electric furnace for rail and ordnance steel, published in this journal, volume 6, pages 6, 225, and 485 (1908) will even now repay rereading. He was one of the earliest men to recognize the importance of dividing properly the metallurgical reactions between the ordinary oxidizing furnace and the electric refining furnace so as to reduce the cost of electric refining. He took out a patent for a process the chief feature of which was to restrict the treatment in the electric furnace simply to the object of finishing and resting the metal so that any gas or slag left in the metal should rise to the top (this journal, volume 8, page 288, 1910).

Mr. Wolff conducted the negotiations which led to the installation of various Heroult electric furnaces in American steel plants and finally resulted in the acquisition of the American rights in the Heroult furnace by the United States Steel Corporation.

Mr. Wolff was also largely instrumental in the establishment of the ferro-silicon plant of Electro-Metals, Ltd., at Welland, Ontario. He was the president of this company for several years and was on its Board of Directors until the time of his death.

Mr. Wolff was a man of strictest business integrity, a man of strong character and a kind heart. His untimely death is mourned by many friends.

Pulverized Coal as a Fuel

By H. R. Barnhurst

Coal properly pulverized and burned may be made to yield higher economic results than are attainable by any other means.

The requirements necessary to success, while simple, are absolute and must be obeyed.

First—The coal must be dried so that it contains not over 1 per cent of moisture.

Second—The coal must be pulverized to a high degree of fineness.

Third—It must be projected into a chamber hot enough to cause instant deflagration.

Fourth—It must be supplied with air sufficient to yield the oxygen necessary to burn the carbon of the coal at once to CO_2 .

Taking up these requirements in order, the drying of the coal to a moisture content of not over 1 per cent is indispensable. Coal does not grind well if moisture in excess of this be present.

In burning coal the moisture, free or combined, must be disposed of either in the process of preparation or in the moment of combustion. In the latter case not only is the efficiency of the furnace lowered by the calorific investment in the superheated steam passing out as a product, but the temperature of the furnace is lowered materially. The drying of wet coal in the furnace itself, is doing this necessary part of the work in the most expensive place and at the cost of temperatures which may be essential to the industrial process of which high heat is a factor.

Fine grinding—With the best type of machines obtainable for this purpose, the coal and its contained impurities may readily be powdered to such a degree that under the screen tests 85 to 90 per cent will pass through apertures $1/400$ in. square, while the total residuum left upon a screen whose apertures are $1/200$ in. square, will be from $2\frac{1}{2}$ to 5 per cent and this residuum would pass through screens of $1/100$ in. square. It must, however, be borne in mind that of the percentage passing the smaller apertures $1/400$ in. square there is a high percentage of absolute dust or impalpable powder not commercially measurable. This is proven by the fact that in tests made upon calibrated screens of $1/600$ in. square apertures, over 70 per cent still passed through. It certainly appears to be safe to assume, therefore, that the average size of the particles would be below a cube measuring $1/600$ in. on the side.

It may be interesting, therefore, to state that the total numbers of particles resulting from the powdering of 1 cubic inch of coal to the dimensions given would yield 216,000,000 grains of dust. Simple calculation on this basis shows that while a cubic inch of coal exposes 6 square inches for the absorption and liberation of heat, the surface exposed for the same purposes by the powdered coal is 25 square feet. Inasmuch as no fuel burns until it is heated to a temperature at which it develops more heat than it receives, the advantage of this enormous absorbing and delivering surface is apparent. The result of this is shown in the clearness and uniformity of the flame produced. Where coarse particles are permitted to enter the furnace, the distinct sparkles are apparent. These larger particles are carried beyond the region of oxygen supply and are for this reason not fully burned.

Third—While coal ignites freely, in a hot chamber, this ignition means the absorption of heat from somewhere, and if the coal rapidly projected by air does not develop its heat near the point of ignition, means must be devised to maintain the heat necessary for ignition where it is needed, i. e. at the first entrance of the coal into the furnace. It is apparent, therefore, that giving the fuel too great velocity upon entrance is not good practice.

Considering the fourth requirement along with the third we would say that some singular errors and misconceptions have attended the practices of many users of powdered coal.

More particularly do we refer to the use of large fans to supply the air necessary for the projection of the fuel, where the air nozzle has been reduced from 16 in. or 18 in. diameter to 4 or 5 in. at the jet under the expectation that all of the air in the 16 in. or 18 in. pipe would be hurried through the 4 or 5 in. nozzle if not a smaller one. The futility of this is apparent.

To describe the operation more clearly, the coal is received in a bin over the feeders. (Fig. 1.) Its weight is about 38 lb. per cubic foot when loose in the bin. Settling awhile brings the weight to about 45 lb. per cubic foot by displacing the entrained air. Across the bottom of this bin and within a pipe extending horizontally from it is a double-flight worm or feed screw. This double-flight screw resists the tendency of the light coal to flow of itself along the feed pipe. This screw extends over a flanged pipe-cross into which the fuel is delivered. The rear end of the screw is supported by a bearing in a flange on the side of the bin near the bottom, the shaft projecting to receive a driving pulley or chain sprocket. The delivery end of the screw shaft is supported by a bearing in the cover of the horizontal opening of the flanged pipe-cross. The top opening of the cross is uncovered to permit the air

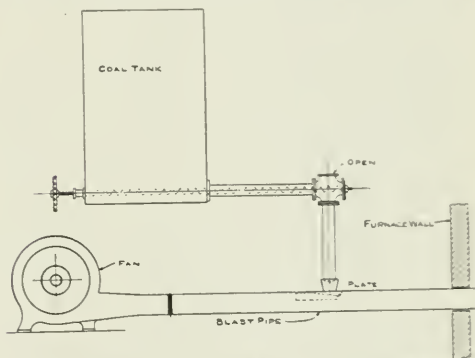


FIG. 1.—ARRANGEMENT OF PLANT FOR USING PULVERIZED COAL AS FUEL

to draw down with the falling fuel. This fuel dispersed in the air so drawn in, descends a vertical pipe attached to the lower opening of the cross, the pipe being long enough to be within the funnel or injection pipe. At the bottom of the funnel is a diagonal plate upon which the fuel falls. The plate is tight against the air pipe up the current and flared open on the side, towards the furnace down the current and takes up about one-fourth the diameter of the pipe. This forms at this point a "vena contracta" producing a suction in the funnel, drawing in through it, supplementary air with the fuel. The fuel spraying upon this plate mixes very thoroughly with the air from the fan, the eddy currents caused by it, assisting very materially its dispersal through the main column of air supplied by the fan.

The admission funnel should be far enough from the furnace to permit this mixture to be thorough. Too high pressures defeat this somewhat, as well as tending to project the fuel too far into the furnace before flashing. As soon as this fuel cloud begins to absorb the heat of the chamber into which it passes, a rapid expansion of the air takes place, separating the particles of fuel in suspension, in the proportions of the absolute temperatures to the temperature of the initial air. It is a matter of discussion whether the best results are obtainable by a delivery of all the air found necessary for combustion by the feed pipe together with the percentage of excess air found to produce the best results, or to use a smaller quantity of air in the feed pipe and look for the further supply from other openings.

Good practice would seem to point to absolute control of

air by the fan and its gates, and the fuel by the varied speed of the feed screw. The furnace should have a good natural draft to a chimney controlled by a damper. It must be remembered that perfect combustion of one pound of carbon demands 2 2/3 pounds of oxygen. This is contained in 11.6 pounds of air or about 15.4 cubic feet; should less than this be supplied a proportionate amount of fuel will be burned to CO with a loss of two-thirds of its initial efficiency, a part of which may be regained by contact with heated oxygen, or it may pass on and burn in the chimney, doing no good if the heated oxygen is brought there in first contact with it. If the oxygen is not heated sufficiently smoke will result and the full heat be undeveloped.

The greater the volatile constituents of the coal the more readily will it deflagrate, as these gases distil from the fuel and ignite at a temperature lower than that required for the carbon itself. Their need for oxygen is, however, greater per pound of fuel (or gases) than that of carbon, and is proportional to the heat evolved. Their average value is in heat units nearly 50 per cent more than that of carbon.

The temperatures attainable with pulverized coal are very high, so high that excess air is commonly admitted in proportions ranging between 50 and 100 per cent. This will be shown by the following table based upon the perfect theoretical combustion of carbon with proportion of air given:

1 lb. carbon with 11.6 lb. air.....	Normal.	4850° F.
1 lb. carbon with 12.76 lb. air.....	10 per cent excess.	4448° F.
1 lb. carbon with 13.92 lb. air.....	20 per cent excess.	4102° F.
1 lb. carbon with 15.08 lb. air.....	30 per cent excess.	3807° F.
1 lb. carbon with 16.24 lb. air.....	40 per cent excess.	3550° F.
1 lb. carbon with 17.40 lb. air.....	50 per cent excess.	3326° F.
1 lb. carbon with 18.56 lb. air.....	60 per cent excess.	3129° F.
1 lb. carbon with 19.72 lb. air.....	70 per cent excess.	2954° F.
1 lb. carbon with 20.88 lb. air.....	80 per cent excess.	2797° F.
1 lb. carbon with 22.04 lb. air.....	90 per cent excess.	2656° F.
1 lb. carbon with 23.20 lb. air.....	100 per cent excess.	2520° F.

In practice the furnace tender speedily becomes educated to the point of judging whether a fire is hot enough by its color and by the length of the flame. The more perfect the conditions the shorter and whiter the flame will be.

The pulverized coal introduces very effectively the element of time into the equation. Given a pound of fuel with say 15,000 heat units, these may all be developed by slow combustion at low temperature, or by burning the fuel in pulverized form, quick combustion gives high temperature. The same quantity of heat developed in both cases, but in one instance in a minute and in the other half an hour.

The influence of preheated air upon the economy of the burning of any fuel resolves itself into ascertaining the quantity of fuel which would be necessary to bring the air to the preheated temperature plus the heating of the excess air also to that temperature. Except as a means of transporting heat excess air has no effect in furnace as far as the fuel combustion is affected. This preheating to be of any economical value must be obtained from heat which would else be wasted. To heat all the primary air necessary to combustion and 50 per cent excess air to a temperature of 1000 degrees in excess of the surrounding temperatures would show a saving of some 4100 heat units or nearly 30 per cent of the fuel value. But few of the industries, however, outside of the metallurgical arts permit the waste heat to pass off at such temperatures and volume as to be available. The regenerative checker-work of open-hearth steel furnaces is the best example of success in this preheating.

In this case it has been a necessity to boost the temperature in this way, because the gases from gas producers burned cold, would not give the temperatures necessary for the work to be done.

We may say that four heat units per degree would represent the saving achieved per pound of fuel fired with 50 per cent excess air by heating all the air admitted to the furnace. The measure of efficiency is dependent upon the loss finally carried away in the rejected gases. In the case of regenerative

furnaces this may be lower than in furnaces not equipped with regenerative checker-work, but if to the percentage of loss with regeneratives be added the losses in the gas producers, the pulverized coal directly fired will afford the greater economy of operation.

With the means at hand of obtaining quickly and safely heat of greater intensity than by the use of coal upon grates or in producers, it would appear to render practicable further steps in many of the arts hitherto restricted by the limitations of furnaces at command.

The fear of explosion of coal need not enter into consideration. Coal lying in a bin or conveyor does not explode. It is only when mixed with air or supported by air currents that coal will "puff." In burning it, therefore, we do not mix the coal and air until just as it enters the furnace at high velocity. Against this column of inrushing air and coal the puff cannot take place.

The air is introduced before the coal is turned on, and the coal is shut off before the air. Only by introducing coal faster than it can burn will an explosion occur and then the effect is trifling. It is the gas produced and not the coal that causes this. It wants oxygen and comes outside to get it.

The presence of impurities in the fuel has not much effect. Of course, only combustibles will burn; the incombustibles are inert and do not affect the operation of the furnace. Their effect is in the lessened results from a dollar's worth of fuel negated by a goodly percentage of waste substance. The writer has burned effectively fuel in which analysis showed 52 per cent of ash. Let us reiterate the conditions—*Dry coal, fine grinding, hot chamber or fire box, proper air supply.*

An important part of the subject must not be overlooked. The durability of the furnace is, of course, vitally essential. In the metallurgical arts when extreme heat is an essential part of the operation, care must be taken to avoid destroying the furnace by its own operation. This is not difficult. Much of the troubles have come from the gases impinging upon the furnace walls at points where change of direction of gas travel is necessary, and from too high velocity of gases due to contracted ports.

If the utilized heat is largely absorbed from the gases by the charge, the waste gases will be proportionately less active in scouring the brickwork. In almost any construction except perhaps a rotary kiln it is found necessary to change the direction of the gases in their progress toward the flue. This change of direction causes the gases to impinge upon the diverting bricks with an energy proportional to their velocity. The brick at these points can be fully protected by a system of water-cooled pipes embedded in the walls. The brick may frit somewhat until the area of protection is reached, when further progress is arrested.

The surprisingly small amount of water which it has been found necessary to introduce, maintaining the outlet below 200° F. proves that the cooling effect is limited to a prevention of cumulative action and is not perceptibly a drawback upon efficiency. Of course, the piping must be so arranged that no air or steam pockets shall exist and that the circulation will be proportional to the heat stimulus. One other point and we will conclude. The pulverized coal furnace has no ups and downs. There is no thick fire or thin fire, fresh coal or old coal to insure fluctuations.

The furnace can be always kept at its best working point and so kept it will be heated equally all over. Of course, a large charge of metal to be heated will by its very volume absorb heat rapidly causing a fall in waste gas temperature and possibly a little smoke at first. This is in the nature of things, but the extremely effective conditions quickly bring the charge to a point where the chill is not sufficient to affect combustion and high temperatures come again and smoke disappears. If the work to be done is constant, there is no reason why high conditions may not be uniformly maintained by proper construction and operation.

We believe the subject has been mastered to a point beyond the experimental stage where the full benefits of high efficiency

may be confidently relied upon in this beautiful method of burning coal. As before mentioned, the quality of the coal is not with this method of supreme importance. Indeed, its great value in the developments of the future may lie in the efficiencies obtainable from low-class or refractory fuels hitherto unavailable.

Fuller Engineering Co., Allentown, Pa.

An Investigation of the Air Lift Pump

The air lift method of pumping, though not highly efficient as compared with some other methods, is nevertheless an important one, owing to the many advantages it possesses over other methods in pumping corrosive liquids and raising large quantities of water from wells of small bore. Notwithstanding the fact that this method of pumping has been known for over a century and is now quite extensively used, the amount of reliable data available to the practicing engineer concerning the performance of this type of pump is very meager.

With the purpose of supplying the demand for reliable data, extensive experiments were undertaken in the hydraulic laboratory of the University of Wisconsin under the direction of Messrs. George J. Davis and Carl R. Weidner, assistant professor and instructor, respectively, in hydraulic engineering in the university. The results of the investigation have been published in *Bulletin* No. 450, Engineering Series, Vol. 6, No. 7, University of Wisconsin, which may be obtained by remitting 40 cents to the Secretary of the Regents, Madison, Wis.

The *Bulletin* gives interesting information on the history and development of the air lift pump, showing that the principle was first used by Carl Emanuel Löscher, a German mining engineer, in 1707. The principle of the air lift is discussed and the various theories of its operation are given. The four types of air lift pumps are illustrated and described, namely, the side inlet, the annular air tube, the central air tube and the combination (annular and central air tube).

Among the disadvantages of the air lift pump are cited its low hydraulic efficiency, 25 per cent to 33 per cent; the great depth of submergence required; its limited efficiency in horizontal pumping, and the aeration, which in some circumstances is a disadvantage, causing rusting of the eduction pipe and the deposition of salts in foot-piece.

The principal advantages lie in its large capacity, low maintenance cost, low operating cost, immunity from effects of high temperatures in the liquid pumped, and its reliability.

The investigations at Wisconsin are described in detail, and the results of other experiments are also given. The variables which affect a particular size and type of pump are the following: (1) percentage of submergence; (2) lift; (3) discharge; (4) volume of air; (5) pressure of air. The conclusions which may be justifiably deduced from the Wisconsin experiments are given below, and hold only for the particular size, type and length of pump on which the experiments were performed. The inference, however, may be drawn that these conclusions would hold for other types and sizes.

(1) The central air tube pump has the greatest theoretical capacity for a given size of well.

(2) The coefficient of pipe friction and slip decreases as the discharge increases, and decreases as the ratio of volume of air to volume of water increases.

(3) The coefficient of pipe friction and slip varies with the length of the pump, but seems to be independent of the percentage of submergence and of the lift.

(4) The length of pump, the percentage of submergence, and therefore the lift, remaining constant, there is a definite quantity of air causing the maximum discharge. This quantity of air for maximum discharge, as also the ratio of volume of air to volume of water, differs for different percentages of submergence and lift, the length of pump remaining constant.

(5) The length of pump remaining constant, the maximum output, e.g., foot-gallons, occurs at about the same percentage of submergence for all rates of air consumption, being from 61 per cent to 65 per cent for the pump used in the Wisconsin

experiments. At other submergences the output varies as the ordinates of a parabola having a vertical axis. Under these conditions the lift does not remain constant as the percentage of submergence varies.

(6) The length of pump and percentage of submergence remaining constant, and therefore constant lift, the efficiency increases as the input decreases, that is, the higher efficiencies are obtained at the lowest rates of pumping.

(7) By varying the percentage of submergence, and therefore the lift, the length of pump remaining constant, the maximum efficiency is obtained at approximately 63 per cent submergence for all rates of input or discharge.

(8) The lift remaining constant, the efficiency increases as the percentage of submergence increases, for all rates of input and all practical percentages of submergence.

(9) With the same size and type of pump, the percentage of submergence remaining constant, the efficiency increased as the lift increased for the small lifts experimented on, that is, up to about 24 ft. From a theoretical study, however, the indications are that a point will be reached from which the efficiency will decrease as the lift increases.

(10) Other conditions remaining constant, there is no advantage to be gained by introducing compressed air above the surface of the water in the well.

(11) The type of the foot-piece has very little effect on the efficiency of the pump, so long as the air is introduced in an efficient manner and the full cross-sectional area of the eduction pipe is realized for the passage of liquid. Anything in the shape of a nozzle for increasing the kinetic energy of the air is detrimental.

(12) A diverging outlet which will conserve the kinetic energy of the velocity head increases the efficiency.

A Test of an Edison Primary Battery (Edison-Lalande Cell)

By Dudley Sanford

A precision to within 1 per cent is sought in this experiment.

The battery tested was a new Edison primary battery, type "RR," charged according to the directions inclosed with the cell. All battery connections were soldered to avoid contact resistances.

The test was made by the well-known condenser method. This method involves the determinations of the p. d. across the terminals of the cell when it is on open circuit and when it is closed through a fixed resistance. To obtain these a condenser is charged to the unknown potential and is immediately discharged through a ballistic galvanometer, whose constant has been previously determined by charging the condenser to the potential of a standard cell and allowing it to discharge through the galvanometer. $d = kCE$, where d is the deflection using the standard cell, C the capacity of the condenser, E the e.m.f. of the cell, and k is a constant. Using another cell, $d' = kCE'$. Consequently

$$\frac{E}{kC} = \frac{d'}{d}$$

The apparatus consisted of a 15-microfarad condenser, an external resistance of 0.126 ohm (from terminal to terminal of the cell), a D'Arsonval galvanometer with telescope and scale, and a charge and discharge key (K in diagram).

The procedure followed was this:

At time 0 min. open circuit reading taken with key K' open.
At time 1 min. closed circuit reading taken with key K' closed.
At time 2 min. open circuit reading taken with key K' open.
At time 3 min. closed circuit reading taken with key K' closed.
etc., to end of 60 min. K was then opened permanently, and readings were taken at intervals of 2 minutes. The internal resistance $x = R(F - E')/F'$, and the current $I = F'/R$, where F represents open-circuit voltages, E' closed-circuit voltages, R the external resistance in ohms, and I the current in amperes. The curves showing these quantities are given.

There are several interesting facts to be noted from these curves. The cell has an abnormally high initial e.m.f., 1.03 volts, which drops very rapidly for a short time and is then practically constant at 0.7 volt, though decreasing slightly. The p. d. at the terminals, however, increases gradually at first, but at the end of 60 minutes is nearly constant at 0.55 volt. The current is seen to rise rapidly at first and even at the end of the test is still rising, though not so rapidly. This is ex-

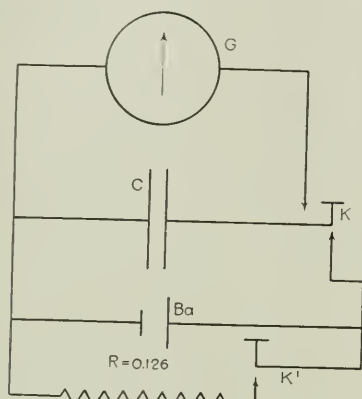


FIG. 1.—ARRANGEMENT OF TEST

plained upon observing the internal resistance curve, for this resistance falls as the cell is used and is at all times low, being but 0.037 ohm at the end of the test. The cell recovers its normal voltage in a comparatively short time.

After running the above test the battery was placed in series with an ammeter and a copper voltmeter and allowed to run until completely exhausted. A voltmeter with a key in circuit was placed across the battery terminals. Readings were taken on the two instruments at suitable intervals of time, which ranged from two hours to thirty hours. The voltmeter readings were correct for the IR drop due to the internal resist-

zinc consumed was 472 g., or 1.2 g. per amp-hour, or 0.000333 g. per coulomb. The accepted value for the electrochemical equivalent of zinc is 0.0003387 g. per coulomb, a difference of only 1.4 per cent.

But, in general, the value of the cell will depend not only upon the quantity of electricity which it will give, but also upon the work it is capable of doing. To know the capacity of a cell only in amp-hours gives inadequate information on which to determine its value to the user. But multiplying the volts by the amperes and integrating over the time of the test gives the watt-hours, a very important factor to be known. This particular cell gave 181 watt-hours of work. Some of this is always lost internally, and the efficiency of a cell varies with the constants of the circuit of which it is a part. But taking the makers' rate of maximum continuous output, 6 amp. and the voltage of 0.7 as given by the test, we find that the efficiency would be 79 per cent. The useful work obtainable would then have been 143 watt-hours. This, however, is the lowest efficiency for continuous output.

From the weighings of the zinc we find that 3.3 g. were consumed per watt-hour, or 3300 g. per kw-hour. The manufacturers' list price for the zinc plates is \$0.00091 per gram. Consequently the cost of producing energy at the lowest efficiency is \$3 per kw-hour. At 100 per cent efficiency it would be \$2.37 per kw-hour. This is on the assumption that the zinc is the only substance consumed in the production of available energy and that the other materials will be recovered. If, however, the user will throw away the remaining materials and buy a complete renewal the cost will be \$11.90 and \$9.40 per kw-hour at the two efficiencies. If in either case the materials are bought in large quantities this cost may be much reduced—50 per cent in lots of \$2,000 or more and proportionately for intermediate quantities. We thus note that under favorable conditions electrical energy obtained from this highly efficient type of primary battery costs around fifty times the ordinary market price of electrical energy.

If the total energy of the cell externally available at its lower efficiency were applied to raising a kilogram weight it would raise it 52,500 meters, or a ton weight a distance of 192 ft. At its full efficiency it would raise a kilogram weight 66,400 meters or a ton weight 240.3 ft.—a total energy of nearly 250 ton-feet. The total weight of the cell was about 20 lb.

The manufacturers of this cell now have a cell (BSCO No. 401) which they claim, on the basis of their tests, is even more efficient than this, yielding 247 watt-hours, or 1.36 times the energy of the type "RR" cell, but its cost for a complete renewal is 1.24 times that of the "RR" cell. Consequently from the cost standpoint it is 10 per cent more efficient than the "RR" cell. The action of the cell, however, is practically the same as is shown in the curves of the first part of this experiment.

NOTE:—The above test was made under my immediate observation and I believe the results are reliable.—EDWIN F. NORTHRUP.

Palmer Physical Laboratory,
Princeton, N. J.

The Data of Geochemistry is the title of Bulletin No. 491, issued by the U. S. Geological Survey, which will be sent free on application. It comprises a manual of geologic chemistry, including chapters on the nature distribution and relative abundance of the

chemical elements, the composition of the atmosphere and of volcanic gases and sublimates, the mineral content of surface and underground waters, the nature of saline residues, the molten magma of the earth's interior, the rock-forming minerals, the composition of igneous, sedimentary and metamorphic rocks, rock metamorphism and decomposition, metallic ores, natural hydrocarbons, coal, lignite and peat. The Bulletin is exceedingly interesting from more than one viewpoint.

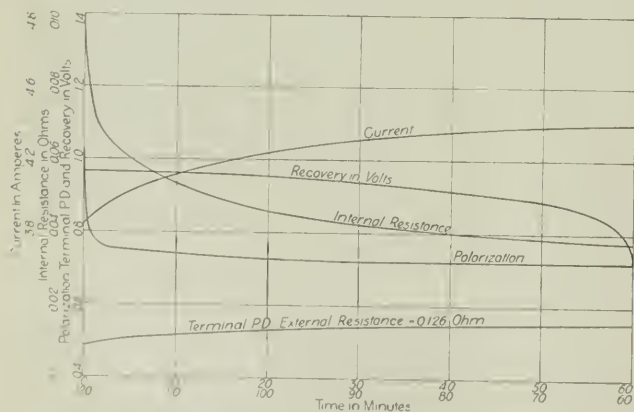


FIG. 2.—TESTS OF EDISON-LALANDE CELL.

ance of the cell. From these readings and the proper weighings these results were obtained:

According to the readings of the instruments the total amp-hours (excluding first test) was 389.3. By weighing it was found that 466.05 g. of copper had been deposited, and this corresponds to 392.88 amp-hours. This would show that the readings were correct to 0.0 of 1 per cent. An accuracy to 1 per cent is therefore assumed in the other data obtained. The

Minerals Separation Flotation Plant at Kyle
Copper Mines, N. L.*

By H. Hardy Smith

The ore treatment is almost pure copper pyrites (chalcopyrite) in a gangue of quartz and quartz-felsite. Very little free iron sulphide (pyrite) occurs, and at the time of the erection of the flotation plant the stopes in the oxidized zone had been almost entirely depleted.

This zone only extends to a shallow depth, and, below the 100-ft. level, the sulphide for the most part is clean and

sorted by boys, both pitl ore and waste being picked out. The conveyor discharges into the mill storage bin, of capacity sufficient to keep the mill going for 24 hours.

The crusher, belt conveyor, and pickers only work day-shift

From the mill storage bin the ore is fed by a push feeder to a bucket elevator, discharging on to a shaking screen (48-in. holes, standard spacing). The oversize goes to 30-in. Cornish rolls, and the product from the rolls is elevated by a bucket elevator back to the screen. The undersize from the screen goes to a classifier; spigot to four-compartment May jig.

When the tables were in use the product from No. 1 hutch ran to concentrate bin; Nos. 2 and 3 hutch spigots to a No. 5 Wilfley table; concentrate to bin; tailing to No. 5 Krupp ball mill; No. 4 hutch spigot direct to ball mill; jig tailing to classifier; spigot to waste as jig tailing, overflow to ball mill.

The ball mill crushed through steel screens with clever 3/64-in. round holes to the linear inch.

From ball mill the pulp ran to a three-compartment classifier: first compartment spigot to No. 5 Wilfley; second to another No. 5 Wilfley; third to a Card table; overflow to a baffle board slime thickener; thickened slime to a True vanner, concentrate to bin; tailing to waste. The concentrates from the two Wilfleys and the Card table ran to bin; the middlings to another No. 5 Wilfley, and the tailings to waste. The concentrate from the last Wilfley ran to bin; the middling was returned over the same table, and the tailing ran to waste. Reference to the flow sheet (Fig. 1) will make the scheme clear.

With this arrangement the average extraction was about 74 per cent of the total copper in the feed to the mill. The tailings varied from 1.5 per cent to 1.8 per cent copper, the loss chiefly occurring in the fine and slime. The concentrates varied from 20 per cent to 21 per cent copper.

First Trial of Flotation

Considering the type of plant, these results were very satisfactory, but the advantages of concentration by flotation having

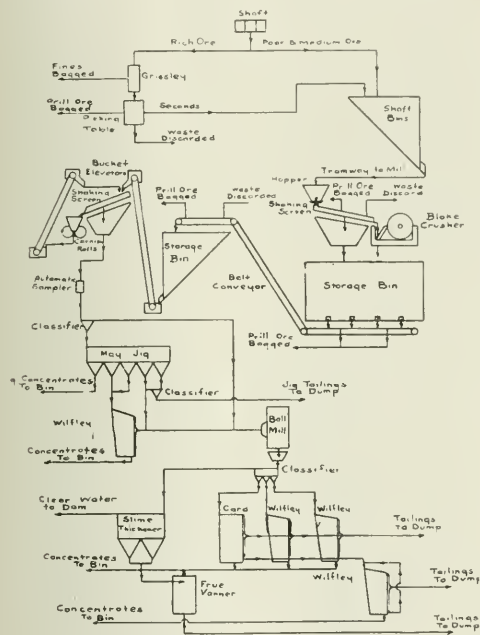


FIG. 1.—ORIGINAL CONCENTRATION PRACTICE.

bright; but in the crushed and brecciated parts of the ore channel, where surface water has circulated freely, the gangue is decomposed and kaolinized, and the copper sulphide badly tarnished.

This class of ore gave considerable trouble at the mill, but, fortunately, was very limited in quantity. A sample of rich ore assayed copper, 21.2 per cent; iron, 24.3 per cent; silica, 28.9 per cent; sulphur, 25.1 per cent.

Original Concentration Practice

The concentration in the original mill was effected by hand-picking, jigging, and tabling. A considerable portion of the copper mineral arrives at the surface in the form of large lumps of very pure quality, eminently suitable for sorting out by hand.

The rich ore is tipped over a grizzly at the shaft bins; the over-sized hand-picked, and the undersize bagged direct.

The medium and poor grades are tipped directly into the bin and trucked thence to the mill. The trucks discharge into a hopper, from which the ore is hand-fed through a chute to a shaking screen (1½-in. round holes) on which it is hand-sorted. The oversize goes to a 15-in. x 9-in. jaw crusher of the Blake type, and is reduced to 1-in. ring size. The undersize from the screen and the crushed ore fall into a bin, and is fed by push feeders to an 18-in. belt conveyor, on which it is hand-

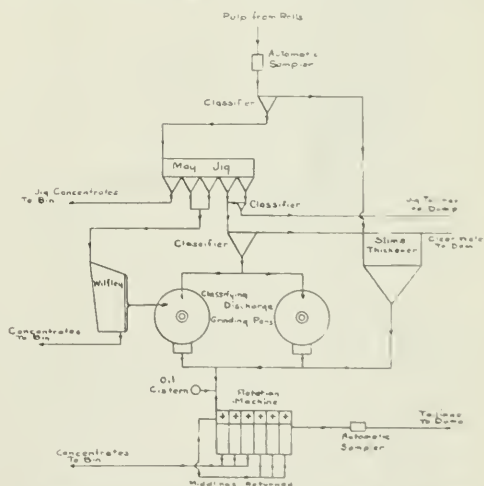


FIG. 2.—CHANGED CONCENTRATION PRACTICE WITH FLOTATION

been brought under the notice of the management, tests were carried out on samples of the milling ore by the Minerals Separation, Ltd., in Melbourne. The results were such that it was decided to replace the tables with a Minerals Separation unit.

The mill building was extended to house the new plant, which, in the original design, consisted of a baffle board slime thickener, two grinding pans, and a flotation machine. The departure from the old flow sheet commenced at the jig.

No. 1 spigot ran to concentrate bin as before; Nos. 2 and 3 to Wiltley table, and table concentrates to bin as before, but tailings to grinding pans. No. 4 spigot product of jig, and

*Excerpt from the *Proceedings*, Australasian Institute of Mining Engineers, No. 7, September, 1912.

classifier spigot product at foot of jig were sent to the pans, and thence direct to flotation machine. Overflow from classifiers at head and foot of jig was sent first to slime thickener and thence to flotation machine. The scheme is shown in flow sheet, Fig. 2.

The grinding pans were 8 ft. in diameter, running at 30 r.p.m., of the classifying discharge type.

Description of Flotation Machine

The flotation machine is of the standard Minerals Separation type, and is classed as a 12-in. unit. For the purpose of making

charged from the bottom of No. 6 hutch through valve *O* which is controlled by suitable means so that the overflow from the lips of the six hatches can be adjusted to what is required.

The stirrers *C* are either driven from a countershaft by quarterturn belts or else by bevel gears, the latter being positive in their action, but objectionable on account of noise and excessive wear and tear.

If required, a middling can be made and returned to the bottom of No. 1 agitating box through pipe *P*.

First Results Unsatisfactory, Necessitating Alterations

When put into operation the results from this plant were not quite satisfactory, and two members of the Minerals Separation staff were sent to investigate. Many minor faults were disclosed, which did not permit proper regulation and adjustment of the machine. The plant was closed, the machine overhauled and radically altered.

The slicing valves, operated by a lever to close or open the mouths of the pipes leading from the bottom of the hatches to the agitating boxes, were replaced by simple flap valves operated through a $\frac{1}{2}$ -in. rod by a thread and hand wheel.

The slicing valve for regulating the discharge of tailings from the machine, originally operated direct by hand, was also altered so as to be controlled by hand wheel and thread.

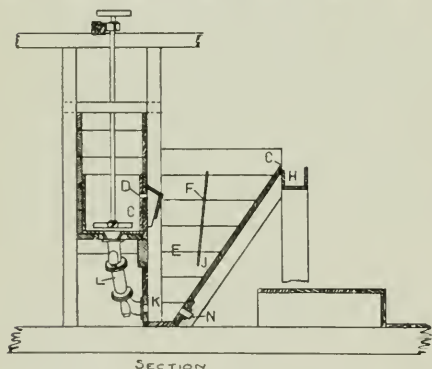
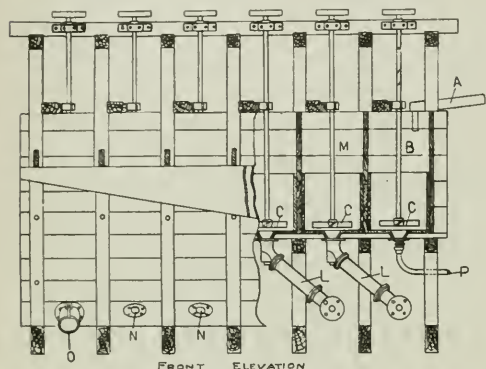


FIG. 3.—FLOTATION MACHINE

its operation clear to those unacquainted with it, the accompanying sketches (Fig. 3) are submitted.

The pulp with requisite amount of oil added, runs to machine by launder *A*, and discharges into No. 1 agitating box *B*, where it is violently agitated by stirrer *C*, revolving at about 450 r.p.m., with an upward throw.

The aerated pulp escapes through a slot *D* (section), and is deflected downwards into hutch *E*. Bubbles with sulphides attached rise against baffle *F* and form a dense froth on the surface of the hutch, which gradually moves forward and overflows at lip *G* into launder *H*. Any gangue particles attached fall out and pass downwards through space *J* to mix with the remaining pulp and pass out through orifice *K* into pipe *L*, leading to the bottom of No. 2 agitating box *M*.

The overflow from lip *G* is controlled by regulating the size of orifice *K* by means of a suitable valve. At the bottom of each hutch is a hole *N* blocked by a wooden plug or suitable valve, for sluicing out the contents of the hutch when closing down. At the elbows of pipes *L*, are similar holes and plugs for the same purpose.

In No. 2 agitating box *M* the operation is repeated, and so on throughout the six boxes and hatches. The residue is dis-

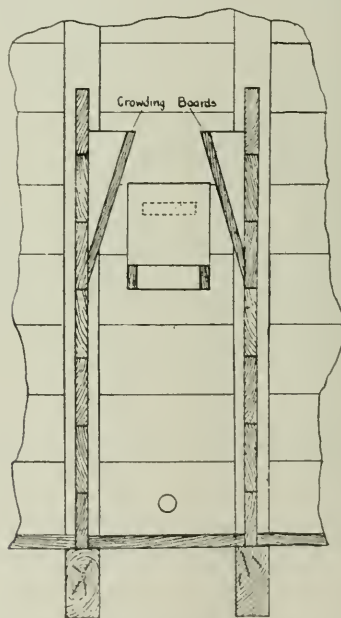


FIG. 4.—CROWDING BOARDS

This arrangement permitted very delicate adjustment of the overflow from the hatches, which was impossible under the old conditions.

The width of all six hatches being the same, the froth became very thin and poor on the last four. For the purpose of contracting the surface and giving greater thickness and body to the froth, "crowding boards" were put in, which gradually decreased the width of the hutch surface from 11½ in. on No. 1 to 4½ in. on No. 5. See Fig. 4.

With the idea of giving the sulphides less chance of dropping out of the froth, No. 6 hutch was shortened by replacing the baffle with a watertight bulkhead and leading the froth to the front of the machine in a launder. By this means the chance of mineral being lost in the tailings was considerably lessened as the settling space was halved.

The surface widths of the hutches were now as follows:

No. 1	11½ in.
No. 2	11 in.
No. 3	7½ in.
No. 4	5½ in.
No. 5	4½ in.
No. 6	4 in.

A sample of the overflow from each hutch gave the following results:

No. 1		No. 2		No. 3		No. 4		No. 5		No. 6	
Cu. %	Insol. %	Cu. %	Insol. %	Cu. %	Insol. %	Cu. %	Insol. %	Cu. %	Insol. %	Cu. %	Insol. %
27.2	18.9	24.1	27.8	22.3	28.0	15.6	39.8	14.3	45.0	9.4	61.4

This test showed that the grade of concentrates from the last three hutches was not high enough to permit of them being sent to the concentrate bin, so arrangements were made for returning a middling to the machine.

The overflow from the last three hutches was fanned back by a bulkhead at the discharge end of the concentrates launder in front of the machine, and returned to No. 1 agitating box by a 1-in. pipe.

The concentrates from the first three hutches were carried by a tin tray over the bulkhead into another launder running to the concentrate bin.

To get at the insides of the agitating boxes the machine practically had to be dismantled. To obviate this, a rectangular hole was cut in the back of each box, so as to leave about 1½ in. of timber on each side, for the purpose of bolting on a door. The insides of the boxes were now readily accessible, and alterations could be carried out.

Examination showed that the 1½ in. redwood had worn badly, although the machine had only been in operation a few weeks. Liners were made from ¼-in. steel plate and bolted to sides and bottom in order to protect the woodwork from wear.

The orifice in the bottom of each box was at the same time made bell-mouthed, as it was found that by this means the suction caused by the stirrer could be largely increased.

After these alterations had been completed the plant was restarted, with the following results:

Run	Feed % Cu.	Concs. % Cu.	Tails. % Cu.	Extraction %
1	5.1	20.1	2.0	67.5
2	4.4	24.2	2.3	52.7
3	4.0	19.1	1.25	73.8
4	4.7	19.0	2.2	60.1
5	4.0	23.6	1.5	66.7
6	4.1	23.5	2.25	49.8

Formula for Calculating Results

The figures given refer to the machine considered as an isolated unit, and the extractions are the theoretical extractions worked out from the assay values of the feed, concentrate and tailing according to a method which was as follows:

Let x = weight in tons of concentrate when 100 tons of feed is treated.

" a = percentage of copper in feed by assay.

" b = percentage of copper in concentrate by assay.

" c = percentage of copper in tailing by assay.

Then

$(100 - x)$ tons = weight of tailing produced.

Now wt. of Cu. in feed = weight of Cu. in conc. + wt. of Cu. in tailing.

Therefore a ton of Cu. = $\frac{bx}{100}$ tons Cu. + $\frac{c(100 - x)}{100}$ tons Cu.

Putting this as an algebraic equation and solving, we get:

$$\frac{bx}{100} + \frac{c(100 - x)}{100} = a$$

$$x = 100(a - c)/(b - c) \text{ tons.}$$

Hence wt. of Cu. in concentrate = $b(a - c)/(b - c)$ tons.

Hence percentage recovery = $100b(a - c)/[a(b - c)]$.

By substituting the assay values for the symbols in this formula, the percentage recovery can be worked out very readily without knowing any actual weights. It was found exceedingly useful throughout the investigations into the working of the plant, and was used not only for estimating extractions, but also in the solving of other problems, such as the most economical grade of concentrate to produce.

A grading analysis was carried out to discover where the losses occurred. The results were as follows:

Mesh	FEED		CONCENTRATE		TAILING		Extraction %
	Weight %	Copper %	Weight %	Copper %	Weight %	Copper %	
+ 20	1.52	2.2	0.18	12.5	1.85	2.8	11.3
+ 40	13.27	1.2	0.3	17.0	17.53	1.2	4.4
+ 60	15.20	0.8	0.31	18.5	20.77	0.7	11.7
+ 80	3.55	1.15	0.52	24.7	6.10	0.6	38.9
+ 100	22.5	1.4	7.42	23.6	21.4	0.6	28.6
+ 130	3.62	3.8	1.55	21.2	4.2	0.5	21.4
- 130	40.4	5.45	89.7	23.75	28.0	1.2	77.9

Close Sizing Required

These figures showed the necessity for finer grinding and closer positive sizing of the feed to the machine. The presence of coarse particles is even more injurious to the result than the figures show, as the coarse sulphides, in falling through the froth, detach some of the finer material which would otherwise float off. For the purpose of finding out what work the grinding pans were doing, a grading analysis was made, with the following results:

Mesh	No. 1 PAN FEED		No. 1 PAN DISCHARGE		No. 2 PAN FEED		No. 2 PAN DISCHARGE	
	% Wt.	% Cu.	% Wt.	% Cu.	% Wt.	% Cu.	% Wt.	% Cu.
+ 16	16.4	2.3	...	...	10.0	1.4	...	...
+ 30	33.9	1.6	10.0	1.7	19.7	1.0	0.3	...
+ 40	13.0	1.3	17.1	1.2	10.5	1.2	11.0	1.4
+ 60	18.3	1.2	31.9	1.4	18.0	1.3	30.9	2.0
+ 100	6.9	2.1	16.9	1.2	17.0	3.0	23.4	4.1
- 130	11.5	3.7	24.1	5.0	24.8	6.7	34.4	6.2

Depth of discharge:—No. 1 pan, 21 in. above top of dies.

No. 2 pan, 22 in. above top of dies.

With the idea of reducing the +40 product, the lips of the pans were slightly raised and a sizing test made, which showed that the quantity of slime (—130) produced was largely increased, while the +40 product was only slightly reduced.

In spite of improvements in the flotation machine, difficulty was experienced in regulating the overflow from the hutches, and consequently the quality of the work done depended largely on the skill and agility of the man in charge.

The trouble appeared to be due to two causes:

(1) Fluctuation in the quantity of feed coming to the machine.

(2) Fluctuation in the speed of the stirrers.

The latter were driven from the main mill engine, the speed of which altered slightly according to the load on the pans and other units.

The froth on the hutches continued to be thin, with little body and holding power. This was probably due to three causes:

(1) The aforementioned fluctuations in the overflow

(2) Excessive amount of oversize in the feed.

(3) Excessive dilution of pulp in the machine

A test showed the latter to be to 1, and owing to the arrangement of the mill no water could be cut out of the circuit.

A theory subsequently exploded was advanced that the poor froth was due to insufficient sulphide mineral in the pulp, and

the return of the overflow from the fourth, fifth and sixth hutches was expected to rectify the trouble by keeping a certain amount of mineral in circulation.

Final Rearrangement of Mill

It was finally decided that satisfactory work was impossible under the existing conditions, and the plant would have to be closed down and rearranged. The new plant was to embody the following improvements:

- (1) Alteration of pans from classifying discharge to positive central feed type.
- (2) Positive screening of all feed to machine.
- (3) Thickening feed to machine, at the same time forming a reservoir from which a constant supply could be drawn.
- (4) Independent motive power for stirrers, so that speed could be kept constant.
- (5) Apparatus for regulating and keeping constant the supply of oil to the pulp.

The flotation plant was closed and the tables re-started. Plans and designs were prepared along the above lines, and the alterations carried out, occupying about eight weeks.

In the rearranged plant tabling is done away with altogether.

No. 1 jig hutch spigot runs to concentrate bin, as before. Nos. 2, 3 and 4 spigots to classifier; spigots to pans; overflow to thickening box before machine; jig tailing to classifier; spigot to waste (temporarily); overflow to pans. The feed is equally distributed between the two pans by a distributing box. Discharge goes from pans to a bucket elevator, thence to revolving screens; undersize from screens to thickening box before machine; oversize back to pans. The overflow from classifier at head of jig also ran on to the screens.

The spigot of the thickening box ran to machine; overflow to slime thickener; thick slime to machine; clear water to dam. The oil dropped from a cistern into the launder leading to the machine. Reference to the flow sheet (Fig. 5) will make the scheme clear.

The only entirely new units were the revolving screens and the bucket elevator; the rest was merely a re-arrangement of existing plant.

The elevator was constructed with wooden boot of ample size to prevent excessive wear; 3-ft. pulleys, 22-ft. centers; 12 in. x 8-ply Balata belt, traveling 300 ft. per minute, with 10 in. x 6 in. x $6\frac{1}{4}$ in. buckets spaced 15 in. apart; lip of discharge 6 ft. below center of top pulley, to allow buckets to clear themselves, and each bucket had three $\frac{3}{16}$ -in. holes bored in the bottom to assist in this. After the buckets were smooth this elevator did very good work, and so far has given next to no trouble. The screens are revolving trommels of the improved jig type, driven by worm gearing so as to revolve at 11 r.p.m. They turn in opposite directions, and are fed from a distributing box placed between them, having two discharge lips, designed to spread the pulp evenly over the screens. By slipping a board in, the pulp can be cut out from either screen at will. The worm gears are attached to the shaft by a clutch, so that either screen can be stopped when necessary. This arrangement is very compact and neat. The trommels take 36 sq. ft. of wire screen to cover them, and the screen cloth lasts from ten to twelve weeks, according to the mesh and gauge.

Screen Analysis

The screens used were Greening's L. W. B. (600 mesh, 0.0164 in. diam. of wire, 0.0244 in. aperture) on one trommel, and Greening's A. K. D. screen (900 mesh, 0.0116 in. diam. of wire, 0.00217 in. aperture) on the other.

A grading analysis of the product from the A. K. D. screen was as follows:

Mesh	% Weight	% Cu	Mesh	% Weight	% Cu
+40	0.8	1.2	+100	7.7	2.7
+60	18.2	1.4	+130	2.6	2.8
+80	22.9	2.1	-130	47.8	4.8

The product from the L. W. B. screen gave the following figures:

Mesh	% Weight	% Cu	Mesh	% Weight	% Cu
+40	0.7	0.8	+100	8.5	3.3
+60	29.0	1.2	+130	4.3	3.7
+80	23.7	2.3	-130	31.8	5.3

These figures were considered more satisfactory, so henceforth L. W. B. trommel alone was used.

The alterations to the pans consisted in the removal of the baffles and classifying discharge lips, enlarging the discharge holes, and placing a cylinder round the center of the pan with watertight joint at bottom, to take the central feed.

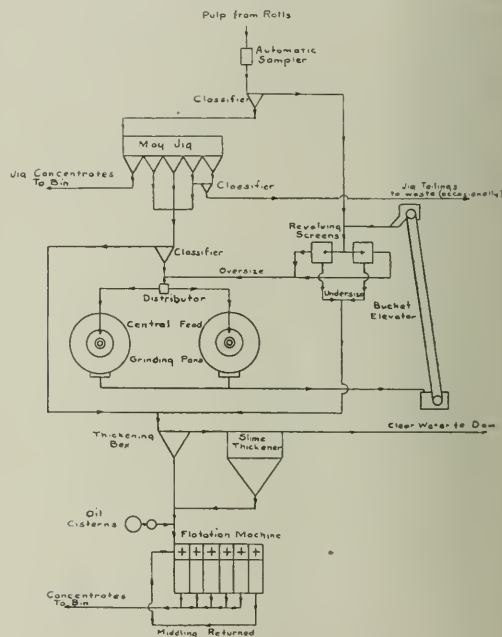


FIG. 5.—TRIAL ARRANGEMENT OF MILL.

It was noticed that, unless the pulp circulated freely from periphery to center by way of the curved channels between the shoes and dies, the pans did bad work, and the launders and screens soon became overloaded. The discharge was baffled to 7 in. above the tops of the dies in the hope that the head thus created would assist the circulation. This resulted in the pans choking so hopelessly that the engine stopped. The pans were then thoroughly cleaned out, the baffle reduced to 4 in., the shoes put hard down, and more water added to the feed.

This brought about the desired result, and the pans did splendid work, as the following grading analyses show:

Mesh	PAN FEED		PAN DISCHARGE	
	% Weight	% Copper	% Weight	% Copper
+16	27.8	1.1	0.4	1.1
+20	10.2	1.2	1.8	1.0
+40	21.2	1.2	12.0	1.0
+60	21.2	1.3	34.0	1.1
+80	8.5	2.5	25.3	1.6
+100	1.8	4.3	7.4	2.7
+130	0.9	4.8	2.9	3.1
-130	6.4	5.1	16.2	3.8

When the sample was taken all the jig tailings were running to the pans. The mill has a capacity of about 60 tons per 24 hours, so, under these conditions, after allowing for jig concentrates and slime eliminated in classifier, each pan was treating pulp at the rate of 25 tons per 24 hours and consuming about 10 hp. in the operation.

Using these figures in conjunction with the above grading analysis, the relative grinding efficiency, according to Stadler's method, works out at 10.4.

Unfortunately, with the pans running under the above conditions (i.e., shoes hard down) the load on the mill engine was too great, and it had a tendency to slow up, so the shoes had to be eased a little. This had the effect of interfering with the circulation in the pans, and a grading analysis showed the relative efficiency to have dropped to 2.6.

Owing to the lack of power the jig tailings had to be run to waste for about 3 hours out of the 24. This trouble has now been overcome by the installation of a small auxiliary engine.

Consistency of Pulp for Flotation

A pulp going about three of water to one of solids can be run to the flotation machine under the following conditions:

Thickening Box.	Slime Thickness.
Depth of discharge, 2 ft.	5 ft. 8 in.
Size of spigot, $\frac{3}{4}$ in.	$\frac{3}{4}$ in.

If necessary, it can be brought as low as two of water to one of solids by altering the spigots and depths of discharge. The dilution in the flotation machine itself is always higher than that of the pulp fed to it, on account of the large proportion of the water in the middling returned. The oil-feeding apparatus consists of two cisterns, one at the back of and above the other. The top one is fitted with a float valve, the float of which rests on the surface of the oil in the one below, and by this means the depth of oil, and likewise the pressure at the tap is kept constant. The drops from the tap are found to remain remarkably constant, except when dirt or water is allowed to get into the oil.

The engine driving the stirrers is a small 6-in. x 6-in. vertical Tangeye engine, running at 250 r.p.m. and 100 lb. boiler pressure. It runs light, developing about 4 hp., and is governed by an ordinary centrifugal ball governor of Pickering type.

The cast-iron stirrers last from 4 to 5 months, and are screwed to the end of the spindles, the direction of rotation tightening them.

The only alterations to the flotation machine were in connection with the returning of the middling and the removal of the concentrate. The middling return pipe was increased to 2 in. in diameter, and holes were left at the bends to enable the pipe to be cleaned out when choked, by inserting a length of iron wire. This operation was very seldom necessary.

Another launder was added to the front of the machine, the inside one carrying the middling and the outside one the concentrate. Removable tin trays carry the concentrate across to the outside launder, and when necessary the tray is removed and the overflow from any hatch is allowed to return with the middling. With the old arrangement enough fall could not be given to the tray carrying the concentrate, and in consequence it had continually to be cleared with a scraper.

When put into operation the new plant soon justified the alterations. Once regulated, the overflow from the hutchers remained steady and constant for an indefinite period, provided nothing went wrong with the rest of the mill.

On several occasions the flotation machine ran for a couple of days without any attention other than oiling. When all the jig tailings were being treated the concentrate from No. 4 hatch was so rich that it was run direct to the concentrate bin, and by adjusting the machine the grade of No. 5 could be raised sufficiently to do the same with it. This left the overflow from No. 6 hatch alone to be returned as a middling.

Strange to say this alteration had no apparent effect on the grade of the bulk concentrate produced, and certainly no deleterious effect on the efficiency of the machine.

The re-agent used is *Eucalyptus* dries oil, distilled in the bush country at no great distance from the mine. It is fed to the launder leading to the machine at the rate of 12 cubic centimeters per minute, which works out at a little under $\frac{1}{2}$ -lb. per ton ore. It costs 8d. (\$0.16) per lb. delivered at the mine.

No acid at all is used, one attempt to do so proving disastrous.

Effect of Oxidized Ore

A certain class of ore frequently gave trouble at the machine, and when present in the feed invariably decreased the extraction. This type of ore was that mentioned at the beginning of these notes—i.e., decomposed, oxidized, and kaolimised lode matter.

On one occasion, when the pulp was discolored from a whitish-grey to a dirty yellow by this material, the process became inoperative, no true froth forming whatsoever, and the pulp leaving the machine at practically the same value at which it came in. Large slimy bubbles and small masses of coagulated sulphides covered the hutchers.

The bubble films were very tenacious, the bubbles clinging to the sides of the hutchers and allowing the turbid water to flow away from beneath them.

No improvement took place while the pulp showed any signs of discoloration. Salts in solution, or acidity of the pulp and excessive fine slime were considered likely causes of the trouble. The above extreme conditions were never again encountered, but on several occasions trouble was experienced with an excessive quantity of froth, made up of large bubbles with slimy, cohesive films.

A sample of pulp taken while these conditions ruled gave the following results:

- (1) The decanted solution contained 36 grains to the gallon of salts in solution.
- (2) The decanted solution was neutral to litmus.
- (3) 52 per cent of the solids in the pulp passed 130-mesh sieve.
- (4) The -130 product assayed 0.11 per cent of oxidized iron

The +130 product assayed 0.03 per cent of oxidized iron

The bulk sample assayed 0.07 per cent of oxidized iron.

(NOTE.—The oxidized iron was taken as a measure of the oxidized material.)

After this each daily sample of feed was assayed for oxidized iron, but no definite relation could be detected between the amount of oxidized iron present and the result of the day's run; but amounts above 0.05 per cent seemed to have bad effects, and when above 0.07 per cent the results were invariably poor.

The deleterious effect of the oxidized and decomposed material seemed to be to its physical rather than its chemical properties, and the peculiar nature and excessive quantity of the slime formed from it appeared to be responsible.

Favorable Percentage of Slime

Daily grading analyses of the feed, extending over a lengthy period, showed that more than 40 per cent of slime was undesirable, the best results being obtained with from 25 per cent to 35 per cent.

Sometimes the presence of even undecomposed felspar in the gangue appeared to be responsible for a dirty, slimy froth.

At Kyloe the ideal ore seems to be a clean sulphide mineral in a clean quartz gangue. Given a feed approaching this ideal, the flotation machine is capable of marvelous work, as the following figures will show:

Feed % Cu	Concentrate % Cu	Tailing % Cu	Extraction %	Ratio of Concentration
3.8	25.1	0.95	77.0	6 to 1
3.0	23.5	0.85	74.3	5 to 1
3.0	24.2	0.65	81.0	8 to 1
3.15	23.7	0.55	84.5	7 to 1
4.1	25.6	1.05	77.5	6 to 1
4.4	24.6	0.9	82.5	5 to 1

These figures represent consecutive daily runs on good clean

ore, the tailing sample being taken automatically. The machine is considered as an isolated unit. It can be safely asserted that no other concentrating apparatus yet devised can show better or even equally good results on feed which contains particles from 40-mesh up to the very finest slime.

After the flotation machine had been running about 5 months the boxes began to leak, and examination showed that the wood work had been worn away behind the liners. The bottoms and sides had to be renewed, and to save similar trouble in future the $\frac{1}{4}$ -in. steel liners were replaced by $\frac{1}{2}$ -in. cast-iron ones, designed with interlocking joints so as to protect the wood-work entirely from wear. These liners can easily be removed and renewed, and have given no trouble.

The concentrates from the machine are run to bins, where they are settled, and the persistent froth beaten down with a water spray. Certain classes of slimy froth give trouble by building up and overflowing the bins. When the concentrates settle the flow is diverted to another bin, and the water syphoned off.

The sloppy mass is then shovelled into barrows and wheeled to the drying floors, which consist of thick cast-iron plates under which hot gases from a furnace flow. Owing to the high percentages of water the concentrates take some considerable time to dry.

Filtering was attempted, but, owing to the slimy nature of the concentrates, was not a success, and the output was not large enough to warrant the installation of some form of pressure filter.

When dry the concentrates are shovelled from the floors into bins, and bagged for despatch to the smelter.

The following figures show the recovery made by the mill each month, calculated from the assay value and weight of feed and the value and weight of concentrate despatched:

Month	Average Value of Tailing % Cu	Extraction %
1st	1.26	82.7
2nd	1.11	86.1
3rd	1.03	85.4
4th	1.0	87.0
5th	0.95	93.0
6th	0.89	90.5
7th	0.81	85.0
8th	0.95	95.0
9th		

The accepted assay value of each lot of concentrate sent to the smelter is given below:

Lot No.	Accepted Assay Value % Cu	Lot No.	Accepted Assay Value % Cu.
1	23.2	7	25.27
2	26.17	8	25.24
3	25.95	9	25.92
4	25.05	10	26.00
5	25.52	11	25.06
6	25.66	12	25.48

General Results Improved by Flotation

These figures show an increase of over 10 per cent in the recovery and a little under 5 per cent in the grade of the concentrate, over the old system of tailing. The importance of the latter may be gauged from the fact that after deducting freight and smelting charges, with copper at £50 (\$250) a ton, a ton of copper in a 20 per cent concentrate is only worth £28.76 (\$144) to the company while a ton of copper in a 25 per cent concentrate is worth £32.52 (\$162)—an increase of £3.76 (\$18) per ton of copper, representing on a 1000-ton output £3,760 (\$18,800) a year. The average cost per ton milled for the twelve months previous to the installation of the flotation plant was 78½ shillings (\$1.90). The average cost since the installation is 8.21 shillings (\$2.00)—an increase of 4.8 pence (£0.10) per ton milled, or about £370 (\$1,850) on the yearly output.

The grade and tonnage of the ore treated was about the same

for 1911 when tables were used, and 1912 when flotation was adopted, thus affording an excellent opportunity for comparing results under the old and new systems of milling.

Copper is taken £60 (\$300) a ton in both cases.

The figures show that, while the expenditure increased £1,300 (\$6,500) the revenue went up over £4,300 (\$21,500), leaving over £3,000 (\$15,000) to the credit of the flotation plant.

The cost of construction and alterations came to about £1,300 (\$6,500) so that the new machinery has paid for itself more than twice over during the first year of operation.

In addition to this, it is only fair to the flotation plant to draw attention to the fact that disbursements and maintenance on the old units have been larger than they were during 1911, owing to depreciation with age.

Nichols-Hesse Dinner

By the joint action of the executive officials of the American Chemical Society, American Electrochemical Society, American Institute of Chemical Engineers, Chemists' Club, the New York Section of the Society of Chemical Industry and the New York Section of the Verein Deutscher Chemiker, it is proposed to give a complimentary dinner to Dr. W. H. Nichols and Dr. B. C. Hesse as a mark of appreciation of their work in connection with the Eighth International Congress just closed. Professor Edward W. Morley, as the honorary president of the Congress, has been asked to preside at this dinner.

It is proposed to hold the dinner on April 19 at the Chemists' Club, the price to be \$3.00 per plate without wine. Rumford Hall will accommodate hardly much more than 250 guests, and the men who intend to go to this dinner should make early reservations.

Milwaukee Meeting of the American Chemical Society

The forty-seventh annual meeting of the American Chemical Society will be held in Milwaukee, Wis., from March 23 to 28. Headquarters will be at the Hotel Pfister. The meetings will be held at Marquette University. A feature of the convention will be an excursion to Madison, Wis.

Atlantic City Meeting of the American Electrochemical Society

The spring meeting of the American Electrochemical Society will be held in Atlantic City, N. J., from April 3 to 5. (Thursday to Saturday of the first week in April). Headquarters will be at the Hotel Traymore.

A meeting of the board of directors will be held on the evening of Wednesday, April 2. The convention will start on Thursday, April 3, with the annual business meeting which will be followed at once by a meeting devoted to the reading and discussion of papers, to be continued in an afternoon session. For the evening it is intended to arrange a smoker.

Practically the whole day of Friday (April 4) will be devoted to an excursion to various chemical plants in and near Philadelphia where the members and guests will be taken by a special train. After their return in the evening to Atlantic City, dinner will be taken and there will be a lecture with demonstrations by Prof. E. B. Kenrick of the University of Toronto.

The Saturday morning meeting will be devoted to the presidential address by Dr. W. Lash Miller which will be in form of an introduction to the symposium of papers on electroplating and electrodeposition of metals, while another professional session in the afternoon, devoted to a continuation of this symposium and to the reading and discussion of other papers, will conclude the meeting.

The fall meeting will be held in Denver, Colo.

Dr. W. Lash Miller, of the University of Toronto, is the president, and Dr. J. W. Richards, of Lehigh University, South Bethlehem, Pa., is the secretary of the American Electrochemical Society.

Symposium of Papers on Alumina

Four papers on the Production and Uses of Alumina, by J. W. Richards, S. A. Tucker, A. H. Cowles, and L. E. Saunders

A symposium of papers on alumina—its production and its uses—was the feature of a joint meeting of the New York Sections of the American Chemical Society, the American Electrochemical Society, and the Society of Chemical Industry, held on February 7, 1913, at the Chemists' Club in New York City.

The meeting was held under the auspices of the New York Section of the American Electrochemical Society and was called to order by the chairman of the section, Mr. Elmer A. Sperry. In a brief business meeting which followed, this section elected officers for the coming year as follows:

Mr. Lawrence Addicks, superintendent of the Chrome, N. J., works of the U. S. Metals Refining Company, was elected chairman, and Mr. H. B. Coho, of the United Lead Company, New York City, was re-elected secretary-treasurer.

Mr. E. A. Sperry then made some introductory remarks on the subject of alumina. He referred to the presence at the meeting of Mr. Bradley and Mr. Cowles, two prominent pioneers of the metallurgy of aluminium and

sketched the effect which cheaper alumina would have on the industry of aluminium and aluminium alloys. He referred especially to the remarkable properties of duralumin for many purposes in the industries. Samples were exhibited.

Then followed two papers by Dr. Jos. W. Richards, of Lehigh University, and by Prof. Sam. A. Tucker, of Columbia University, on the production of alumina and ammonia by the Serpek process which is attracting so much attention at present.

The Serpek Process

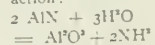
By J. W. Richards

The production of aluminium nitride (AlN) on a large scale is proposed in the process of Ottokar Serpek, followed by the decomposition of the same by water or caustic soda solution, thereby liberating the nitrogen as ammonia and producing alkaline aluminate solution from which pure alumina can be obtained. This rather daring chemical proposition has directed attention to aluminium nitride, the conditions of its formation and its properties.

The existence of aluminium nitride was suspected for some time before it was isolated. A rather indefinite number of years ago, probably about 1890, the attention of the writer was

directed by his father to the fact that when metallic aluminium in a melted condition was skimmed, and the skimmings or dross laid to one side, that on pouring water upon them while still warm these skimmings gave off an odor of ammonia. The explanation which was then figured out for this phenomenon was that the hot aluminium in the skimmings oxidized to alumina, and that the rather high temperature thus produced locally caused the aluminium also to unite with the nitrogen of the air and form the nitride. In other words, that some of the aluminium united first with the oxygen and immediately thereafter some also with the nitrogen of the air. On sprinkling

with water the nitride would react according to the following reaction:

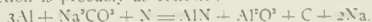


In the light of subsequent investigation and the formation of aluminium nitride in other ways, it appears that the explanation given in these early days was correct and that aluminium nitride is formed directly under such conditions from metallic aluminium and the nitrogen of the air.

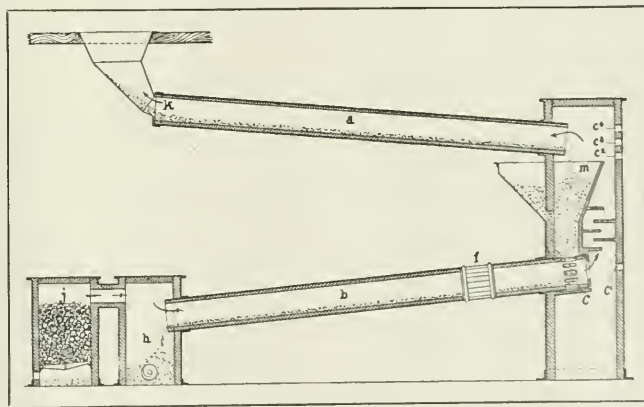
LeVerrier, later, proved the presence of traces of

nitrogen in commercial aluminium by dissolving it in caustic potash and passing the nitrogen evolved into Nessler solution, obtaining from it the ammonia reaction. He thus proved the presence of nitrogen in the metal and suspected it to occur as aluminium nitride dissolved in the excess of metal. In further experiments he treated aluminium with a current of nitrogen gas until it was saturated, and noted that this produced considerable diminution in the tensile strength, elastic limit and elongation of the metal.

Prof. Mallet later obtained aluminium nitride corresponding closely to the formula AlN by heating metallic aluminium in the carbon crucible to a moderate temperature for several hours in contact with dry sodium carbonate. Some alumina is formed, some sodium vaporizes and some carbon is deposited: the reaction is probably as follows:



Aluminium nitride containing 34 per cent of nitrogen and 66 per cent of aluminium was obtained as little yellow crystals and amorphous drops. When calcined in the air it was found to lose nitrogen and form alumina; it was decomposed in moist air, losing its transparency, becoming lighter yellow, evolving ammonia gas, and finally leaving the alumina residue.



Apparatus for carrying out Serpek process (*b*, supply of ground bauxite and exit of gases; *a*, kiln for calcining; *m*, hopper for supply of calcined bauxite to kiln; *b*, kiln for nitride reaction with electric resistance furnace; *h*, chamber for removal of nitride; *j*, gas producer; *c*, chamber for deposition of silica; *c*¹, *c*², *c*³, *c*⁴, air supply inlets for combustion of gases. From Serpek patent 996,032).

Caustic potash solution and melted caustic potash both attacked it actively, disengaging ammonia and forming potassium aluminate.

A later experimenter described the compound as a grey amorphous powder, but otherwise with properties similar to those described.

Coming now to the work of Serpek we find his first proposition to be the production of aluminium nitride from aluminium carbide, which is indeed a possible reaction. In his U. S. patent 867,615 (filed June 19, 1906, issued October 8, 1907) he describes the passing of nitrogen or a gas containing nitrogen over aluminium carbide as an initial material heated to red heat. He describes the volume of the carbide increasing as it is converted into nitride, and that the absorption of nitrogen was increased by diluting the carbide with such materials as carbon or alumina or aluminium chloride: also that traces of hydrochloric acid or sulphur dioxide in the gas currents facilitated the conversion. The claims of this patent are essentially for the method of producing aluminium nitrides by heating aluminium carbide in an atmosphere containing nitrogen to a red heat with or without the diluting substances admixed and with or without traces of acid gas in the nitrogen used.

Attention may be called to the term "aluminium nitrides" in this patent, corresponding to the fact that in the product thus obtained by Serpek he found that part of the nitrogen in the product was driven off by contact with air, and a further part by boiling with water, and a still further part by treatment with caustic alkali solutions. These facts pointed to the non-homogeneity of the product, which thus appeared to consist of a mixture of different nitrides instead of one distinctive chemical compound.

Continuing his work we find that Serpek later concluded that it might be possible to obtain the nitride directly from mixtures of alumina and carbon, which it will be recalled he used as diluent substances in the conversion of the carbide. He was apparently not able at that time to produce nitride directly from these two substances, but found that in the presence of a small amount of metal such as copper or iron, which is capable of forming an alloy with aluminium the formation of nitride from the mixture took place. In his U. S. patent 888,044 (filed May 24, 1907, issued May 19, 1908), he describes mixing alumina with carbon in the proportions necessary for formation of CO gas (Al_2O_3 3C), adding about 5 per cent of either copper or iron or a mixture of the two, and heating the mixture to a red heat in a current of nitrogen gas, at which temperature the partial formation of an alloy of aluminium produced by reduction, with the copper or iron would be initiated. The idea seems to have been to produce a mixture on the point of reaction from its constituents and then to super-add the influence of the current of nitrogen gas. Traces of HCl or SO_2 gases are said to facilitate this reaction also, and the product is described as almost pure aluminium nitride. The reaction is described as developing a high temperature as soon as it is initiated. The claims are for producing aluminium nitrides by heating the mixture of alumina with carbon and a metal capable of forming an alloy with aluminium, to red heat in an atmosphere containing nitrogen, with or without the presence of small amounts of acid gases.

It is evident that in this method the conversion of alumina into aluminium nitride was aimed at, with the assistance of small amounts of metals as catalytic or assisting agents. Whatever may have been the results of the previously described process it finally led the investigator in the end, to the method described in Serpek U. S. patent 987,408 (filed December 15, 1909, issued March 21, 1911) in which aluminium nitride is produced directly from Al_2O_3 and carbon in the presence of nitrogen, at a much higher temperature than previously recommended. This patent appears to be the result of considerable investigation on the conditions of this direct production and specifies very exactly the temperatures at which the direct conversion takes place. The patentee speaks of the general supposition that extremely high temperatures would be required for such direct conversion, such as the highest temperatures

of the electric furnace, but explains that by careful investigation he had found that if the temperature is closely watched some combination of nitrogen with aluminium can be observed even at 1100 deg. C.; that at 1500 deg. nitrogen is absorbed fairly rapidly, at 1700 deg. energetically, and that at 1800 to 1850 deg. the reaction may be almost designated as violent producing almost chemically pure nitride. Further, Serpek found that temperatures higher than this gave a smaller yield, and that somewhere above 2000 deg. the production practically ceased, the decomposition temperature of the nitride appearing to be about 2120 deg. C. This brilliant piece of experimental work really laid the foundation for the Serpek process as it now exists. The inventor further discovered that impure alumina, such as bauxite ore, was converted at somewhat lower temperatures than the pure material, evidently owing to the catalytic effect of the impurities present.

The claims of this patent are for producing aluminium nitride by heating aluminous compounds and carbon, or more specifically alumina and carbon, in an atmosphere containing nitrogen to temperatures not exceeding 2000 deg. C., and preferably about 1800 deg.

Reference may very properly be made at this point to the experimental investigation and confirmation of these results by Prof. S. A. Tucker and Henry L. Read given before the Eighth International Congress of Applied Chemistry, published in its Transactions and in volume 22 of the Transactions of the American Electrochemical Society. This investigation, carried on in the electrochemical laboratory of Columbia University under carefully regulated conditions showed traces of nitrogen absorbed at temperatures of 1000 to 1100 deg. and as much as 30 per cent of nitrogen in selected portions of the product made between 1800 and 1900 deg. The product made at higher temperatures contained less nitrogen and was either sintered or fused. Impure alumina, such as bauxite, was found to be converted more easily than pure alumina, and the definite statements of the last Serpek patent were confirmed and verified.

Serpek's U. S. patent 996,032 (filed June 21, 1910, issued June 20, 1911), contained a detailed description of the apparatus intended for the commercial manufacture of the nitride from bauxite or other aluminium ores. The plant consists in two superposed rotating cylindrical kilns, similar in construction to a cement clinkering kiln. The kilns are inclined in opposite directions, so that the material being preheated in the upper kiln and passing from right to left for instance, falls into the opening of the lower kiln for treatment at higher temperature where it passes from left to right. Bauxite alone is passed through the upper kiln, and there calcined; then mixed with the necessary carbon in its passage from the upper to the lower kiln and the mixture treated by nitrogen at a high temperature in the lower kiln, which is provided with a detachable electric resistance furnace about midway of this length and which is intended to subject the charge therein to the reacting temperature about 1800 to 1900 deg. C. The material is discharged from the lower end of this kiln into an air-tight receiver. A large gas producer of the ordinary type furnishes producer gas (approximately $\frac{1}{4}$ CO, $\frac{3}{4}$ nitrogen) to the lower end of the lower kiln. This gas entering at a temperature of about 400 deg. becomes highly heated as it passes through the kiln in a direction contrary to the descending charge, and at the electrically heated zone at a temperature of 1800 to 1900 reacts upon the mixture forming nitride. After leaving the high temperature zone the gas enriched in CO coming from the reaction preheats the descending charge, issues from the upper end of the kiln into a vertical closed chamber and passes through it to the opening of the upper kiln where it meets a blast of air which burns it for the purpose of heating the contents of the upper or calcining kiln.

The apparatus as thus described achieves a methodical heating of the reacting mixture, utilizes the CO gas made in the producer and also that evolved by the reaction in the furnace; aims at an intimate mixture of the charge by the rotation of the cylinders and intimate contact of the nitrogen with the charge. The inventor states that the silica impurity of the

charge is mostly volatilized (probably as reduced silicon) from the charge at the reacting temperature and is thus carried out in the gases; and that the apparatus may be used for producing other nitrides made by analogous reactions.

The claims cover an apparatus comprising the features of a revolving inclined upper cylinder discharging into an inclined lower cylinder revolving in the opposite direction, through a vertical chamber in which the gas from the lower cylinder may be burned combined with an electric detachable electric resistance heating furnace within the lower cylinder. One claim is for the single feature of a rotary calcining cylinder carrying a detachable electric resistance furnace.

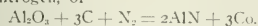
Undoubtedly a large amount of excellent investigation has been put upon this process, and it certainly deserves commercial success. The method is being tested in France upon a large experimental scale.

The Relation of the Production of Alumina to the Fixation of Nitrogen

By Samuel A. Tucker

The connection of nitrogen fixation and the production of alumina is to be found in the Serpek process and probably in this process only. The Serpek process is primarily one for the fixation of nitrogen, but necessarily involves the production of alumina as a by-product and this by-product is all important to the success of the process commercially.

Briefly the Serpek process takes alumina and carbon in the proportions to form aluminium nitride in a strongly heated atmosphere of nitrogen, or



The range of temperature necessary for this reaction is from 1600 deg. to 2000 deg. C. according to Serpek. The product obtained contains about 30 per cent nitrogen in the form of aluminium nitride, and is thus considerably richer in fixed nitrogen than calcium cyanamide which rarely runs over 20 per cent.

This reaction appeared so important to me that I thought it was worth while to investigate it and in conjunction with Mr. H. L. Read, the process was tried on a small scale. The results¹ of this work have been read at the recent Congress of Applied Chemistry, and it will be unnecessary to repeat them here, excepting that we found that the reaction worked satisfactorily on a small scale, the product contained 30 per cent of nitrogen, and that bauxite was the best material with which to supply the aluminum. The temperature is important. Below 1600 deg. practically no nitrogen is fixed, while above 2000 deg. C. decomposition is likely to occur. I would like to know more as to the influence of temperature, and I had expected to conduct some experiments on this point so that I might give you this information now, but an accident to the electric furnace prevented it.

The commercial product must be nearly pure aluminium nitride, and at first sight seems to consist of two forms, the blue and the yellow. Professor Luquer has examined these specimens microscopically and while his examination has not yet been completed, he tells me that the internal structure is the same, both are strong double refracting and show uniaxial structure. With our present information, it would be difficult to figure the cost of production of this material, and even the power necessary for its manufacture is probably not generally known.

The question of the necessary time to convert the charge is important and in our experiments we found that it had a great influence on the nitrogen content of the product. It, therefore, seems necessary that the mixture should be heated to a high temperature for a considerable time.

This is different from the Frank process for cyanamide in that the temperature for the actual nitrogen fixation is relatively low. Granting that the process can be worked out on a satisfactory cost basis, we see that the production of alumina is an all important consideration. By proper treatment of the

Serpek product there seems to be no difficulty in obtaining the nitrogen as ammonia and the aluminium as alumina, and the latter should be practically pure Al_2O_3 .

It might be supposed that the alumina so produced could be used over again in the process, but aside from its value in connection with the aluminium industry, pure alumina is not advantageous as a raw material, as it requires a higher temperature than bauxite. The process is, therefore, necessarily associated with the production of aluminium. If we might suppose that the Serpek process were to be conducted on the same scale as the Frank process for cyanamide manufacture it is evident that it would involve the production of a large amount of alumina. From figures recently published regarding the Frank process, we find that the capacity output of all the plants covering various parts of the world for 1912 is roughly 200,000 tons of cyanamide. If we figure this as containing 18 per cent. nitrogen, it involves the fixation of 36,000 tons of nitrogen. With the Serpek product at 30 per cent. nitrogen content, it would require the production of 120,000 tons to equal this, and after decomposing to obtain the ammonia as follows:



we obtain 140,200 tons of alumina. This quantity of alumina should give 79,704 tons of metallic aluminium.

In the year 1911 we find that the capacity of all the plants producing aluminium was 62,100 tons, but that the estimated production actually was about 45,000 tons. We thus have raw material for the production of 34,000 tons of alumina in excess of present requirements. Of course, the process will not be conducted on such a scale for some time to come, and by that time there might well be a market for the alumina.

There appears to be no danger from the lack of bauxite, as I find that in this country 155,618 tons were produced in 1911 and 106,000 tons in France for the year 1910.

As the Serpek process would necessarily come into competition with the other means of fixing nitrogen, and particularly with the cyanamide process, it might be well to consider for a moment a few points in connection with the latter.

Briefly the Frank-Caro process for making cyanamide is to subject finely ground calcium carbide to the action of pure nitrogen at a temperature of from 1000 to 1200 deg. C. The energy necessary at this absorption stage is probably very small, as heat is evolved by the reaction:



and the expenditure of energy is chiefly in the manufacture of the calcium carbide. Various conditions have to be observed in this absorption part of the process. The nitrogen must be practically free of oxygen and the temperature must not rise to the dissociation point of the cyanamide, which is probably about 1370 deg. C. The product so obtained contains anywhere from 15 to 22 per cent of nitrogen. The nitrogen is obtained by passing air over copper to take up the oxygen, or by fractionally distilling liquid air. Both systems seem to be in use, but with the latter some treatment of the resulting nitrogen with copper would probably have to be employed as nitrogen obtained from liquid air contains upwards of 8 per cent oxygen.

The process seems fairly complicated, but possesses the advantage over the high-tension arc systems of fixation in that a better return of nitrogen is obtained for the power used. In the Frank process the chief expenditure for power is in the formation of calcium carbide. The average of several plants making carbide is 5.2 kilograms of 80 per cent carbide per kilowatt-day of 24 hours. This gives 4.16 kilograms of pure calcium carbide, which corresponds to 173 grams per kilowatt-hour. Now if this is converted to cyanamide having a 20 per cent nitrogen content, we get 43.2 grams N per kilowatt-hour.

The Birkeland-Eyde process obtains about 57.1 grams nitric acid per kilowatt-hour and the nitrogen content is 12.7 grams. We thus have about three and one-half times the nitrogen fixed by the Frank process as cyanamide as we do for the Birkeland-Eyde process as nitric acid, and this is probably the reason for its being established on such a large scale.

¹See this Journal, volume X, p. 745 (1912).

As to the relative value of the fixed nitrogen fertilizers, we have no very precise information. Taking the various products adapted to this purpose in the different processes, we get calcium nitrate by the Birkeland-Eyde, calcium cyanamide by the Frank, and lastly ammonia by the Serpek process. These must all be compared with Chili sodium nitrate and the general opinion seems to be that the latter is the most valuable.

Field experiments have been conducted with cyanamide and calcium nitrate, but precise information is lacking. It would seem, however, from what has so far been done that we should put them in the following order: Chili saltpeter, calcium nitrate, cyanamide, ammonia.

The action of cyanamide is quite different from the others; it supplies a large amount of calcium which may or may not be an advantage. Its slow solubility is, however, advantageous.

Ammonia in the form of sulphate is pretty well understood in its relation to fertilization and for that reason the Serpek product should have a ready market in this connection.

* * *

The two papers by Professor Richards and Professor Tucker were discussed together. Some of the points touched upon in the discussion were the possibility of an intermediate reaction preceding the formation of the aluminium nitride, and the way how the impurities, especially silica, are removed in the process. Attention was called to the fact that French bauxite is in the average low in silicon and high in iron and American bauxite high in silica and low in iron, and that this might have some effect on the operation of the process with American bauxite.

After the conclusion of this discussion, in which some ten speakers participated, Mr. Alfred H. Cowles, of Sewaren, N. J., presented his paper on his own new alumina process, the fundamental points of which were already described by him in his paper before the International Congress of Applied Chemistry in September, 1912 (our Vol. X, page 659). Mr. Cowles' present paper contained, however, much that was new. We give it herewith practically in full:

Cheaper Alumina and Aluminium from Mineral Silicates

Through Reactions with Alkali Chlorides, Lime and Steam, and Between Florida Phosphate Rock, Potash Feldspars and Acids with the Production of Cheap Hydrochloric Acid, Potash Salts, Phosphate Fertilizers and the Production of Material for the Manufacture of Glass or Hydraulic Cement

By Alfred H. Cowles

Let two briquets be formed, one of kaolin or clay containing 19 parts of anhydrous pure clay and 23.4 parts of salt, and the other briquet containing the same relative proportion of salt and dry kaolin as the first, and fifteen parts of charcoal per one hundred parts of the mixture be present in the second briquet. If now these two briquets, each of the same size and form, be kept heated to a temperature equal to or just above the vaporizing temperature of salt, and if vapor of water in an oxidizing atmosphere surrounds them, it will be found that each briquet begins to be converted over its surface into a sodic-silico-aluminate which would have the formula of $(\text{Na}_2\text{O})_2(\text{SiO}_2)_2\text{Al}_2\text{O}_3$ if the clay and salt be pure. The vibrating gaseous molecules of watery vapor and oxygen bring about this change over all exposed surfaces of the two briquets. This difference in the conversion will, however, be found. The briquet containing no charcoal will require nine times as much time for the conversion to penetrate to its center between walls, as the briquet containing charcoal.

This result is not self-apparent as the reaction requires an oxidizing atmosphere to oxidize the sodium produced before it volatilizes out of the briquet. That this result would occur, was based upon hypothetical reasoning on my part and its demonstration as a truth was brought about by experiments per-

formed in the laboratory of The Electric Smelting & Aluminum Company at Lockport, New York. By coincidence Dr. Adolf Kayser was my assistant in performing the experiments.

Dr. Kayser in the early '90's, attempted to develop this reaction commercially using briquets containing no charcoal in large kilns of a down-draft nature. The work had proven a failure as had also the work of William Gossage in 1862, and that of Gruneberg and Vorster about 1874-76. During these experiments Dr. Kayser was of the opinion that they could not succeed because, as he thought, it would produce reducing conditions. The experiments, however, proved that the conversion penetrated the briquets nine times faster with charcoal in them, than was the case when briquets contained no charcoal. This result taught that it was necessary to expose large surfaces or porous masses to bring about a rapid conversion. This could be effected in more than one way, but the problem of cooling the gases and simplifying the elimination of dust led to the adoption of a tunnel furnace of the Gröndal type, with modifications to render it suitable for the process.

The fact that the reaction could be made to proceed rapidly opened up an entirely new vista, one of assured success to the process. It meant that the column of fuel gases and nitrogen necessary to pass through the charged furnace would become so far reduced as to permit of efficient commercial condensation of the hydrochloric acid evolved. It meant the employment of cheap fuel, sawdust or charcoal made from sawdust with possible by-products from the same, and far greater capacity of output with the same cost of plant. It meant that the process could be applied to other aluminous materials, such as potash feldspar mixed with chloride of potash or salt, and securing the chemically equivalent reaction.

Dr. Kayser had evolved a method of opening or rendering leachable alkali-silico-aluminate containing sufficient alkali, which means nearly one molecule more than that which occurs in feldspar. This he did by heating to a sintering temperature two molecular weights of lime to each molecular weight of silica in said compound, after which, the product may be leached and the alkali aluminate very effectively removed therefrom. Our company acquired this patent from him.

The process is easily understood. In the manner in which we are attempting to perfect it at Sewaren, N. J., in our present work, briquets are formed by the same clay-working machinery that produces what is known as hollow-ware or conduit bricks. When the clay, salt, and sawdust are mixed together and water is added, this mixture acts almost the same as clay as to plasticity in passing through the briquetting machinery, but unlike the clay brick, it does not shrink or expand from the period of its first formation during drying and until it is formed into a finished product containing soda, silica, and alumina.

These briquets are loaded onto flat cars, the surface of each car as now used being 12 ft. long and 5½ ft. wide, holding fifty-five bricks. (See Fig. 3, p. 661 of Vol. X of this journal.) These loaded cars are passed first through a drying furnace where the free water is driven off, and then by a transfer track, into a vestibule connected with the long kiln. (See Fig. 4, p. 661, Vol. X.) By a hydraulic pusher, after the door of the furnace is opened, and the door of the vestibule is closed, the loaded car is gradually pushed into the charging end of the furnace, and a train of fourteen cars in front of it are also gradually pushed forward towards the discharging end of the furnace, while a finished carload is coming out at that end.

The cars are covered with firebrick, the latter of a nature to protect them from being materially attacked by salt vapors. A sand seal extends the whole length of the furnace, and pure air, by a blower, is blown under the train of cars in such a manner as to prevent any leakage of hydrochloric acid fumes downward. In operation, the temperature of all exposed iron parts of the furnace are kept above the boiling point of mixed acid and water, and as is well known, at this temperature vapor of hydrochloric acid does not attack iron.

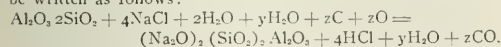
Knowing the composition of the clay and the percentage of salt in the briquets steam is blown into the discharge end of

the furnace, and the rate by pounds weight of steam per minute is read by an indicating steam-flow meter. Suppose this rate is adjusted to the rate of feed of the material on the cars to effect the decomposition of salt with the formation of, say, 33 per cent hydrochloric acid in water. In this case there is required in the chemical reaction only about one part of water to thirteen parts to produce the 33 per cent acid, hence it will be noted that the quantity of steam blown in to form such acid is thirteen times greater than that which takes place in the reaction. The amount of steam, if the heated zone in the furnace be long enough, can be so reduced as to still permit much water being admitted to the coke towers to complete the condensation of all the acid.

Above, and on the end of the furnace where the cars are pushed in, is a large exhaust fan which draws all the gases from the furnace and forces them onward through a condensing system where the hydrochloric acid is condensed. The furnace is 184 ft. long, or ignoring the vestibule is 170 ft. long. Beyond the zone of high temperature towards the fan there is a distance of 79 ft. where the briquet-loaded cars act as a scrubber to remove any sublimed salt and other dust from the gases at the same time that the incoming briquets are being heated by the hot gases passing over them.

In addition to this we are constructing a dust-settling and cooling chamber in the iron pipe line between the furnace and the acid condensing system.

The reaction that takes place in the furnace will vary according to the composition of the aluminous material and the amount of alkali chloride mixed therewith, but typically it may be written as follows:



The gaseous products in the above are carried into the condensing system after cooling. The briquets maintain their original form, and before the cars issue from the discharge end of the furnace they are cooled down by the inflow of the measured quantity of steam and air that is sucked in. This steam and air become preheated by the hot bricks before it reaches the combustion zone of the furnace, and the finished bricks are cool enough for men to handle them with gloves as they discharge from the furnace.

In our work at Sewaren we have discovered that our acid condensing apparatus with air cooling has only about one-third the capacity that it should have. It is at the present time being enlarged, and tourells submerged in water being added to it into which about 18 Beaumé acid will be run against the flow of the gases before the latter reach small coke towers. The escape of acid fumes has been so great that we have only been able to make four experimental runs of about thirty hours' duration each with the large furnace, but the results secured have taught us the necessary changes to be made in the acid condensation system and the necessary changes to get a full conversion in the furnace. The whole product of the last two runs has been analyzed, but not of the first two. The furnace was operated with only one set of oil burners and a car of briquets fed into the furnace during every forty minutes.

Before starting to build our plant at Sewaren briquets had been made containing as high as 33 per cent of sodic oxide and were white and free from iron.

We found by the analysis that the Sewaren briquets were but little more than half converted, hence it was rather against my desire to read this paper at this time. The briquets still contain a large amount of salt. The furnace is provided with another set of oil burners to meet this contingency, but we have not yet used them.

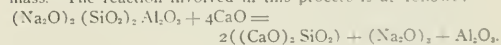
After completing the enlargement of our acid condensing system we contemplate using both sets of oil burners and adding a trifle more sawdust to the bricks, in which case there is no doubt but that we will produce the same results as in our laboratory work, with one exception. In the laboratory work the furnace was a deep, circular furnace filled with briquets and an oil flame and vapor of water were passed downward through the briquets. Under these conditions the iron as an

impurity in the clay was carried downward as chloride of iron and small portions of it passed off with the acid and other portions condensed in the bottom of the furnace and in the iron pipes through which were drawn the gaseous product from the furnace from under a grate at its bottom.

As the bricks were secured white, it was concluded that we would secure the same result in our large work, but this conclusion, as given in a paper read before the Eighth International Congress of Applied Chemistry (this journal, Vol. X, p. 659, October, 1912) has, by practice, proven erroneous. As the surfaces of the cars are cooler than the briquets, we find a portion of the iron as either ferric or magnetic oxide on the car surfaces and a part still remaining mainly on the surface of the lower portions of the briquets. Thus it is the iron oxide remains on the cars and briquets and does not pass out with the gases to vitiate a part of the acid. The acid produced thus far only shows the presence of sulphuric acid by the most delicate test. It is free from arsenic and has but traces of free chlorine, iron chloride, sulphuric acid and ammonium chloride, and with our final settling dust chamber completed, it will be practically free from salt and aluminium chloride.

LIME OPENING PROCESS

In order to secure from the sodic-silico-aluminate material of the briquets the sodic aluminate contained therein we intend to put down a kiln of the rotary type to sinter this material, which is easily friable, after mixing with it lime in such proportions as to form di-calcium-silicate from the silica in their composition. This compound is insoluble, while the sodium aluminate is soluble. The material as thus treated is not fused, but sintered in such condition as to form an easily leachable mass. The reaction involved in this process is as follows:



Limestone, calcite or marl in their chemical equivalents may be used instead of anhydrous lime.

The product thus secured will be leached by ordinary methods.

In the foregoing reaction any amount over two molecular weights of calcium oxide to each molecular weight of silica leads to the formation of insoluble calcic aluminate, and less than two molecular weights is not sufficient and leaves undecomposed a portion of the insoluble sodic-silico-aluminate. Under either of these contingencies, marked loss in the recovery of alumina may occur. Experimentally up to 90 per cent of the sodic aluminate has been leached from the above clinker, while there was left behind a mass of di-calcium silicate.

PRODUCTION OF HYDRAULIC CEMENT

According to Newbury's formula hydraulic cement is tri-calcium-silicate with up to 8 per cent of aluminium and iron oxides in a mixed condition being permissible in the formula. The above-mentioned di-calcium-silicate product is hydrated. If it now be ground and then heated with one added molecular weight of lime it forms a product that is in composition the same as that of hydraulic cement, but unless the material used is very pure, experience at Sewaren discloses that it will give a light gray cement and not a white, as the writer believed when he read a paper before the Eighth International Congress of Applied Chemistry.

Inadvertently there was printed in the paper read at that time the following statement:

"When briquets are made containing 32 per cent of sodic oxide the output of 30 per cent hydrochloric acid should be about 1½ tons of acid to 1 ton of salt and clay in the charge"

The statement should have read: To 1 ton of salt in the charge, about 1.75 tons of 33 per cent acid.

USE OF DI-CALCIUM-SILICATE AS A MATERIAL FOR GLASS MAKING.

If in the foregoing lime sintering process should there intentionally be left in the insoluble di-calcium-silicate after leaching 15 or 20 per cent of its alumina and alkali contents, there remains a product that is excellent for the manufacture of window and bottle glasses. Its use would call for the addition of silica and more alkali to its composition. Glass can stand a considerable percentage of ferric oxide and alumina in its manufacture.

OXALIC ACID

Nearly all the ingredients needed to produce oxalic acid, excepting sawdust, come from the products of these processes. The carbonic oxide from the acid towers can be scrubbed to free it from the last traces of hydrochloric acid and then passed over or through sawdust mixed with caustic alkali heated to the proper temperature producing an oxalate of the alkali, which treated with calcium chloride gives calcium oxalate and with sulphuric acid gives oxalic acid with calcium sulphate precipitated, frees the oxalic acid. This acid is most largely manufactured in Europe, but has a great consumption in America in our laundries.

USE OF HYDROCHLORIC ACID IN PRODUCING PHOSPHATE FERTILIZERS

The question may well be asked what use can be made of the large amount of hydrochloric acid that this process will produce should it supply as much alumina as will be needed for the production of the world's future supply of the metal aluminium. The author has pending a number of United States and foreign patents which may place hydrochloric acid in competition with sulphuric acid as now used in the manufacture of soluble forms of calcium phosphate mixed with calcium sulphate so largely used as a fertilizer. A pure calcium phosphate rock is anhydrous tri-calcium-phosphate. With it there is almost always associated considerable calcium carbonate, silica, alumina, and other impurities.

If this rock be ground to an even degree of fineness and then thoroughly digested with weak hydrochloric acid, the quantity of acid employed being theoretically just sufficient to dissolve all the free calcium carbonate, ferric oxide and one molecular weight of calcium oxide contained in the tri-calcium phosphate, the insoluble residue that is left is di-calcium phosphate. The soluble products, calcium, magnesium, and iron chloride, will exist in solution. By filtering and passing water through this mass, the soluble chlorides formed can easily be removed and either run into the sea or submitted to other possible uses.

The insoluble di-calcium-phosphate remains in a non-hydroscopic pulverent condition that can readily be applied to the soil by present methods of distribution. This is the product that all soluble phosphates revert to when they are applied to the soil as fertilizers. It has, however, one advantage over the soluble phosphates produced with sulphuric acid, namely, it is not loaded down with a large burden of calcium phosphate upon which transportation charges must be paid. If calcium sulphate is desired for fertilizer use it can more cheaply be produced from native gypsum rock. If the digestion is not thorough some soluble phosphates pass into the rich solution of chlorides. These phosphates can be precipitated by the use of a proper amount of calcium carbonate.

In Idaho, Utah, Montana, and Wyoming the United States Geological Survey have in recent years discovered and examined the largest deposits of phosphate-bearing rock ever known to the world. In this general region these rocks that have now been withdrawn from location covered an area as of Sept. 1, 1912, of nearly 2,212,000 acres.

The process as thus far described ends in securing the alumina as a sodic aluminate solution which can be completely purified from silica, titanitic acid, and the iron oxide by a very simple means of precipitation. The final precipitation of the alumina in its purest form is brought about by Dr. K. H. Bayer's process, patented about 1894. It involves a very peculiar and interesting phenomenon. When previously precipitated alumina is added to the sodic aluminate solution even without shaking; if one observes a beaker containing this solution and the precipitate he will note the precipitate to begin to grow almost like a mushroom and in a few days' time nearly all of the alumina precipitates itself leaving a liquid containing nothing but caustic soda and a comparatively small amount of alumina. By an added step, which on account of patent matters I ought not to discuss, the last traces of alumina from the solution may be recovered and the solution left as strong solution of caustic soda needing slight concentration to prepare it for the market.

This general process opens the way to secure alumina

cheaply or as a by-product in conjunction with these other products of great consumption in the industrial world. There is required 1.9 lb. of alumina to produce 1 lb. of aluminium, there being no appreciable loss in the present method of manufacture of the metal over the theoretical amount. This weight of alumina costs from $6\frac{1}{2}$ to $7\frac{1}{2}$ cents according to the latest data I have regarding the same and is the heaviest cost in producing the aluminium. By a general introduction of this process we may hope to see the cost of production of aluminium reduced 4 or 5 cents per pound, and when it comes to a general electrification of our world's railroads and a greater use of electric transmission lines and trolley wires, we will find aluminium a much more serious competitor of copper than at present, and eventually it will almost entirely supplant copper except in motors and generators.

POTASH ALUM FROM POTASH FELDSPAR AND ROCK PHOSPHATE

We all know the great need of potash salts as plant food and the scarcity of their supply, also the abundance of potash feldspar rocks containing great quantities of potash and the abundance of rock phosphate. Several years ago I was experimenting with the mineral wavelite, which is a hydrated phosphate of aluminium. Knowing that calcium sulphate or burned gypsum was slightly soluble in water, and that aluminium sulphate was very soluble in water, I reasoned that if proper proportions of aluminium phosphate and calcium phosphate were ground together and a minute amount of sulphuric acid were added to start the reaction, that the two bases should trade their acids if enough acid were present to form an insoluble form of calcium phosphate.

On trying the experiment, I found this reversal occurred and that aluminium sulphate could abundantly be leached away from the mixture. Further, knowing that di-calcium silicate was quite an insoluble body and if sulphuric acid were added to it the calcium sulphate formed would be but slightly soluble, it occurred to me that were I to take potash feldspar, for instance, orthoclase, whose formula may be written $K_2O \cdot Al_2O_3 \cdot 6SiO_2$, and grind it with just sufficient rock phosphate so that there would be present in the mixture two molecular weights of lime to each molecular weight of silica in the feldspar, or other potash silicates, such as leucite, and that if I added just the right proportion of sulphuric acid to this mixture I would have a chemical system which under conditions of solubility would degenerate to the form of insoluble di-calcium silicate and soluble aluminium sulphate, potash sulphate and ortho-phosphoric acid, H_3PO_4 .

Upon having heated together the above mixture to merely a sintering temperature of about 1000 deg. C., and leaching with sulphuric acid, my first experiments gave almost 100 per cent.

For the purpose of this lecture I have had the work repeated. In figuring the amount of sulphuric acid to use, my assistant has taken a little less acid than was used in the former work and the result of the leaching has given a product containing all the potash and all the alumina from the feldspar, but only part of the phosphoric acid. Duplicate results have been secured substituting hydrochloric acid for sulphuric acid. The liquor in these two cases will give us either potash alum or a double chloride of aluminium and potassium associated with all the phosphoric acid contained in the rock phosphate employed. By this process the di-calcium-silicate or a slightly lower form of calcium silicate would be available for use in either the manufacture of cement or the manufacture of glass. Feldspar can be secured very free from iron, so also can rock phosphate. The iron, however, follows the other products into solution. From either potash, alum or the double chloride of aluminium and potassium the chances of securing pure alumina are very favorable. Upon heating to a dull red heat, aluminium sulphate breaks down into alumina with liberations of sulphuric acid (SO_3) and the potassium sulphate can then be leached away. In the case of a double chloride of aluminium and potassium at a very low heat in the presence of vapor of water, hydrochloric acid is given off with the formation of alumina, the chloride of potassium remaining as such, which can be leached away from the alumina.

To secure pure alumina from the above solution of chloride or sulphate salts with phosphoric acid is promising. The potash alum, however, cannot be separated by crystallization. By precipitation with sodic or potassic aluminate, if too much of either of these salts be not used, pure phosphate of aluminium may be secured. This audience will appreciate other possibilities.

POTASH PHOSPHATE FERTILIZER

In the foregoing I have described the sintering at a comparatively low temperature of rock phosphate and potash silicates. I wish to call attention to the fact that this mixture in



LOCKPORT PLANT OF ELECTRIC SMELTING AND ALUMINIUM COMPANY

some cases, and sinter in this instance with a proper amount of sulphuric acid added to it, or hydrochloric acid, if the hydroscopic properties be not detrimental in the latter case, is of greater fertilizer value than rock phosphate when used alone. By this new process the silica has taken place of much acid that would be required, and the product is not loaded down with calcium sulphate, but, instead, is loaded with potassium sulphate and the lime and silica is in such form as to be sufficiently soluble to furnish silica to plant food, and, further, the soluble lime is in a condition to act on potash minerals in the soil, displacing potash therefrom, thus rendering more potash available for plant food.

The soap industry is founded on the use of alkalis, and I have pointed out to you the possibilities of the fertilizer industry, the glass industry, and the cement industry as all being possible future contributors to the cheap production of aluminium. For the production of cheaper aluminium must come with the expansion of electricity to furnish power for the electrification of our railroads and other industrial uses.

Mr. Cowles then showed in the form of lantern slides the views of his Seward plant, which were given in our Vol. X, p. 650 (October, 1912), and also a view of his Lockport plant, which is herewith reproduced.

Mr. Cowles' paper was discussed at some length by various speakers.

The last paper on the program related to new uses of alumina, the author being Mr. L. F. Saunders, of the Norton Company, of Niagara Falls, N. Y.

Uses of Alumina as an Abrasive and Refractory

By L. E. Saunders

Most people undoubtedly consider aluminium metal manufacture as the only significant factor in the consumption of bauxite and other aluminous ores, and the refined product alumina.

I am therefore going to call attention to another industry which places dependence on aluminous ores, the abrasive business. Unless something calamitous happens to all of us during the next few months, this year will see a consumption

of aluminous ores in the world corresponding to some 10,000 tons of aluminium metal in the manufacture of grinding wheels. It may not be realized to what extent abrasives enter in the equipment of the modern machine shop. Hardly an essential in our daily walks of life but is dependent in this new machine tool for its manufacture. The breakfast food with which we start the day, the stove on which it is prepared, the morning paper which gives it flavor, the street car, railroad train or motor car which takes us to business, the marble panels in our palatial offices, the typewriters which write our letters, the pearl buttons on our gloves, the leather gloves themselves, nearly all our activities, and finally, the polished granite columns at our graves would not be here in their present perfection were it not for the artificial abrasive which either makes an integral part or a machine to accomplish the end in view.

Broadly speaking, grinding is done in two classes, first, metals of high tensile strength, such as steel of all classes, malleable iron, etc., and, second, metals of low tensile strength, such as cast iron, brass, aluminium, etc. As about 75 per cent of the world's pig iron is converted into steel, and finished and wearing parts are largely made of this material, it may be seen that the first class covers the wider field. Aluminous artificial abrasives are used almost exclusively for this purpose and this fact offers a reason for the figures given above. The change from natural to artificial aluminous abrasives is world-wide, the use of emery and corundum showing a steady decrease. The reason for this lies in the lack of uniformity and grinding efficiency of the natural minerals and the possibility of exact duplication of quality in the electric furnace product. It is also possible to give such products varying characteristics for various uses, the accomplishment of which depends on the skill of the manufacturer and knowledge of the laws governing its properties. Such materials are therefore in use varying from an alumina content of 70 per cent to one of over 99 per cent.

As an adjunct to and an offshoot of this industry the aluminous refractory is coming into its own. (This journal, Vol. IX, p. 257, 1911.) While these articles are of little account as consumers of alumina, they are of considerable interest and value to the laboratory worker. The demands are increasing steadily as the users become better acquainted with the properties. One of the problems we have to contend with is to persuade the chemist that we have not attempted to furnish a sovereign remedy for all laboratory ills. The products may be varied in their properties, and an advantage in one direction is usually accomplished by the sacrifice of some other quality. Numerous published criticisms have appeared, most of which would have been quite uncalled for if the problems had been submitted first to the manufacturer so that a selection of the proper grade could have been made—neither in the abrasive or refractory field is alundum a panacea. When we are told the job, it is not very often that we fail.

In closing, let me say that new processes for purifying aluminous materials are still of great interest to us. A chance of securing cheaper raw materials looks good to us and from the user's standpoint we wish each experimenter success in his own work.

Industrial fellowships are established at the Universities of Kansas and Pittsburgh, under the direction of Dr. Robert K. Duncan, for the purpose of promoting the increase of scientific knowledge in chemical industries. The scheme involves financial assistance from any industrial concern with a problem to investigate, the co-operation of the university and the personal work of one or more fellows. The results, if any, are shared by the company, the university and the fellow. According to a recent announcement by Dr. Duncan before the Franklin Institute, the scheme is working well and has developed no weaknesses during the five years of its trial. Eleven fellowships have been completed at the University of Kansas and nine are still in force. At the University of Pittsburgh there are sixteen in force. The subjects of investigation have a wide range in industrial chemistry and metallurgy.

Titanium as Used in Steel Making

By E. F. Lake

During the last decade a great deal of progress has been made in the iron and steel industry and its products greatly improved by the perfection of new methods and the discovery and adoption of different elements for use as purifying or alloying material. One of the last elements in this category, but by no means the least, is titanium, which was first reduced to a fused condition by Moissan in his electric furnace. Although he separated it from all other elements he could not get it quite free from carbon. Between that time and the present, much experimental work has been done on metallic titanium but only very recently has it been made to better the various metal products when used in their manufacture. It was not until 1907 that the first heat of titanium steel was teemed and rolled in America and yet more tons of it are now made than of all the other alloy steels put together.

Metallic titanium is hard and brittle and has a dark grey color with a decided metallic luster; it being not unlike silicon and also resembling iron in appearance. It melts at about 2300 deg. C., and its atomic weight is 48.1. In nature it is never found native. It either occurs as titanium dioxide, (TiO_2), in the three distinct crystallized varieties of rutile, brookite, and anatase; in combination with iron, as titaniferous iron, containing ferrous titanate, (FeTiO_3), which is of the most interest to steel makers and sometimes contains 40 per cent of titanium, and in a number of iron ores and minerals. It is found in combination with the protoxide of iron, mixed with more or less of the peroxide of the same metal. It has been found in many minerals and rocks, as well as in clays and soils resulting from their decomposition. Thus it appears to be very widely distributed in the earth's crust but no one locality holds any large quantity. It has also been detected in meteorites and in the sun.

Titanium belongs to the same chemical group as silicon, zirconium, cerium, and thorium, but it occupies a position nearly midway between tin and silicon and by some might be grouped with the former element. In other ways it is allied to iron, chromium, and aluminium. It is attacked by common mineral acids and has a great affinity for both oxygen and nitrogen. Titanium burns more energetically in oxygen than any known substance and when heated in it creates an instantaneous dazzling flame like lightning. Its combination with nitrogen gas is attended with the evolution of heat. Moissan's investigations led him to the conclusion that it was the only undisputed example of the combustion of an element in nitrogen; while Wöhler stated that no other element burns in nitrogen with so much energy as titanium. No other facts have since been brought to light to disprove these statements.

The titaniferous iron ores are very abundant throughout the world and in some parts exist in very large deposits. In fact whole mountains of it are located in the United States and Canada, and in Sweden and Norway. These were considered worthless for many years as the titanic acid produced a very infusible slag and attacked the lining of the blast furnace. It was even considered dangerous to the operations of the furnace to have too much of this ore present. In many cases the titanium combined with the nitrogen, set free by combustion, in the high heat of the blast furnace and formed a compound that entered the slag and made it so heavy it completely choked the furnace. Violent boiling has been caused in experiments carried on in the electric furnace and in some cases the charge has risen completely out of the furnace.

Most of this is due to improper fluxing, as the best of results were obtained in a blast furnace in the Adirondack Mountains in New York State, U. S. A., where iron ores very high in titanium were smelted for a period of something like twenty years. In England, at Norton-on-Tyne, the same thing has been done for several years. At the present time many tons of high-grade pig iron are being made from such ores in several different countries. They are generally so low in phosphorus and sulphur that ways and means are continually

being found to use more and more of these ores. Many blast furnaces, that do not use titaniferous ores entirely, are mixing them with other ores for this reason. Another use to which they can be put is, as an addition to the bath in basic open-hearth furnaces. Here they would act as a desulphurizer and flux for the basic slag in place of fluor-spar as they perform exactly the same service as this latter material.

The greatest benefit that has accrued to the steel maker's art from titanium is not in the ways and means that have been devised for using the titaniferous ores, but from using the titanium in the ferro form, as a purifying and alloying material.

No method was found for reducing the ore and extracting the pure titanium until the electric furnace, with its high temperatures, was made a practical success. This was due to its comparatively high melting point of 2300 deg. C. Titanium also has a specific gravity of 5.174, which is considerably lower than that of iron at 7.2. This, in combination with the high point of fusion, made it impossible to combine metallic titanium with the molten bath of iron or steel, when held at workable temperatures, and hence no beneficial results could be obtained until ferro-titanium was produced and experimented with.

The nature and action of the metal was such that it encouraged investigation and experimentation. Through these it was found that raising the titanium content of the ferro alloy multiplied the difficulty encountered when trying to induce it to combine with molten iron or steel, and made impossible the task of getting it to enter into solution. On the other hand it was necessary to get enough titanium into the bath to perform the intended mission. The many tests made conclusively proved that the best results were obtained from ferro-titanium that contained from 10 to 15 per cent of titanic acid. When 20 per cent was present it would not dissolve thoroughly at the usual working temperature of an iron or steel bath, and higher percentages caused a segregation that produced hard spots in the metal. The 10 to 15 per cent ferro-titanium generally contains about 6 per cent of carbon and 5 per cent of all other impurities. As one-half of 1 per cent of titanium is all that is usually added to the molten steel, the percentage of carbon and other impurities, in the finished product, can easily be controlled.

The great affinity of titanium for both oxygen and nitrogen makes it of special interest to the steel maker and also to the steel user. It has been proven, beyond a doubt, that these two latter elements lower the static strengths and dynamic properties of steel, the wearing qualities and resistance to friction, and increase its tendency to corrode. That they are more injurious to steel than sulphur and phosphorus and present in larger quantities than has hitherto been supposed, is also considered a fact by those that have made the most careful investigations along this line. That a sulphur content as high as 0.13 did not develop "hot-shortness" in steel, and had no other injurious effects, was fully demonstrated by one series of tests in which the other impurities in the metal were reduced to traces only.

Removing the gases from steels, by the addition of titanium, directly increases the cohesive force in the mass by removing the microscopic, or even larger, bubbles which they form. These destroy the cohesive force between the molecules surrounding such spaces. Indirectly it strengthens the metal by preventing these gases from forming injurious compounds with other elements that are present.

As both oxygen and nitrogen are gaseous by nature it is difficult to analyze steels for these elements, but satisfactory results are now being obtained. In one lot of twenty-four specimens the oxygen content was found to range from 0.021 to 0.046 per cent. These may seem small enough to be ignored, but it is the combinations the oxygen will form, and not alone the percentage of this element, that must be considered when its effect on the steel is judged. As an example of this, we talk about the injurious effects of 0.05 per cent of sulphur, when it is really the 0.13 per cent of manganese sulphide that alters the quality of steel. Oxygen has

only one-half of the atomic weight of sulphur and, owing to its being capable of forming many more compounds, it exerts a much greater influence. Thus 0.05 per cent of oxygen may mean that 0.22 per cent of ferrous oxide is present and this is a large enough percentage to have an important influence on the metal. The oxygen was determined by heating very fine drillings of steel in a current of pure dry hydrogen, and weighing the water, formed by the reduction of the oxide.

When oxygen comes in contact with steel, either in a moist or heated atmosphere, oxides are formed. These show as tiny black specks under the microscope, when the surface is highly specular and magnified at least 1000 diameters. These oxides are always found in steels that develop blisters when being pickled. Bessemer steels contain the largest amount, with open-hearth steels second; crucible steels next, and electric furnace steels the most free from oxides. In fact steels have been made in the electric furnace that are apparently entirely free from oxides. The blisters are probably formed by the reduction of the oxides by the nascent hydrogen evolved during the pickling operation. When pickling high carbon steel rods they sometimes fracture in the bath and the same force that forms blisters on the softer steels might cause the harder steels to rupture. When titanium is added, in the proper manner, to the bath of molten steel it forms, with the oxygen, a titanium-oxide. This, being lighter than steel, floats to the top of the bath; enters the slag and is removed therewith. Thus no injurious compounds are formed with the oxygen.

That the reduction of the oxides is brought about by the nascent hydrogen may be easily demonstrated. To do this a polished surface should be made the negative electrode in a solution that is just strong enough to make it an electrolyte. The solution may be composed of 0.002 per cent, or two parts of a neutral salt, or an alkali, in 100,000 parts of distilled water. This should be heated to boiling and an electric current passed through it, to evolve the hydrogen in contact with the polished surface. After exposing the specimen for a few minutes it should be withdrawn and dipped in alcohol to prevent oxidation before it is examined.

Such a specimen will show that the manganese sulphide is unaltered, and the oxide is removed. The pits left by this removal of oxide will be larger than the specks of oxide formerly seen, as any oxide that may be in solution around these specks will also be removed. This explains why more oxide is found on chemical analysis than can be seen under the microscope. That both the current and the hydrogen are necessary can be proven by the fact that no pitting is produced by the same process minus the electric current.

That oxygen increases the tendency of steels to corrode is shown by the fact that polished specimens, containing oxides, begin to rust much sooner than those that are practically free from them. In dilute acid solutions this is greatly accelerated. That all impurities except oxide are electro-negative, as compared with iron, is generally accepted as the reason for the more rapid corrosion. Oxide of iron being electropositive when compared to iron, the electromotive force causes a dissolution of the steel in the vicinity of the oxide. This action continues until the oxide is removed by other means as, of itself, it does not dissolve with the surrounding steel. Welds are never perfect as we are not able to squeeze all of the oxide out of the joint. It there forms into small globules that interrupt the cohesive force which binds the molecules together.

These, as well as other, instances of the injurious compounds that are formed in steel by oxygen, show the necessity of removing as much of this element from steel as possible. While silicon, manganese, and other of the more common elements have been used for a long time for this purpose by steel makers, they still leave a percentage in the metal that is large enough to become an important factor. Quite recently, therefore, aluminium, vanadium, titanium, zirconium, etc., have been brought into use to remove the smaller percentage that is left by the ordinary steel making methods, and very good results have been obtained with all of them. Titanium does this work as effectively as any of the foregoing and is destined to be

much more universally used as it is easily made to perform its mission and is much cheaper in cost. In the manufacture of steel rails with the Bessemer process it only adds about \$1.20 per ton to their cost. In higher grades of steel the additional cost is seldom up to \$2.00 per ton.

That nitrogen is injurious to steel is shown by the fact that each increase causes the elongation to rapidly diminish and reduce its ductility. At first a slight increase is caused in the toughness but after that this also rapidly diminishes, together with the loss of ductility and elongation. Generally speaking steels have lost their elasticity, and their elongation and contraction have become nil when a nitrogen content of between 0.030 per cent and 0.035 per cent is in the steel. An 0.5 per cent carbon steel might not lose its ductility, however, until from 0.04 to 0.05 per cent of nitrogen was reached, and in the softer steels from 0.05 to 0.065 per cent might be present.

It might be claimed by some that hydrogen gas caused this brittleness that is assigned to nitrogen, or at least a part of it, as hydrogen imparts extreme hardness to otherwise pure iron. Hydrogen gas, however, is easily driven off by any of the re-heatings the iron or steel gets and doubtless none of it can be found in the finished product.

In open-hearth steels the nitrogen content is usually between 0.020 and 0.025 per cent; in Bessemer steels from 0.018 to 0.062 per cent; and crucible steels from 0.010 to 0.015 per cent. Thus at least 0.012 per cent of nitrogen must generally be reckoned with. Steels made in the resistance electric furnace are practically free from nitrogen and hence are an exception to the above rule. Those, however, that are made in the arc electric furnace, in the presence of basic slags, are liable to contain enough nitrogen to be injurious.

The Manchester Steam Users' Association made some investigations of boiler plates that had failed. They found that the nitrogen content was from 0.0146 to 0.023 per cent and that this was from three to five times as great as the maximum found in any of the good plates that were tested. The notorious boiler plate from the Imperial Russian yacht *Livadia* contained 0.0123 per cent of nitrogen and 0.047 per cent of phosphorus. In the above tests a sample cut from a plate that burst under the hydraulic test showed 0.020 per cent of nitrogen and 0.052 per cent of phosphorus. Here the nitrogen was four times the amount of that found in any good plate tested. From these results the conclusion was drawn that nitrogen was more injurious to steel than phosphorus and that the one was as liable to produce brittleness and "cold shortness" as the other. It was concluded that no investigation was complete that did not determine the percentage of both phosphorus and nitrogen. They also established a rule as follows: The sum of the percentage of phosphorus, plus five times the percentage of nitrogen, should not exceed 0.08 per cent.

Nitrogen will not combine with steel by heating the two together, but in an atmosphere of ammonia they will unite. If the coking of the fuel has not entirely removed the nitrogen in the coal, ammonia is doubtless introduced into the blast furnace in that way. When nitrogen has once entered the pig iron in the blast furnace it cannot be removed by any subsequent reheating and hence is found in the finished steel.

Titanium, however, has a great affinity for nitrogen and when properly added to a steel bath it forms with it stable nitride which is lighter than the molten steel and raises to the slag. This nitride shows as tiny red crystals under the microscope. The very energetic reaction of titanium and nitrogen takes place at a temperature of about 800 deg. C. Combinations that are of no value are formed, however, if the air is carelessly shut off during this reaction and thus the good effects of their union will be greatly lessened, if not completely nullified.

Titanium is about the only element that readily combines with the nitrogen in steel and carries it away to the slag. While both silicon and manganese reduce the oxides they have practically no affinity for nitrogen. Chromium burns in oxygen at 1700 deg. C., but does not combine with nitrogen any more than does silicon or manganese. Molybdenum, even in pow-

dered form, will not form into combinations with nitrogen at temperatures up to 1200 deg. C. Tungsten will not form a nitride with nitrogen. Nickel also has no effect on nitrogen. While vanadium is a great scavenger of oxygen, it only effects the nitrogen in steel to a very limited extent. This leaves titanium alone as the nitrogen scavenger.

The proper method of adding the ferro-titanium to the molten metal is of vital importance. When improperly added it passes into the slag without going through the steel to search out and seize the oxygen and nitrogen and carry them with it to the slag. Some have advocated putting the ferro-titanium into the furnace but this only comes from a misconception of its nature and action, as silicon, manganese or spiegeleisen can better do the rougher work of oxidizing the steel and are much cheaper materials to use. Furthermore, adding the titanium in the furnace might cause it to work off into the slag and allow the gases to re-enter the steel before teeming. Titanium should only be used for removing the last traces of these gases, after the ferro-silicon and manganese have done their work, and hence should be added in the ladle.

The ferro-titanium comes well broken and ready for use. Being lighter than iron, it will not sink and disseminate when added near the top. Therefore, it should be put in, a shovelful at a time, while the molten metal is flowing into the ladle. If the stream is made to strike the ladle at one side a swirling and churning action is given the molten mass in the ladle and any ferro-titanium that might be floating on the surface will be engulfed and disseminated through the mass where it is quickly dissolved and starts on its mission. It should never be preheated as this sometimes prevents a complete reaction in the steel. Some begin by adding 1 per cent of titanium and gradually reducing this until the beneficial results obtained begin to diminish from the maximum point obtained.

The steel in the ladle should never be above the normal temperature for good untreated steel, as the reaction set up by the titanium causes the temperature to rise to a certain extent. After adding the titanium the ladle should be allowed to stand from five to fifteen minutes so the impurities and slag, released by this reaction, will have time to raise to the top of the ladle and not segregate in the steel. It is very difficult to make steel makers realize this as no other material they use acts in this manner. They fear the molten metal will become chilled and hence want to pour it into ingots before the titanium has accomplished its purpose. After standing, however, the metal is in better condition for pouring than it was before. Owing to an accident, one ladle had to be held for twenty minutes after teeming and it was then found to be in better condition for pouring into ingots, than was a freshly teemed ladle of ordinary steel.

Virtually there is no segregation in steel ingots containing titanium. Segregation is caused by accumulations of phosphorus and sulphur combinations, with iron and manganese, carbides, etc., along the main axis of the ingot. Where the metal remains longest fluid, near the top, there will be the most segregation. Selective crystallization as the steel begins to harden and the lower specific gravity of these combinations are the two principal causes of this segregation.

The gases liberated from the steel as it cools and hardens, raise in vertical streams and carry up the specifically lightest elements. Selective crystallization entraps them as the metal cools and the top of the ingot will hold the most segregation. If these gases are made to combine with an element like titanium, which carries them into the slag, the steel will harden with practically no occlusion of gases. Then very little segregation can be present, as the force is removed which caused the specifically lighter impurities to rise and collect in the top of the ingot. Titanium steels are also practically free from blow-holes and the above described action of titanium is doubtless the reason.

If there is any titanium in the steel after the occluded gases have been removed it has a tendency to attack the sulphur and phosphorus and either counteract their injurious effects or remove them. The sulphur is reacted on and carried away to

the slag in the form of a sulphide, or sulpho-cyanide, of titanium. Special means have made the phosphorus pass into the slag as phosphate of titanium. Thus the mission of titanium does not seem completed after the oxides and nitrides have been removed.

Ferro-titanium has been used in all of the steel making processes, including the electric, and the metal has been made stronger and longer lived thereby. It was thought that the electric furnace steels would be made so free from gases of all kind that they would not need the service of any purifying element. This, however, has been proven erroneous, for titanium is now used for the removal of the last of the gases in electric furnace steels. It is less expensive and troublesome to use this material than to remove all of the occluded gases with the electric process. An exception must be made in the resistance electric furnace however, as steels made in that type are practically free from oxygen and nitrogen.

The abrasive or frictional wearing properties of titanium steel are well illustrated by the sections of 90-lb. rails which have recently been taken up after being in service for twenty months on the high-side of two curves, of practically the same curvature and grade, on the Lehigh Valley R. R. The average of a number of rails, measured at this same time, gave a ratio of wear of 1 to 1.54, in favor of the titanium treated rails. The deflection was 1.40 inches for the titanium and 1.56 inches for the ordinary rail.

All of the tests that have been given steels for endurance show that titanium greatly increases this property. In one test of open-hearth steel, on the White-Southern rotary vibrational machine, it withstood 2,660,000 revolutions before it broke. The same steel, treated with titanium, withstood 18,274,900 revolutions before it broke. Other tests such as impact, cold bend, alternating vibrational, compressive and tensile strength show an improvement when titanium has been properly added to the metal. Such encouraging results have been obtained with steels that a large amount of work is now being done on the brasses and bronzes so titanium can be successfully used in the non-ferrous field. It has been used for some time in copper and greatly reduces the blow-holes when casting this metal. Very good results have also been obtained when using it in iron and steel castings.

The production of gold, silver, copper, lead and zinc in the Western States and Alaska in 1911 has been announced by the United States Geological Survey in two advance chapters from Mineral Resources of the United States, 1911. Many interesting facts and figures are given showing the sources and distribution of the different metals. Milling processes produce over half the gold, while smelting produces over four-fifths of the silver. The report gives the ore tonnages and metal contents for the different districts in the various states. Copies of the reports may be had on application to the Director, United States Geological Survey, Washington, D. C.

The size of the feed to Chilean mills is an important factor in their operation and efficiency. According to Mr. G. A. Denny, *Informes y Memorias del Inst. Mexicano de Minas y Metalurgia*, Vol. III, No. 1, it is a wholly mistaken idea that a Chilean mill has a larger output if fed with fine material than with coarse. Contrariwise, it has a considerably higher output on ore fed from breakers set at 1.5 in. than on a pulp discharged from a 3-mesh screen, and probably four times the efficiency on the coarse feed cited than on ordinary middlings or tailings.

The effect of flotation on the world's supply of zinc was first noted in 1904, and in 1906 the spectacular rise in the production from Broken Hill began. Between 1906 and 1911 the production rose from 100,000 tons of concentrate to 500,000 tons. As a result of a statistical study of zinc production at Broken Hill Mr. T. J. Hoover is of the opinion that no further large zinc concentrating mills will be erected there, that there are no further large discoveries of new deposits or lodes in the district, and that the quantity of tailings now stacked is accurately known, as is the quantity being treated. He assumes that the production for 1913, 1914 and 1915 will be between 400,000 and 450,000 tons of concentrate per annum.

The Gayley Dry Blast Process*

By Prof. Henry M. Howe

Interesting as are the other aspects of the Gayley process, its greatest interest lies, I think, in the light which it throws on the nature of expert evidence and on the value of expert opinion.

What is this process? It is simply drying the air used for burning the fuel in the iron blast furnace, apparently a rather simple matter. To refresh your memory, the blast furnace is a huge vertical firebrick cylinder, roughly speaking. In it iron ore, which is essentially iron oxide, is converted into cast iron or pig iron by prolonged exposure to coke or its equivalent, in an atmosphere of mixed carbonic oxide, carbonic acid, and nitrogen, which results from burning this coke by means of atmospheric air forced in through appropriate openings near the bottom of the furnace, and called "the blast." The furnace is full from top to bottom of an intimate mixture of this coke and ore, together with limestone added for the purpose of forming with the barren mineral matter of the ore and with the ash of the coke a fusible silicate or slag, and for other purposes into which we need not enter here. These three solid materials collectively are called "the stock."

The burning of the coke generates at the bottom of the furnace a temperature so high as to melt away the bottom of this column of stock, or mixed coke ore and limestone, of which the last two have by this time been converted locally into metallic iron and lime, and as the column thus descends it is renewed at its top by adding more of this same mixture so as to keep the furnace continuously full.

There are, as it were, two rivers passing through this great cylinder in opposite directions, a sluggish river of solid stock which descends as its bottom is melted away, and a swift river of ascending gases resulting from the burning of the coke by the injected air or blast. The former traverses the length of the furnace in from 12 to 15 hours, the latter in a very few seconds. These rivers interact as they interpenetrate and flow past each other, the rising gaseous column progressively giving up its heat to the solid column and taking from that solid column the oxygen of its iron oxide, so that the gaseous stream as it emerges from the top of the furnace has taken from the descending ore all the oxygen that it is capable of removing, and has delivered over to that ore and its accompanying coke and limestone all the heat that they are capable of taking from it.

So much for the blast furnace process, which, like every mundane process that seems simple, is in fact of a complexity so overwhelming that the human mind is inherently and incurably impotent to grasp it. We rub our eyes and seeing as far as the ends of our noses, assume that we see to the end of the universe.

As a heat engine the blast furnace was known to be extremely efficient, as human heat engines go. Nevertheless by the extremely simple step of drying the blast Mr. Gayley made a further important saving of fuel amounting in some cases to 20 per cent, and according to our present evidence to about 10 per cent on an average, for usual American conditions. To those who had taken up the logical stick by the wrong end, such a saving by thus drying the blast seemed simply preposterous, as preposterous as talking a thousand miles through an iron wire, or waving a message to Europe without any wire, or as any invention is till you understand it.

Its preposterousness was promptly and convincingly exposed by the public spirited experts whose geographical misfortune prevented their knowing Mr. Gayley's character, though those of us more favored geographically followed "Br'er Fox" till our question "Why and how?" could be answered.

The heat required for heating and dissociating the moisture of the blast is a small fraction of the total heat requirement of the blast furnace process; how then dare we say that the

removal of this heat requirement may save 10 or even 20 per cent of the fuel?

Not a few of us had grown so used to calculating the heat needed for a given chemical and physical work like that of the blast furnace, and the heat evolved by the combustion of a given weight of fuel under known conditions, and to calculating thus the thermal balance, that we had lost sight of the supreme importance of temperature.

The atmospheric moisture, calorimetrically considered, is indeed slight, but thermometrically considered it is sometimes of overwhelming importance. As to its effect on climate Tyndall says "To say that on a day of average humidity in England, the atmospheric vapor exerts 100 times the action of the air itself, would certainly be an under-statement of the fact."

The sum and substance of it is that the blast furnace process has among its various duties to supply a certain quantity of heat at or above a certain high temperature, which for brevity we may call the critical temperature, while shamefacedly confessing that we are further overworking an already grossly overworked word. No quantity of heat offered at any lower temperature will do this work. All the heat in all the suns of the milky way, if offered at a temperature of 50 deg. C., could not boil one egg.

Drying the blast saves fuel by improving the temperature-distribution of the heat generated.

Perhaps this is more readily understood if we consider an imaginary case. Suppose that matters were so ordained that he who wished to build an iron building was obliged to take, along with his iron work, a certain fixed proportion of accessories, windows, flooring, tiling, etc., the proportion fitted for a ten-story building. That would be all well enough for those who build nine, ten, or eleven-story buildings; but for the builder of a forty-story building it would be most uneconomical, because whereas the quantity of accessories, windows, flooring, tiling, etc., increases directly as the height of the building, the iron work needed increases in a higher ratio; so that, in order to get enough iron work for his forty-story building he might have to get enough accessories for a sixty-story building, and the excess would be left on his hands. This is because of had proportioning of his supplies. A mere change in the proportion between iron work and accessories would cause a great saving.

It is somewhat so with a thermal process like that of the iron blast furnace. When combustion has raised the temperature above the critical point there is available first a certain quantity of heat above that temperature, the hyper-critical heat, and second another and larger quantity of heat below that temperature, the hypo-critical heat, the heat remaining when the temperature has sunk to the critical point. The hyper-critical heat is needed for certain work which can be done only at hyper-critical temperatures, the hypo-critical heat is available for work which can be done at hypo-critical temperatures; so that, as we have hyper and hypo-critical heat, we have also hyper and hypo-critical work.

For heat economy the hyper-critical heat should be to the hypo-critical at least as the hyper-critical work is to the hypo-critical work. This is indeed true of every specific temperature covered by the blast furnace process. The proportion between the two evidently depends on the temperature reached by combustion itself. The higher this temperature, the higher proportion does the hyper-critical heat bear to the hypo-critical heat.

If the temperature developed by combustion is so low that the ratio of hyper to hypo-critical heat is too low, is below the ratio of hyper to hypo-critical work, then in generating enough hyper-critical heat to do the hyper-critical work we are forced to generate an excess of hypo-critical heat over and above the hypo-critical needs of the process, and this excess of hypo-critical heat is used to poor advantage or is wasted. If my ratio of iron work to accessories is too low, then in providing my building with enough iron work for forty stories I am forced to provide enough of these accessories for sixty stories, and the excess is left on my hands. And as a mere change in

*Extracts from the address made by Prof. Howe at the presentation of the Perkin medal to Dr. James Gayley at the meeting of the New York section of the Society of Chemical Industry on January 24, 1913. An account of the proceedings, together with Dr. Gayley's address in full, was given in the February issue of this journal, page 71.

the ratio between iron work and accessories can enrich or ruin a builder, so a mere change in the ratio between hyper- and hypo-critical heat, induced by a change in combustion temperature, can lead to a wholly disproportionate change in the economy of our own vital processes or of the process of the blast furnace.

Thus it was with Neilson's introduction of the hot blast. Formerly blast furnaces were fed with cold air, with the result that, because of the low temperature of combustion, the proportion of hyper-critical to hypo-critical heat was far below the proportion of hyper- to hypo-critical work to be done, with the result that the burning of enough fuel to provide enough hyper-critical heat yielded a quantity of hypo-critical heat far in excess of the hypo-critical work to be done, and this excess was used to poor advantage or wasted. Hence raising the temperature of combustion by heating the blast led to a saving of fuel which, to those unable to think, was miraculous.

The degree of economy caused by blast-drying should vary from case to case with the initial lack of hyper-criticalness in the combustion temperature; and if there is no such lack initially, as may happen, then blast drying should cause no economy.

Other means of adjusting the ratio of hyper- to hypo-critical heat suggest themselves, such as raising the temperature of combustion by further preheating the blast, by enriching it in oxygen by removing part of the atmospheric nitrogen, or by electric induction at the very focus where the hyper-critical work goes on; and lowering the critical temperature by changes in the conduct of the process. We have not reached the end of knowledge in general, or of the improvement of the blast furnace process in particular. But the fact that blast drying, in removing the effects of the fluctuations in the atmospheric moisture removes a serious cause of irregularity in the working of the furnace and in the quality of its product, gives it an administrative and commercial advantage over other means of raising the combustion temperature which may well be decisive.

The deliberateness and caution, indeed the tardiness, with which the iron trade has proceeded in adapting dry blast is referable to various reasons, such as hesitation to incur the certainly great expense of its installation, the competition of other devices for increasing earnings in other directions, the wish of each to get the benefit of the experience gained by its earlier adopters, uncertainty as to whether the saving found by John Doe will apply to the different conditions of Richard Roe's furnace, and the like.

But, after all is said and done, that which interests us most is not the invention itself, important as that is, nor the great saving of fuel. The striking thing is the contrast between the mental attitude of the certainly very learned men of science who immediately stamped Mr. Gayley's claims as preposterous, and the attitude of this great Captain of Industry, who not only saw the saving to be effected but saw it so clearly that he was able to bring to pass the very costly experiments needed to prove his faith.

Let us learn the lesson of humility. Natural human caution is likely to prevent the cautious from saying "I know that so and so can be done" unless they do know it; but such cases as this show that it does not prevent even the well qualified, the expert, and the prudent from saying "So and so can not be done," though in fact it ought to prevent them, in view of the almost infinite excess of our ignorance over our knowledge.

*Columbia University,
New York City.*

The Utah Society of Engineers has recently secured new quarters in the Salt Lake Stock and Mining Exchange. The business office is in room 21, and the regular monthly meetings will be held in the large exchange room. The removal of the Society from its small quarters in the Newhouse building was necessitated by its rapid growth in the last few months. Mr. R. K. Brown is president of the Society this year.

Ore-Testing at the Colorado School of Mines

The Board of Trustees of the Colorado School of Mines has approved a system of charges to be made for the use of the experimental ore dressing and metallurgical plant of the school. Communications for further information should be addressed to the manager, William G. Haldane, Golden, Colo.

PRELIMINARY TESTS

The equipment consists of one 24-in. Wilfley table, one 4-in. by 6-in. two-compartment jig, one set hydraulic classifiers, one 6-in. by 48-in. amalgamating plate and one set of cyanide agitators. Lots of 10 to 75 lb. of ore are suitable for these tests. Price for each lot tested, \$25, including assays and analyses.

INTERMEDIATE TESTS

Small jigs and quarter-size Wilfley and Card tables. Lots of 100 to 1000 lb. of ore are desirable. Price for each lot tested, \$75, including necessary assays and analyses.

CYANIDATION

Preliminary cyanide tests will be made on lots of 25 lb. of ore. Price for each lot tested, \$30, including necessary assays and analyses. A detailed statement of results will be given, but the report will not embrace conclusions or advice. A consulting engineer should be employed to interpret the results and advise his client.

CARLOAD LOTS

In this work the full-size equipment of crushers, rolls, screens, jigs, regrinders, classifiers, tables and vanners may be used. These tests should be supervised by the customer's consulting engineer. The school does not assume responsibility for the results nor advise the adoption of any special treatment, but will place the appliances of the plant at the disposal of the consulting engineer. Charges will be from \$10 to \$100 per day for the use of such portion of the plant as needed.

DAILY CHARGES FOR USE OF PORTION OR ALL OF PLANT

Preparation and Sampling.—This includes weighing, crushing in Blake or Gates gyratory crushers, sampling by Vezin or Brunton samplers, and delivery of the ore to roll or stamp bins. Price, \$50.

Concentration.—Rolls, elevators, trommels, jigs, regrinding machines, classifiers, sand pump, tables and vanners. Price, \$50.

Amalgamation.—Stamp mill, with plate or inside amalgamation, classifiers, tables and vanners. Price, \$30.

Cyanidation.—Dorr classifier, tube mill, Dorr thickener, sand leaching tank, mechanical agitation tank, Paterson circulation treatment tank, pumps for sand, slime, solution and vacuum. Moore filter, Butters filter, zinc boxes and lead-lined acid tanks. Price, \$40.

Magnetic Separation.—Dings two-magnet separator with means of securing magnetic fields of high and low intensity. Price, \$10.

Magnetic Roasting.—Wilfley annular hearth revolving furnace, with pyrometers for temperature control. Price, \$15.

Electrostatic Separation.—Huff electrostatic separator, current-producing devices, and two types of separators for fine and coarse material. Price, \$10.

Use of entire plant per day, \$100.

ORE SHIPMENTS

Ore in small quantities may be shipped by Adams express to Golden, Colo. Larger lots should be sent by freight via Colorado & Southern Railway, or via Denver & Intermountain railway. All charges should be prepaid. Ore sent by freight must be hauled by wagon from the railroad to the plant at the shipper's expense. A deposit to cover all charges for ore treatment must be made in advance. When different tests are made on the same ore, only one charge will be made for assaying and analysis.

In gaging wire it is better to use a micrometer than either a disk or forked gage. The two latter are not as free from objections as the first. With a disk gage fine sizes of wire can be stretched in forcing the wire in the notch, while with forked gages it is possible to spring the gage to show wide variations. Neither the disk nor forked gage will give as close a measurement as the micrometer.

Annual Meeting of American Institute of Mining Engineers

Adoption of New Constitution—An Embarrassment of Riches in Professional Papers —The Symposium on Sound Steel Ingots

THE annual meeting of the American Institute of Mining Engineers, held in New York City on February 18 and 19, will go down into history not only as one of the most successful meetings from a professional engineering standpoint, but as the beginning of a new era in the Institute's history, as the result of the adoption of the new constitution.

On the evening of Monday, February 17, the members assembled for an informal social gathering at the Institute headquarters in the United Engineering Building. They listened to a gracious word of welcome by Dr. George F. Kunz and to a delightful address by the president, Professor Kemp, giving a sketch of the life work of the new president to be elected on the next day, Mr. Charles F. Rand, and enlivened by lantern slides in Professor Kemp's unique style which leaves the audience in constant reflection as to the extent of the genuineness of the photographs.

Business Meeting and Presidential Address

The morning session of Tuesday, February 18, was given over almost completely to the business meeting of the Institute. The chief result was the announcement that by the mail vote and the additional vote of those present the new constitution of the Institute has been adopted in the whole with an overwhelming majority, practically unanimously.

There was, however, an extra vote to be taken on four amendments by Messrs. Corning and Stone on minor matters, all of which were adopted, and on the provisional clause in the constitution providing for a special class of "Fellows" which should have the right to express an opinion with regard to pending or proposed legislation affecting the mining and metallurgical industries. The mail vote on the Fellows proposition showed a very slight majority against it (270 against the class of Fellows and 260 in favor of it). As this result could have easily been changed by the vote of those present, the Committee on Constitution, through Prof. J. W. Richards, explained that they had submitted the amendment concerning the class of Fellows by request and that they did not favor it now in view of the proposed union of the Mining and Metallurgical Society of America with the American Institute of Mining Engineers. Thereupon, the vote of those present was practically unanimous against the class of Fellows and a few who had voted by mail changed their votes, and the final joint result of the mail vote and of the votes of those present was 313 against the clause of Fellows and 258 for it.

Dr. Douglas in his warmly applauded report for the Land Committee stated that on the debt of \$68,000 there

had been subscribed already more than \$59,000, and urged those members who had not yet contributed to make up the small balance. This result seems to make sure that it will not be necessary to raise the membership fees, as with the land debt paid off the fixed charges of the Institute will be reduced by \$3,000 a year. Resolutions of thanks were voted to Dr. Douglas.

The result of the election of officers was then announced as follows:

Mr. Charles F. Rand, president of the Spanish-American Iron Company, was elected president.

Messrs. F. W. Denton, Karl Eilers, Sidney J. Jennings, T. H. Leggett, Waldemar Lindgren and B. B. Thayer were elected vice-presidents.

Messrs. J. W. Finch, James Gayley, J. A. Holmes, R. W. Hunt, John H. Janeway, E. P. Mathewson, W. J. Olcott, J. W. Richards, L. D. Ricketts, C. S. Robinson and E. L. Young were elected directors.

By the new constitution the secretary has ceased to be elected by the members at large and is being appointed by the Board of Directors. The proxies which had been made out were, therefore, not voted. The Board of Directors held meetings on Tuesday and Wednesday and during the afternoon session of Wednesday the new president, Mr. Charles F. Rand, brought to the members a "message from the Board of Directors," announcing that after several hours of consideration the Board had selected Mr. Bradley Stoughton for secretary.

The morning session of Tuesday was concluded by the address of the retiring president, Professor Kemp, which dealt with the subject of the ground waters in a very interesting manner. Lunch was then served and the reading and discussion of papers were taken up in the afternoon session.

Tuesday Afternoon Session

The papers presented in the afternoon session of Tuesday related to geology, mining and allied subjects.

The Role of Certain Metallic Minerals in Precipitating Silver and Gold

A paper by Dr. Chase Palmer and Dr. Edson S. Bastin, of the U. S. Geological Survey, was presented by Dr. Bastin and illustrated by experiments. The authors have made extended laboratory experiments on the rôle which certain metallic minerals play in precipitating silver and gold from solution and apply their chemical results to draw some interesting geological conclusions.

Their experiments show that certain sulphides, arsenides and sulph-arsenides of copper, nickel and cobalt precipi-



CHARLES F. RAND

President American Institute of Mining Engineers

tate metallic silver very efficiently from dilute aqueous solutions of silver sulphate. As the waters descending through the upper portions of most sulphide ore bodies are known to be sulphate waters, similar precipitative actions would be expected under natural conditions. The frequent association of silver in ore deposits with chalcocite and bornite, and particularly with niccolite and cobaltite, minerals which in the authors' experiments were among the most efficient precipitants, warrants the belief that such reactions are of great importance in secondary enrichment of ore bodies.

The more common sulphides such as pyrite, galena and sphalerite were relatively inactive as precipitants of silver from aqueous solutions of its sulphate.

The quantitative results obtained with niccolite and chalcocite indicate that the essential chemical changes in reactions of this type are due to oxidation through the hydrolitic action of water. It is apparent, therefore, that certain water solutions may act as potent oxidizing agents below the ground-water level.

The experiments indicated that nearly all of the sulphides and arsenides common in ore deposits were capable of reducing gold from a solution of its chloride, although important differences in the rapidity of the precipitation were observed with different minerals. Most of the minerals that are especially efficient as precipitants of silver are also effective precipitants of gold and a number of other minerals such as galena, pyrite, stibnite and millerite that are inefficient in precipitating silver are efficient in depositing gold.

It is known that the waters descending through the upper portions of sulphide ore bodies universally carry chlorides and it is probable that these chlorides have effected the solution of the gold. It is probable, therefore, that phenomena similar to those exhibited in the authors' experiments with gold chloride solution play an important part in secondary enrichment in gold.

While the phenomena described by the authors find their most immediate application in secondary enrichment it is perfectly possible that such reducing effects of the sulphides may be responsible in part for the primary association of the precious metals with certain sulphides in preference to others. It is recognized, of course, that certain mineral associations in ore deposits are probably the result of processes analogous to differentiation in rock magmas, but it is quite possible that other associations such as the apparent preference of gold for chalcocite and tetrahedrite rather than for pyrite in deposits carrying these three minerals may be due to differences in the reducing power of these sulphides themselves. Light could probably be thrown upon this point by the investigation of the reducing effect of various sulphides upon silver and gold salts dissolved in solutions having the composition of certain deep mine waters.

It is a generally recognized fact that the purity of alluvial gold is greater than that of the veins in the neighborhood. This superiority in fineness has generally been explained by the well-known fact that silver is more readily soluble in natural waters than gold, and is by them removed from the natural alloy, thus increasing its purity.

Mr Lindgren has recently discussed (U. S. Geo. Survey, No. 73, 1911) this matter at some length in a report on the Tertiary Gravels of California, and has presented a large amount of statistical data leading to the same conclusion. It has been thought by certain geologists that this refinement of the gold was accomplished by solutions circulating through the gravels themselves, but Mr. Lindgren states that "so far as the tertiary gravels of California are concerned, the conclusion of the writer is that solution and precipitation of gold have played an absolutely insignificant part."

Under the conditions of the experiments reported by the present authors it was found that nearly all of the

metallic minerals common in precious metal deposits were capable of precipitating gold while a much smaller number, and these not the most common ones, were active precipitants of silver. When it is remembered that the source of the placer gold is the oxidized zone of the original deposit and that the gold may have been dissolved and re-deposited several times within the vein before erosion carried it into the alluvium, it seems not improbable that such selective precipitation may be an important factor in this natural refining of gold.

A Microscope Study of Sulphide Ores of Copper

A paper by Dr. L. C. Gradon, of New York, and Dr. Joseph Murdoch, of Harvard University, Cambridge, Mass., was entitled "The Sulphide Ores of Copper—Some Results of Microscopic Study." The paper gives a very large amount of facts, experimentally established, and is illustrated by a series of thirty-six microphotographs of striking significance.

In the introduction the authors discuss the relation of scale in geologic work and the general characteristics of copper sulphide ores and outline the scope of their investigation. Then follow descriptions of the important minerals—pyrite, pyrrhotite, chalcocopyrite, bornite, covellite, chalcocite, enargite, tetrahedrite and tennantite. Each of these minerals is discussed as to primary character, alteration and secondary derivation.

Some of the more important conclusions are as follows:

Conclusion as to whether an ore or a given mineral in an ore is primary or secondary can be safely drawn in the great majority of cases from the type of structure involved.

In the case of chalcocite, internal or molecular structure as expressed in cleavage may also be used with caution as a criterion of the primary or secondary character of the mineral.

The following sulphide minerals common in copper ores are primary, i. e., they are products of original precipitation, probably from ascending, heated, alkaline solutions; pyrite, pyrrhotite, chalcocopyrite, bornite, chalcocite, enargite, tetrahedrite, tennantite, spalerite, galena, and probably covellite.

The following minerals are secondary, i. e., they are derived by the action of descending cold acid waters upon the other sulphides, i. e., secondary sulphide enrichment, chalcocopyrite, bornite, covellite, chalcocite and enargite.

The most important result of this study is perhaps "the abundant evidence afforded that primary copper deposits are definite, characteristic, clear-cut phenomena of striking similarity in the main, formed by well-defined processes as a result of certain specific causes acting through a limited and probably brief period of time. Among the ores that have been studied, at least, there are absolutely no indications favoring—rather, there is plentiful testimony against—the hypothesis that many or most ore bodies (excluding the enriched zone) are the result of 'repeated reworkings' of their constituents by various processes, of different causes, at separated times or over long extended periods. This idea, never ringing true nor supported by competent evidence, has served only to complicate and bfog the situation, and thus postpone universal acceptance of the fast-growing and now almost overwhelming proof of the direct and sole influence of igneous agencies in the formation of many ore bodies."

The following papers were also presented during the Tuesday afternoon session by their authors:

A Problem in Mining together with Some Data on Tunnel Driving. By F. M. Simonds and E. Z. Burns.

The Geographical Distribution of Mining Development in the United States. By E. W. Parker.

Electric Power Installation at El Tigre, Sonora, Mexico. By J. W. Malcolmson. (This paper contains some interesting figures of cost.)

Structure of the Northern Anthracite Basin Relative to Forms of Folds. By N. H. Darton.

Among the papers which were on the program for the afternoon, but which were read by title only, are the following:

The Hardinge Conical Mill. By **H. W. Hardinge**.

Fire Clay Deposits in Canada. By **Heinrich Ries**.

School Laboratory Work; the Sampling of an Ore Containing Coarse Gold. By **Charles F. Locke**.

In the evening a very interesting illustrated lecture was given by **Dr. Fred. Haynes Newell**, director of the United States Reclamation Bureau, on "Immigration and Its Relation to the Mining Industry." This lecture was greatly enjoyed by a large audience.

Symposium on Sound Ingots

The morning session of Wednesday was devoted to the reading of five papers relating to methods for improving the soundness of steel ingots and to a general discussion of the subject. There were some 200 members and guests



BRADLEY STOUGHTON

Secretary American Institute of Mining Engineers.

present, and the attendance was truly representative of the industries interested in sounder ingots. But the representatives of most of the big steel companies showed a rather regrettable reluctance to express their views. Nevertheless, the meeting was exceedingly interesting and profitable.

This symposium had been arranged by the Iron and Steel Division of the Institute, with Mr. Charles Kirchhoff, chairman, and Mr. Bradley Stoughton, secretary. As mentioned before, Mr. Stoughton is now also secretary of the Institute as a whole.

Piping and Segregation of Ingots of Steel and Ductility Tests for Open-Hearth Steel Rails

A paper by **Dr. P. H. Dudley** of the New York Central Lines on the above subject was first presented. It was an elaborate discussion of extended tests made on this subject.

Mr. Dudley called attention to the fact in the specifications for the New York Central Lines that in mill practice as soon as the ingots are stripped they should be charged into the reheating furnaces to prevent the setting steel to cool from its molten temperature to that of cold metal, and thus avoid the formation of the full shrinkage cavities in the ingots. It has been shown by the cutting of a large number of blooms that it is possible to prevent a shrinkage cavity from forming of not more than from 1.20 to 1.30 of the size in the top of the hot ingot by this method of good mill practice, of what would be formed by permitting

the ingot to become completely cold before it was put into the reheating furnace for rolling.

It is first necessary, however, to provide the hot molten metal with a chemical composition which will produce sound ingots and definite physical properties in the finished product. The first effort is to secure a well-deoxidized steel, and by proper mill practice make sound ingots.

Dr. Dudley does not consider it advisable to follow the suggestion of Mr. Talbot and Sir Robert Hadfield to use a large percentage of aluminium in the ingots to reduce more completely the oxides, in view of the difficulties already experienced with aluminium so used in rail steel for heavy wheel loads. "It would be better to use silicon or a combination of silicon and ferro-titanium to secure the desired results. We do not use as high percentages of silicon in steel as is employed abroad, except for tires."

Dr. Dudley then dealt at greater length with the segregation of basic open-hearth steel ingots, giving analytical diagrams of various ingots and finally discussed the drop and exhausted ductility tests for basic open-hearth steel rails.

Dr. Dudley's final conclusions for the manufacture of the present basic open-hearth rails are as follows:

"1. The chemical composition should provide for sound steel of ample physical properties of tenacity and toughness rather than hardness combined with brittleness.

"2. The impurities, phosphorus and sulphur, should be of minor content so the bath of metal can be purified to produce the large percentage of toughness and ductility due to the specified chemical composition.

"3. The ingot should have such relations of area of base compared to the height and weight that under good mill practice and suitable deoxidizers it can be made with controlled segregation, and only a trace of a shrinkage cavity in the top; then, when bloomed under its equalized initial heat, it is rendered pipeless by the usual 8 to 10 per cent discard.

"4. Aluminium can be replaced and silicon partly, as deoxidizers, with advantage by the use of ferro-titanium to purify, solidify and check segregation in rail, tire and axle steels, and also some of the lower grades of carbon steels where great purity is desired.

"5. The ductility and elongation tests to date furnish the best and only prompt means of determining the degree of purification of the steel per melt as it is made by indicating the physical properties secured before another melt is tapped from the same furnace, and are of decided advantage to the manufacturer as well as to the consumer. These tests are so advanced that they must be applied with knowledge and understanding for proper results, and not made mechanically for specified records.

"6. Every process or step of the entire manufacture of the steel, rolling and finishing of the rails, must contribute its part to secure the highest quality of the product incident to the chemical composition.

"7. Specifications should be drawn to indicate some of the major necessities of the consumer, and the tests and inspection conducted in a spirit to aid and invite the co-operation of the manufacturers to meet the progressive requirements in rail steel."

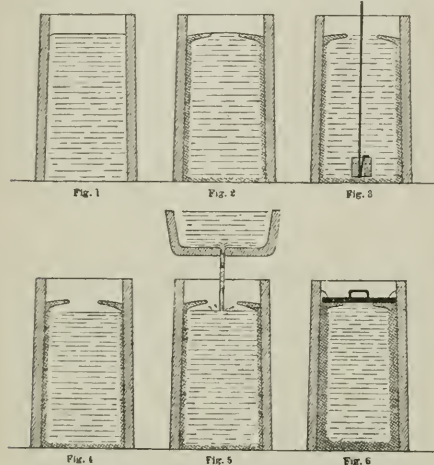
The Use of Anti-Piping Thermit In Casting Steel Ingots

The second paper of the session was presented by **Dr. E. A. Beck**, of the Goldschmidt Thermit Company, New York City. It deals with the interesting new application of anti-piping thermit for producing sound steel ingots. In contradistinction to an older method of using anti-piping thermit (in which a can filled with thermit was introduced into the upper part of the ingot and was intended to produce a heating effect), a new method has been more recently evolved in which the can with thermit is introduced as far down into the ingot as possible and simply a stirring action is intended proceeding from the

bottom to the top. This method has already been described in some detail in a paper by Dr. C. Canaris (this journal, Vol. X, page 232, April, 1912) and in a paper by Dr. Hans Goldschmidt (this journal, Vol. X, p. 739, Nov., 1912).

The application of the method is very clearly shown in Figs. 1 to 6, reproduced from Dr. Beck's paper and illustrating the different stages in the treatment of steel ingots with anti-piping thermit.

Fig. 1 represents a section of the mold filled in the usual way, either from the top or bottom. This mold



FIGS. 1 TO 6.—DIFFERENT STAGES IN THE TREATMENT OF STEEL INGOTS WITH ANTI-PIPING THERMIT

stands until a certain cooling off of the steel has taken place. The degree of the cooling can be determined approximately by watching the formation of the crust on the surface.

Fig. 2 shows how far the crust should extend before the introduction of thermit, in order to get the best results. In the case of a 5-ton ingot it takes about 7 minutes for the crust to form properly.

Fig. 3 shows the can introduced. The can is quickly pushed down through the steel to the bottom of the mold and kept there until the reaction is over. The resulting seething of the steel will last about 5 seconds.

Fig. 4 represents a cross-section through the mold after the reaction has taken place. The empty space which formed indicates that the steel has become denser.

Fig. 5 illustrates the addition of new steel. Care must be taken that the added liquid steel does not rise over the edge of the solidified crust.

Fig. 6 shows the treated ingot after the lid has been properly put on.

The advantages of this use of anti-piping thermit are due to I, improved quality, and II, better results in rolling.

I. Quality.—1. The steel becomes denser in proportion to amount of metal added after the thermit has been applied. The increased density is caused by the expulsion of the occluded gases due to the effluvia following the thermit reaction.

2. The removal of the already formed segregation toward the middle of the ingot and by their transfer to the surface of the ingot; further, segregations are avoided through equalization of the temperature of the steel.

II. Rolling.—1. Elimination of the laps which occur from piped ingots, or from ingots with a sunken or irregular head.

2. Better physical properties of the steel in the heads.

3. A saving of from 3 to 10 per cent of the finished

material will be obtained. An average of 5 per cent would give a profit of about 75 cents per ton.

4. Elimination of defective plates caused by large blow holes on secondary pipes.

5. In small ingots the cost of the treatment amounts to about 35 cents per ton in the open-hearth department; against this there is an estimated saving of about 75 cents per ton in the rolling department. In large ingots (about 8 tons and more) the cost of the treatment is only 20 cents per ton and the saving is the same as in a small one.

6. A still further saving, which could not be calculated exactly, will be derived from the decreased percentage of defective plates, which is cut down to about 0.3 per cent, according to Dr. Canaris. In this case all plates which do not give the required dimensions or have not the desired physical properties in the head are classed as defective. Untreated ingots of the same group are often not uniform. Frequently there is a sound ingot, which will give good results without thermit treatment, and next to it there may be one with a large pipe or blow hole. Uniformity is secured by the thermit treatment.

Commercial Production of Sound Steel Ingots

A paper by Mr. Emil Gathmann, of Baltimore, described his process for producing sound steel in an economical and hence commercial manner, which is adaptable to the production of practically all steel manufacturers, by readily effected rational changes in the methods of casting, cooling, and subsequent handling of the ingots.

The process is based on the observation that the freezing or solidifying of an ingot which has been practically deoxidized or, as the term is used, "killed" in the mold, depends entirely upon the shape of the horizontal cross-section of the ingot at its various planes from top to bottom, and also upon the thickness and consequent heat-absorptive power of various parts of the mold walls.

This fact is clearly brought out and in a series of photographs in Mr. Gathmann's paper. As a result of these facts observed in practice, Mr. Gathmann designed a metallic mold constructed to accelerate the cooling of the lower or greater portion of the molten mass and teemed ingot (approximately from 80 to 85 per cent) and to retard the cooling of the uppermost portion of the ingot, thus causing the upper portion to remain liquid longer and to act as a feeder. The upper portion of the ingot does not actually remain liquid much longer than in the usual practice for similar cross-sectional area, but as the cooling of the lower portion is greatly hastened a differential in cooling is obtained, which is really what is to be desired.

According to Mr. Gathmann's method it is essential that there be considerable lag in the cooling of the upper portion of the ingot compared with the time of freezing of the lower portion, hence, the distance from the vertical central axis of ingot is made much less at the bottom of ingot and progressively increases toward the top of the ingot. In crucible steel ingots, which are largely made in split molds, the actual sectional area of the top and bottom may remain practically constant throughout the length of the ingot, but the distance from the vertical longitudinal axis to the surface of the ingot progressively increases toward the top. In ordinary big-end-up practice, where sufficient taper or differential in distance from the vertical axis to the surface of the ingot is given, to accomplish any notable reduction in depth of piping, the actual cross-sectional area of lower part of the ingot is much less than that at the upper part, hence the depth of pipe is not the true index of the actual volume or weight of cropping necessary to obtain physically sound steel.

One of the advantages of the system or type of ingot is that it is possible to obtain a practically uniform cross-sectional area at top and bottom of ingot and still obtain the benefits of the big-end-up type of mold.

Lifting of segregation is generally conceded to follow the

reduction of pipe, and where the pipe is lifted the steel below undoubtedly becomes more homogeneous and freer from segregation.

For open-hearth practice with the big-end-up ingot, one of the greatest difficulties has been to devise a method of stripping and handling the ingots.

Fig. 7 shows the Gathmann ingot mold and stool on a car or buggy after teeming. The big-end-up mold will

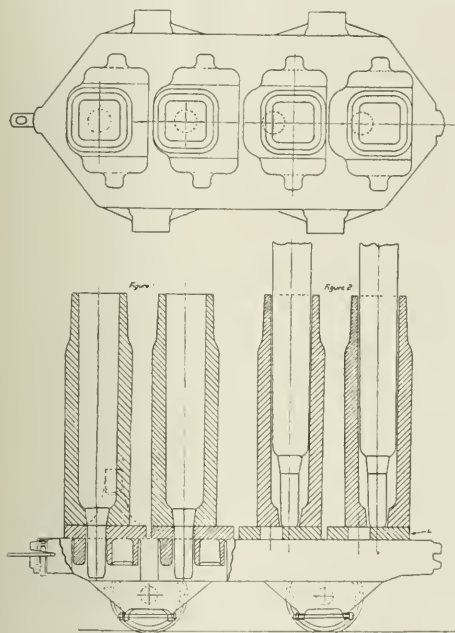


FIG. 7.—INGOT MOLDS FOR PRODUCING SOUND CASTINGS

require a special type of stool to avoid such difficulties as might ordinarily be experienced in stripping, due to fins forming at the base of ingot and locking it to the mold. A downwardly tapered plug, $c-c'$, seals the base of the mold and projects through the stool, $p-p'$. When the teeming is finished the metal, by reason of the wedge-like taper at the lower portion of the ingot, is forced into virtual contact with the walls of the mold, which, due to the thickness of same, rapidly absorb the heat of the ingot. A slight differential or reverse taper of the ingot cavity is made from 15 to 20 per cent from the top of the mold. The ingot in shrinking will automatically provide an air space at this portion, thus breaking the contact of ingot from mold walls and retarding the flow or loss of heat from this portion of ingot.

Other illustrations in the paper, not reproduced here for lack of space, show a modification of the Gathmann mold for bottom-cast ingots, a stripper with provisions to suit the stripping of the big-end-up mold, and a slab ingot mold for plate or sheet mills desiring to roll directly from the ingots.

The author concludes that the general practice employing his process "reduces the pipe in the deoxidized or killed steel so that with an average discard of 12 per cent sound metal will be obtained.

"Segregation will undoubtedly be disposed of in direct ratio to the lifting of the pipe." . . . "This system is not in an experimental stage as many tons of steel are daily being cast in both open-hearth and crucible plants in accordance therewith."

Production of Solid Steel Ingots

A paper by Mr. Benjamin Talbot, of Middlebrough, England, with the above title was presented in abstract by Mr. Charles Kirchhoff.

It was essentially the same paper as presented by Mr. Talbot in a special meeting held on November 7, 1912, in New York, and reported in our December, 1912, issue (Vol. X, page 772).

To supplement the information given in that paper and reply to some criticisms made at the former meeting, Mr. Talbot had communicated the following remarks. In the discussion on his paper very great surprise was shown at the fact that in the sulphur prints exhibited there was a regular area of higher carbon and sulphur metal upon the inner wall of the solid envelope, and a purer center inside that.

The question was then raised whether this formation was regular when the ingots were treated uniformly, and doubts were expressed of its being consistently found. Further experiments have shown that it can be consistently relied upon, and there is no break in the formation of this structure.

A photograph of the structure of an ingot is given in the paper, showing a gradual gradation of one structure to another. Results of drop tests and tensile strength tests are also given.

"Assuming that a sound rail with this structure (a hard working face, harder ring at back of this, and softer center) is accepted by rail users, we have then to consider the practical adoption of the process in steel works. There is no doubt that for open-hearth steel rails the large 250-ton tilting furnaces, giving heats of from 50 to 60 ton, irrespective of how the furnaces are operated, can give the steel in a more regular sequence, and are under better control as regards time of tapping than the so-called 60-ton fixed furnaces.

"As to the casting department, I may say that in my opinion the present ordinary methods have to be revised, and the casting must become an important department of steel manufacture. To-day both metallurgists and works' operators have neglected this department, and have not considered it as of anything like the same importance as the actual manufacture of the pig iron, or the conversion of the iron into steel.

"There is no doubt that the railway engineers have hit upon that point in the manufacture of steel rails which is causing most defects, and if given sound ingots the troubles now encountered in rail manufacture will be avoided. To my mind the casting department should have the attention of men skilled in the art to remove the slur now put upon steel makers in reference to cavities and blow holes, which form in the ingot during the solidification of the steel in the mold.

"I fully realize that if liquid compression of the ingot is to be adopted steel furnaces must not be allowed to 'bunch' together, and to tap a large number of heats at once as is the present practice. If a continuous set of rolls be used for liquid compression, then such rolls must be equal to dealing with the maximum quantity of steel tapped in any given time, but speaking as a work's engineer and practical operator, I see no difficulty in designing a plant to squeeze the required number of ingots to keep a modern rail mill supplied.

"Assuming that the necessary preparatory treatment has been made, we find that the size of the ingot must be standardized when such treatment is used and a 4-ton ingot seems to be a most useful size in this respect.

"Finally, in my opinion, the characteristic structure developed in rails treated by my squeezing process must cause these rails to be better than rails from the upper portion of ingots from ordinary heats which are always more or less segregated in the center."

Heating the Top

The Hadfield process for producing sound ingots by heating the top in a special manner was the subject of a paper by Sir Robert Hadfield, presented by Mr. Henry D. Hibbard.

Sir Robert Hadfield's process was described and illustrated in our November, 1912 issue (Vol. X, page 738), and his arrangements for applying the method on a large scale were described and illustrated in our February, 1913, issue (Vol. XI, page 79).

Mr. Hibbard gave an outline of the process, and exhibited a great many photographs and diagrams.

Comparative Notes on Steel Rail Rolling

A paper by Colonel Robert W. Hunt, of Chicago, Ill., was available in mimeographed form.

"I have frequently stated that while the chemical composition of steel was important, yet even greater importance was connected with the mechanical and heat treatment of the metal. During the past year I encountered such a positive example of that fact that I deem it worth putting upon record.

"A prominent railway system divided an order for open-hearth steel rails between two steel works, both of which are under the control of the same corporation, giving to one about 18,000 tons, and to the other 7500 tons of the same section and to be made under the same specifications. The rolling results obtained in two mills varied so widely that a study of the figures is intensely interesting and serves to illustrate the advantages obtained by careful ingot casting, and quite as pertinently the possible benefits of careful subsequent heating and rolling with moderate reductions in the rolling process. The smaller order was rolled complete in four installments during the same months that the larger order was being made in eight separate installments; but, to permit of exact comparisons, the totals of the four installments (completing the order with the mill which I will call A) are given with the totals of the first four installments of the larger order, made by the mill designated as B. These rolling results are:

	Mill A.	Mill B.
Number of rollings.....	4	4
Total number of rails rolled.....	18,278	27,832
Percentage of rails cut to short length because of flaws near ends, etc.....	0.7	7.6
Percentage of rails made second quality for flaws, etc.....	0.9	6.3
Percentage of rails scrapped for flaws, etc.	0.6	4.5
Percentage of rails scrapped for failure at drop test	none	3.4

"It should be stated that the figures above shown cover all the reasons for putting the rails in the classes stated. Thus a part of the percentage of rails cut to short lengths may have been because of bad drilling or bad sawing. Such classification, however, in this case is entirely proper, as neither mill suffered from unusual or abnormal difficulties in any way, and the figures indicate ordinary performance uninfluenced by unusual errors of either workmanship or mechanical troubles in rolling.

"Emphasis should be laid on the fact that both mills were working to exactly the same specification, and producing a section which has been in use for several years and in large tonnage. It is an 85-lb. one, having 36.7 per cent of metal in the head, 22.2 per cent in the web and 41.1 per cent in the base, being, therefore, well proportion for the avoiding of torn flanges or other rolling difficulties sometimes encountered with sections having thin flanges."

The chemical composition and drop test specified with average results obtained in the two mills are given in the paper. Both mills took advantage of the full range permitted in the chemical compositions. Though with respect to obtaining consistent carbon results, the results of mill

A were more consistent than those of mill B, the "chemical composition obtained at the two mills agrees so closely that some other reason must be sought to explain the divergent physical results obtained in the product, and, therefore, data on the actual performance of the mill operations is important."

This is given in form of various tables in the paper.

It appears that "both mills were casting large heats, approximating 85 tons. These were made by almost identical methods of the usual scrap and pig-iron process, the iron taken from mixers, as required, and varying in proportion to the scrap used. As far as possible, at both mills, this mixer metal was used for recarbonizing in the furnace, but many heats had coke or coal added to the ladle on tapping, in addition to the usual ferro-manganese and ferro-silicon. While the heats at mill A were slightly larger than at B, the difference is not of importance, but it is pertinent to note that there are but six furnaces represented at mill A, as against 27 which furnished the rail steel at B; and, consequently, there were probably but two steel melters working at A, as against at least six at B.

"The personal equation may, therefore, have played an important part in making the steel.

"Equally noticeable is the fact that while the average for the time intervals of the various operations at the two mills is not much different, still the range between the maximum and minimum for B are consistently greater than for A.

"There can be but very little doubt that a delay between casting and stripping ingots and between stripping and charging in the soaking pits is likely to be seriously reflected in the soundness of the ingots, and of at least equal importance is the necessity for teeming the steel at uniform temperature. A variation of 50 minutes in the time heats were held in the ladle prior to casting must have caused variable teeming temperatures and produced many blow holes in some of the ingots. Admitting the presence of blow holes near the surface of the sides of the ingots, and remembering the oxidizing action in the soaking pits, it is not surprising that heavy reductions in the blooming mill had an extremely detrimental effect on the product.

"This is reflected, no doubt, in the large number of rails found containing flaws at mill B.

"Appreciating, therefore, the probable difference in the ingots produced at the two mills and the actual difference in the blooming practice, the principal other variable existing was in the reheating of the blooms at mill A. There the ingots were cut into four blooms, which were then given a wash heat in reheating furnaces after which they were rolled into two rails in a rail train of eleven passes. At mill B the rail train consisted of nine passes with no reheating of the blooms."

The reported history of the heats rejected at mill B is also given, together with analyses of the rejected heats.

Two heats were rejected at the drop test because two of the three test pieces broke on the first blow of the tup. The steel of the first heat was reported as being somewhat low in temperature when tapped. While pouring, the nozzle froze up and while the ingots were in the soaking pits an average of four hours and twenty minutes, they bloomed cold and rough. The second heat was quite the opposite, the tapping temperature being high and while casting the stopper head was finally lost, but the heating and the blooming were normal.

Four heats were lost at the drop test, because they exceeded the deflection limits. On the first, all conditions were reported normal, save that the ingots were held in the pits for an average time of 24 hours. The second heat tapped cold, the pouring nozzle froze, and the ingot tops were spongy. The third tapped hot, but otherwise, like the fourth, had normal conditions.

General Discussion on Sound Ingots

A general and quite extended discussion on improving the soundness of ingots followed.

The following four questions on piping had been formulated by the committee of arrangements as a basis for the discussion:

1. Is the present method of getting rid of the pipe by cropping a safe and reliable device for making sound steel?
2. Is it desirable to increase the amount of metal arbitrarily cropped off the top of an ingot, and if so what would be a fair excess price to pay for cropping off 20 per cent?
3. Is it commercially practicable to make ingots without pipes or blow-holes and what addition of expense of manufacture would be justified to accomplish this result?
4. What process for making pipeless ingots seems the most promising of commercial success and why?

Dr. A. Sauveur, of Harvard University, opened the general discussion by referring to the fact that years ago he had pointed out how great a source of danger the pipes in ingots were for rails, but his word of warning had met only with opposition and it had been said that the pipes would be welded in manufacture of the rails, etc. It was a great advance that now both rail makers and railroads were willing to admit the danger. If the processes for producing sound ingots, described in the preceding papers, are right or if any one of them is right, there will be no more excuse in future.

Mr. Robert Job commented on the questions which were the basis of discussion. He pointed out that with good mill practice very little piping is present, while with poor mill practice 50 per cent discard might not produce sound rails. For this reason the question (2) where it is desirable to increase the discard of metal arbitrarily should be answered without hesitation in the negative. Such practice would simply penalize good mill practice. Tests should be made to detect piping in any ingot to be used for rail steel. The tests of the Lehigh Valley Railroad were described. A confirmative answer was given by Mr. Job to question (3) whether it is commercially practicable to make ingots without pipes or blow-holes. He pointed out the need of thoroughly deoxidizing the steel; soundness presupposes deoxidation.

Mr. Max H. Wickhorst, engineer of tests of the Rail Committee of the American Railway Engineering Association divided rail failures into head failures (split heads), base failures, and "broken rails" (square or angular breaks). The first class—that of head failures—makes up one-half of all the rail failures and is due to the interior condition of the ingots from which the rail was made. Segregation is the source of the trouble. Segregation must be distinguished very sharply from piping. Silicon, titanium, aluminium reduce segregation. When they are used, the ingot shows a deep pipe, but less spongy condition. The origin of split heads of rails is not piping, but the condition of segregation. We need not be so much afraid of the pipe itself, but rather of segregation. If this is avoided 50 per cent of all the rail trouble is solved.

Mr. L. E. Howard, of the Simonds Manufacturing Company, described a method of his own used successfully by his company in crucible steel practice. The process employs essentially lateral compression. Two sample ingots showing the successful suppression of piping were exhibited. In reply to question three he pointed out that his method was a commercial success, as the average cost including everything is less than 3 per cent of the cost of the ingots. For ingots of larger size his method must be somewhat modified.

Mr. J. E. Sague, engineer of the Public Service Commission of the Second District, summarized the chief results of the report which he had made as a result of the inquiry into the limits of the rail failure proposition. He found that from the accident standpoint the failure of rails is very much exaggerated in newspaper discussions. Nevertheless, as it really is, it is a grave proposition and a great deal more

knowledge is necessary on the subject. As far as the steel maker is concerned, the start must be made at the ingot. One cannot expect sound rails, except when starting with sound ingots. But the railroads have also their own problems to solve. It is a great advance that the fact has been recognized that a rush from New York to Chicago in 18 hours is not safe under any and all conditions of weather. To have their trains arrive on time, not to rush them through in the shortest possible time, should be the ideal of railway managers. In the whole there is no reason for despair.

Mr. Henry D. Hibbard, of Plainfield, N. J., commented at some length on the Hadfield process and then made some remarks on segregation. The problem is probably more serious than usually imagined. The segregated material heretofore used for analysis is probably always diluted by admixtures.

Mr. James E. Howard pointed out that in the first few passes there is evidence of elimination of the possibility of pipes and blow-holes. But what cannot be eliminated at all are particles of slag, etc., disseminated through the steel. Against them there is no correction. It is most important that these minute slag inclusions should be avoided.

Mr. W. C. Cushing, of the Pennsylvania R. R. Co., said that question (2), whether it would be desirable to increase the amount of metal arbitrarily cropped off the top of an ingot, should surely be answered in the negative. Progressive discard upon examination of soundness of the ingots seems to be the best at present. As to the statistics of rail failures in the United States head failures are 60 to 70 per cent of the total failures, web failures 5 per cent, base failures 5 to 10 per cent, broken rails 20 to 25 per cent. The rail failure statistics of the United States includes all these failures, not simply broken rails, as European critics have wrongly assumed. It is the broken rail that gives the most trouble and thought. Mr. Cushing asked finally as to the relative efficiency of aluminium, silicon and titanium as deoxidizers.

Mr. G. H. Clamer, of the Ajax Metal Company, described an electric device based on the principle that if the top of the ingot is kept hot, no piping will take place. There are different methods possible for keeping the top hot. The device described by him makes use of the pinch effect (as in the Hering electric furnace). It consists chiefly of a magnesia block with a cavity and a narrow channel connecting this cavity with the outside. This block is placed into the metal at the top of the ingot so that the steel enters the block through the channel. By providing suitable steel electrodes with connections to a small transformer an electric current is passed through the metal, the pinch effect is produced in the channel, and a heating and circulating effect is obtained as in the Hering steel furnace.

Dr. A. S. Cushman, of Washington, D. C., spoke on the extended investigation which is to be carried out under his direction by the American Rolling Mill Company. No final results have yet been obtained, but a special ingot splitting apparatus has just been installed and with this facility it is hoped to get decisive results as to the effectiveness of the different methods.

Mr. N. Petinot, of the Titanium Alloys Manufacturing Company, first dealt on the significance of some of the segregation analyses mentioned in Dr. Dudley's paper. He then discussed the relative efficiency of aluminium, silicon and titanium as deoxidizers. While all three remove oxygen, the results of the deoxidizing reactions are different. Aluminium is oxidized to alumina and the alumina particles are liable to remain disseminated in the steel and cause trouble. Silicon forms silica, silicides, silicates. With titanium on the other hand titanate acid is formed which rises to the top and is eliminated. Silicon should be used moderately and should be followed by titanium which will not only complete the deoxidation but will remove the iron silicates. Dr. Dudley, later in the discussion, confirmed the efficiency of titanium as a deoxidizer.

Prof. Henry M. Howe discussed the reason why pipes are formed at all and emphasized that it was all important to let solidification of the top lag behind the solidification of the bottom. Millmen have not gone to the limit in this respect. To cast from the bottom is worst in this respect, to cast from the top with the large end up is the best. There are special methods of heating the top, but they represent necessarily some expense. But the same result may be obtained by purely administrative methods. As to pouring, it may be asked whether it is possible in pouring to fill first only part of the mold and then come back a little later and pour again and fill up the top. As to the possibility of this, millmen may say no; but if yes as answer is at all possible, it would be well worth while trying it. In the whole purely administrative methods may perhaps prove just as good as special and more expensive processes.

Mr. K. W. Zimmerschied, of the General Motors Company, divided the troubles which remain in the steel from the ingot, into pipes, segregation, and enclosures of non-metallic material. Pipes cause the smallest trouble, the enclosures of non-metallic material (oxides of iron and manganese, alumina, etc.) cause the most trouble. To avoid them is a more important problem than that of piping. If aluminium is used as a deoxidizer properly, the alumina, though solid, will probably rise as quickly to the top as titanium oxide on account of its much lighter weight.

Mr. Zimmerschied then took up the four questions which formed the basis of discussion. Question (1), whether the present method of getting rid of the pipe by cropping is a safe and reliable device for making sound steel, he answered with yes. In reply to question (2), whether it was desirable to increase the amount of discard arbitrarily, he said that this was out of the question. Question (3), whether it is commercially practicable to make ingots without pipes and blow-holes, he answered with yes and added that the amount of the additional cost was a commercial consideration depending on the circumstances. As to question (4), what process for making pipeless ingots seemed the most promising of commercial success, he said that the cure must be a continuous one and must progress from the bottom to the top. In conclusion he made what he called a "wild suggestion": with our advance in cutting methods, why not split every ingot before you roll it?

Mr. Henry Hess, of the Hess-Bright Manufacturing Company, described some experiments made by himself with a method used formerly in a similar way by Professors Howe and Stoughton. To look into the interior of an ingot while in course of solidification, he made solidification experiments with resin in test tubes. He could form pipes of any form and at any place by changing the method of heating the test tube. Where you heat it, and where you heat it last, there you get the pipe.

After conclusion of the discussion lunch was served in an adjoining room.

Wednesday Afternoon Session

Mr. John Birkinbine presided during the afternoon session. A paper by Mr. James R. Finley, of New York City, on the "valuation of iron mines" was read, in the absence of the author, by Mr. Stoughton and discussed by Messrs. Birkinbine, Norris, and Parker.

Dr. Albert Sauveur, of Harvard University, then presented a paper entitled "Notes on Cast Iron," with special reference to the recent paper of Mr. J. E. Johnson, Jr. We reserve an abstract of Dr. Sauveur's paper for a future issue. The paper was discussed by Dr. Howe and Dr. Moldenke.

Dr. Henry M. Howe, of Columbia University, then presented his paper, "Why does lag increase with the temperature from which cooling starts?" We reserve an abstract for a future issue.

Sintering Iron Bearing Materials

A paper by Mr. B. G. Klugh, of Birdsboro, Pa., on "The microstructure of sintered iron-bearing materials" was read, in

the absence of the author, by Mr. Arthur S. Dwight and illustrated by a long series of lantern slides of structural photographs. The chief results of the paper are as follows:

The permeability of the cell-wall of a sintered product varies, inversely, as the degree of fusion to which it has been subjected.

In a product of complete fusion, the silica present combines with its equivalent of iron oxide to form a perfect glass which, from its greater fluidity, envelops and seals up the remaining iron oxide from the action of gases.

Conversely of the foregoing conclusion, in the product of the lowest degree of fusion, the iron oxide and slag-forming materials as a unit are bonded together by incipient fusion, leaving the predominant iron oxide, free and vulnerable to the action of the gases, in the highest degree attainable in solid products.

The above salient facts show that the Dwight and Lloyd products when properly made, do possess those properties which distinguish it from the products of other sintering processes or agglomerating methods, by freedom from those constituents, to which scouring action in the blast furnace is attributed.

New Design of Open-Hearth Steel Furnace Using Producer Gas

Mr. Herbert F. Miller, Jr., of Verona, Pa., in his paper on this subject, expressed the opinion that the gas and brick costs of open-hearth furnaces using producer gas could be greatly decreased by a change in the design of the port, which would materially reduce the first cost of the furnace, the rebuilding cost, and the repair cost.

He first outlined the defect of the present design and then described two new designs, both of which involve the same principle of putting a single air uptake directly back of the gas uptake as is shown in Figs. 8 and 9.

In Fig. 8 the port has about one-third of the width of the hearth, whereas the present type has the full width. The roof of the port dips sharply, compressing the air down on the gas; and the roof of the hearth is gradually dropped to meet it.

The gas-port arch is retained although flattened to the width of the port. The end C is from 6 to 10 ft. from the edge of the bath.

There is a small seal-door on each side of the port at C to permit the repair of the port or the removal of the brick which might fall from the gas-port arch.

The point C will move back slower than in the present type, because it will be less exposed to the flame, which will always be under the air.

The possible defect of this design is that when the gas-port arch has become short, combustion will commence too far from the bath, and make a high gas cost. Another objection is, that this design could not be applied to existing furnaces because their ports would have to be lengthened considerably. If the flame in the above design can always be controlled, even when the gas-port arch has burned back to the gas uptake, why have a gas-port arch at all?

The design shown in Fig. 9 is the solution. Here it will be seen that the furnace is similar in outline to the present open-port type in use at Homestead. The air uptake A is back of the gas uptake, from which it is separated by a substantial wall. The gas uptake, which is as wide as the port, is from 6 to 10 ft. from the bath. The port is open and smooth, narrow and low. The small cross section of this port assures a speedy union of the preheated air and gas, resulting in a short flame of intense heat, which will be under control throughout the run.

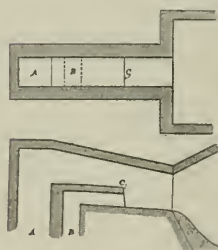


FIG. 8.—PART OF O. H. FURNACE USING PRODUCER GAS

Some may think that the gas rising at right angles to the air will make a high flame. If the port is low enough this will not occur. The author has put a stream of air having a pressure of 70 lb. per sq. in. under a natural gas flame, and found that there was very little deflection of the flame.

This design could be applied to the present type of producer-gas furnaces. The only alteration of importance would

be a reversal of the gas checkers from the outside to the inside position.

In building a new furnace, a great saving of brick would be made in the ports, as can be readily seen.

Repairs to the dividing wall can be made through small seals by the side of the gas port. The flame will always be under control and the furnace should slow up only because of dirty checkers and insufficient draft. Natural gas could be used if desired by introducing it as usual at the sides and in

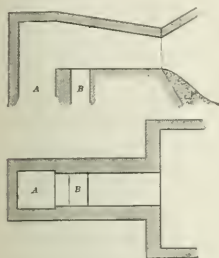


FIG. 9.—PART OF O. H. FURNACE USING PRODUCER GAS

front of the uptake *G*. In the author's judgment, the adoption of this design would do away with much of the trouble and expense attending the present type of producer-gas furnace and greatly decrease the operating cost.

In the discussion Mr. Hibbard questioned whether with this design a good spreading of the flame over the hearth could be obtained. Mr. Miller replied that the design worked well with natural gas.

The last paper on the program was presented by Mr. F. L. Grammer, of Leesburg, Va., on "Blast furnace slag analyses for 24 hours."

The meeting then adjourned.

Notes on Chemistry and Metallurgy in Great Britain

Institution of Mining and Metallurgy

At the third general meeting of this Institution, held at the rooms of the Geological Society, Burlington House, on Dec. 19, three papers of unusual interest were discussed together, as all bearing upon questions connected with tin metallurgy. These were Mr. W. Fischer Wilkinson's "On the Dressing of Tin Ores in Cornwall," Mr. H. J. B. Rawlin's "Notes on the Direct Volumetric Determination of Tin," and Mr. R. T. Hancock's "Notes of the Valuation of Nigerian Tin Concentrates."

Mr. Holloway, who opened the discussion, pointed out that as Mr. Rawlin's paper dealt entirely with the analysis of metallic tin, it might at first sight appear to be more a paper for an analytical society than for themselves; but that it was important in connection with the chemical testing of the metallic tin obtained by such methods as the old Cornish, or culm method, or the more modern cyanide method of determining tin in black tin concentrates. Mr. Wilkinson's paper related entirely to Cornish tin dressing. Mr. Holloway was sorry to see heavy Californian stamps so largely used in Cornwall. The recovery of the tin was rendered less direct by the use of heavy stamps.

The discussion generally treated of the most correct method of estimating the recovery, and the applications of the chemical assay and the "vanning" test. Mr. E. S. King said that he found the only way to get at the correct extraction was by the chemical assay of the heads and the output of pure black tin. The tin account was kept in just the same way as a gold account; on one side was entered so many tons of ore containing so many tons of black tin; against that they had the amount of pure black tin sold. This was at Carn Brea.

With reference to the vanning assay, he said the most obvious explanation of tin which escaped from the vanning and was

caught by the wet assay was the smallness of grain of cassiterite in the tinstone. It should be remembered, also, that there were tin-carrying minerals other than cassiterite.

Mr. Josiah Paull estimated the recovery at the mine with which he was connected at 78 per cent to 80 per cent. Mr. Wilkinson had given the average for Cornwall as something like 50 per cent to 55 per cent. His results were from samples of pulp from the screens taken practically every hour throughout the twenty-four. Afterwards replying to Mr. Flowers, Mr. Paull modified his percentage by a small allowance for wet weight.

Mr. H. L. Twite, in a written communication, pointed out the importance of even 1 per cent in recovery. He had known samples sent to two different assayers and the difference in the assays returned was 0.3 per cent, which was repeated on several assays.

Mr. Amos Treloar gave the recovery for the Carn Brea and Tincroft plant, one of the oldest and most out of date in Cornwall, as 73.7 per cent. Mr. Frank Merricks inquired if Mr. Paull, in his figure of 78 per cent to 80 per cent of recovery at South Crofty, meant a 62 per cent extraction?

The irregularity of results quoted and the disagreement between the figures obtained on assay and test show how extremely difficult it is to obtain a standard method of sampling and testing which would admit of exact comparison.

Oil Fuel

An example of the growing interest in oil fuel in British scientific circles is afforded by the lectures on the subject delivered at the Society of Arts by Prof. Vivian B. Lewes (Cantor Lectures). On the question of oil fuel for power purposes, the professor noted that the method of obtaining the heat from the fuel had evolved into spraying the oil into the furnace, with a swiveling motion to the cloud formed, thus retarding the passage through the furnace and combustion space, so as to give sufficient time for complete combustion. When used as a fuel for raising steam, petroleum might be taken as having an evaporative power, weight for weight, 50 per cent higher than Welsh steam coal. It was known that 0.5 lb. of water evaporated was the best duty which could be obtained per pound of coal, which is a difference of 4.5 lb. from the calculated amount, due to losses of heat. Allowing the same ratio of loss for the oil fuel, the evaporative duty should be 13.2 lb. in practice, or even higher, according to the type of boiler and length of time under steam.

With regard to the internal-combustion engines, taking the highly efficient Diesel engine as an example, in spite of its only consuming one-third of the weight of fuel used by the best over-type superheated condensing steam plant, yet for a given power with coal at 18/- per ton and oil at 42/- per ton, the annual cost of running the steam plant would be less than that of the Diesel engine when all items of expense were taken into consideration. On surface combustion, Prof. W. A. Bone had found that when a mixture of a combustible gas with air was forced under pressure through a porous diaphragm of asbestos and fireclay, if the mixture were ignited as it leaked through the porous mass, it could not explode, but burnt on the surface of the diaphragm and quickly raised it to a very high degree of incandescence. This had been adapted for practical work with a boiler, and a very high efficiency had been obtained.

Steel Metallurgy

"Recent Advances in Steel Metallurgy" was the subject of a lecture delivered at the Royal Institution by Prof. J. O. Arnold, of Sheffield, in January. After noting the improvements in steel for cutting tools introduced by Mushet, and other modifications of the same process, involving the use of tungsten, the lecturer goes on to the introduction of vanadium steels in 1800-1902. The addition of about 0.6 per cent of vanadium to a pure carbon steel containing about 1 per cent of carbon raised the yield point from 35 to 65 tons per square inch, and the maximum stress from 60 to 80 tons, the elongation being 7 per cent, and the reduction of area 8 per cent.

According to researches made by the lecturer and Prof. A. A. Read, of the University of Wales, vanadium did not seem to form a double carbide with iron. It gradually wrested the carbon from the carbide of iron till when about 5 per cent of vanadium was present Fe_3C could not exist, and only a vanadium carbide V_4C containing 15 per cent of carbon was present, this constituent being constant at any rate in tool steels containing up to 14 per cent of vanadium.

Strains in Metals

Attention is called by Mr. T. Vaughan Hughes, A. R. S. M., in the *Electrical Review* to strains set up in the manufacture of copper and other metallic parts, such as condenser tubes. He advocates that at least two causes of failure of metals and alloys under the control of the makers should be removed, namely (a) uneven heating appliances and (b) antiquated heat treatment in chemically and physically active atmospheres.

Oil Explosions in Transformers and Switches

Two remarkable accidents, accompanied by fatal consequences, recently occurred at a main transformer station of the Victoria Falls & Transvaal Power Company. The station takes current at 40,000 volts from the trunk lines and transforms down to 20,000 volts with oil-filled transformers, the oil being cooled by constant circulation of cold water through pipes. "Expansion tanks" are attached to and above the transformers to ensure their being always filled with oil and to receive any outflow resulting from increase of volume of the oil when its temperature rises.

While a thunderstorm was in progress there was a "surge" on the main which broke down one transformer with the result that it was cut out. Three attendants examined the transformer and one of them, wishing to see the level of the oil in the tank, held a lighted match to the aperture at the top, when a violent explosion occurred which killed one man and severely injured the two others. It appears, too, that instantly after the explosion a large quantity of oil was ejected from the tank and set the transformer in a terrific mass of flames.

About a week later another explosion took place at the same station in a 20,000-volt oil switch controlling some low-voltage apparatus. The cause is attributed to a "surge" on the 20,000-volt lines which discharged across the busbars and made the oil switch trip; and the instantaneous effect was that a strong fireproof dividing door was burst open, a man at a telephone in the adjacent room was enveloped by the vast mass of flame and died soon afterward and others were seriously burned.

The commonly accepted explanation of the first of these explosions is that arcing, resulting from a "short" in the transformer, decomposed the oil with liberation of hydrogen or gaseous hydrocarbons, such as methane, or perhaps a mixture of these gases, which, rising into the expansion tank, formed an explosive mixture with the air therein. A similar explanation has also been offered for switch explosions. But in both cases the intense and rapid combustion of the ejected oil—which would, of course, originally have a high flashpoint, probably not lower than 160 deg. C.—suggests that there may also have been some action analogous to "cracking" which split up the higher liquid paraffins into paraffins of considerably lower density and flashpoint together with some olefins.

A chemical examination of the residual oil might throw some light on the question, which manifestly deserves more thorough investigation than it has yet received.

Detection and Estimation of Carbon Monoxide

Two methods of determining small percentages of carbon monoxide in air have recently been communicated to the Académie des Sciences of Paris. MM. A. Lévey and A. Pécoul adopted a suggestion of M. Armand Gautier and have found that the reduction of iodic acid by carbon monoxide between 60 deg and 80 deg. C. affords a means of detecting one part of the gas in 200,000 parts of air and yields accurate quantitative results. Three liters of air are passed from the aspirator into a vessel containing iodic acid and maintained at a temperature of about 70 deg. C., the iodine liberated is absorbed by chloroform, and its amount is determined colori-

metrically against solutions of iodine, of known strength, in chloroform. The authors have successfully applied this process to the examination of air in Paris schools, hospitals and railways.

The other method, described by M. Guasco, is an application of the well-known fact that platinum black absorbs combustible gases, and especially carbon monoxide, with evolution of heat. It differs from earlier proposals to utilize this phenomenon for the estimation of carbon monoxide in the employment of a Leslie differential thermometer, one bulb of which is coated with platinum black, while both bulbs are protected by porous envelopes preventing direct access of air, but permitting osmotic penetration.

It is obvious that M. Guasco's apparatus can only be relied on when the air is ascertained to be free from combustible gases other than carbon monoxide. An apparatus on the same principle was devised by Messrs. J. Pitkin and T. Niblett in 1889; but they used two ordinary sensitive thermometers, side by side, one having the bulb coated with platinum black, and the instrument was not sufficiently sensitive for the estimation of less than about 1/10 per cent of carbon monoxide in air.

A New Method of Manufacturing Gold Leaf

A London firm has introduced a novel, patented, process of making gold leaf even thinner than the best leaf produced in the usual way, although the new leaf really consists of three metallic films. A wheel of aluminium, some 5 ft. in diameter and $3\frac{1}{2}$ in. wide, has its periphery highly polished and then evenly coated with a soluble gummy material which, when nearly dry, receives a coating of very fine metallic powder. This metallic film is then polished and is electrolytically coated with a film of nickel by rotating the wheel with its lowest point making contact with the surface of a suitable solution of a salt of nickel containing the anode, while the rim of the wheel is the cathode. After washing, and again polishing if necessary, a film of gold is in the same way deposited on the nickel.

The composite film is removed by making one cut across its width, immersing the rim at this point in a liquid solvent of the adhesive substance and slowly rotating the wheel. As the film falls away a paper band, the motion of which synchronizes with the peripheral speed of the wheel, carries it off to be cut into slips and made up into books.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the year 1912 show considerable increases in both imports and exports as compared with 1911, except imports of new ships. Iron and steel, including manufactures, were imported to the value of £12,070,862, an increase of £1,837,008; and were exported to the value of £4,862,918, an increase of £4,808,626. Imports in other metals, including manufactures, amounted to £31,100,808, an increase of £3,618,654; and exports reached £12,209,149, an increase of £1,276,613.

In electrical goods, the imports were £1,457,646, a rise of £22,154; and exports went to £6,369,877, or £1,550,503 more than for the previous year.

Imports of machinery are put at £6,820,744, an increase of £1,052,682; and the exports at £3,161,772, an increase of £2,201,104. The value of imports of new ships was only £33,654, showing a decrease of £30,830; but exports touched £7,031,809, an improvement of £1,368,784.

Taking the figures for December alone, imports of iron and steel increased £248,235 and exports rose £628,433; imports and exports of other metals went up £313,718 and £154,136 respectively; imports of electrical goods declined £1,117, but exports rose by £102,000; imports of machinery increased £160,001 and exports were £62,431 higher; new ships showed a drop of £500 in imports, but exports improved by £20,501.

Market Prices—January, 1913

Copper opened at £76-13 and declined slowly till the 8th (76) when it dropped sharply, reaching 75 on the 14th, and after short recovery to £71, 0/8 on the 20th. Improved to £71 on the 23rd, but again dropped, selling 69-0 on the 25th and closing 68-15.

Tin opened at £220.10 and showed only slight falling off to the 8th, then became more jerky and touched £227.15 on the 14th, and again declined sharply on the 18th, reaching £226 on the 21st. Has since risen, closing £230.

Cleveland opened 67/5, rose by the 7th to 68/- and has since been weak, being under 65/- on the 17th and down to 65/3 on the 24th. Since unsteady, but keeping near this price, it closes 65/3.

Hematite opened 82/- and remained firm till the 17th when it dropped away to 80/5, since recovering slightly to 81/-; closes 81/3.

Scotch Pig has been in sympathy with Cleveland, opening 74/- and dropping to 70/10 by the 17th. It closes a little better at 71/3.

Lead (English) opened £187.6 and has shown unusual variations. It was £18.18 on the 8th, and declined thence to £17 on the 12th, recovering to £17.10 by the 23rd, but afterwards weak; closing £16.17.6.

Aluminium	£80. 0.0
Alum, lump, loose, ton	5.15.0
Antimony, black, sulphide, powder, ton	21. 0.0
Borax, British, refined crystal	17. 0.0
Copper ore, 10 to 25 per cent per unit	12.10 1/2 to 13.4 1/2
Copper sulphate, ton	25.10.0
Carbolic acid, liquid, 97-99 per cent, per gal	1.9
Creosote, ordinary good liquid, per gal	0. 0.3
Caustic soda, ash, 48 per cent, ordinary, ton	5.10.0
Hydrochloric acid, cwt	5.0
India rubber, Para	4.4
Naphtha, solvent, 60 per cent @ 160 deg. C, per gal	1.1
Petroleum, Russian, spot, per gal	8
Quicksilver, bottle	7.15.0
Sal ammoniac, lump, firsts, delivered U. K.	44. 0.0
Sulphate of ammonia, f.o.b. Liverpool	14. 7.6
Sulphur, recovered, ton	5. 5.0
Shellac, cwt	3.12.6
Platinum, ounce	9. 5.0
Zinc, V. M., f.o.b. Antwerp	29.15.0
Tin, ore, 70 per cent, ton	£148 to 150. 0.0

Differences

<i>Higher.</i>		<i>Lower.</i>	
Aluminium	£1. 0.0	Copper ore	£0. 0.7 ¹ / ₂
Carbolic acid	4	India rubber	23 ³ / ₄
Quicksilver	6.6	Naphtha	1
Sulphate of ammonia	8.0	Shellac	11.6
Tin	1.10.0	Zinc	10.0
		Copper	7.5
		Cleveland iron	2.6
		Scotch pig	2.6
		Hæmatite	1.0
		Lead	1.12.6

Recent Chemical and Metallurgical Patents

Gold and Silver

Sand-Slime Classifier.—The separate treatment of sand and slime products in cyanidation has created a demand for efficient means of separating these materials. A recently patented device for this purpose is shown in Figs. 1 and 2, being the invention of MR. PHILIP ARMIT, of Denver, Colo. The classifier consists of an inclined trough E, in which are placed two spiral conveyors I which rotate upwardly and inwardly. The trough may be divided into three portions, namely, B, in which settlement of sand occurs C where the sand is unwatered, and M, a filtering portion for withdrawing additional moisture from the sand. The inclination of the trough will vary from ten to twenty-five degrees, but for ordinary silicious ores probably will be about fifteen degrees. A weir G consists of a level strip over which water and slime may flow into a launder G. A feed-box is shown at A, provided with openings a', through which the mixture of sand and slime is introduced into the trough.

In operation the pulp is fed through the box A. The sand settles and is worked upwardly by the spiral conveyors, while the slime remains in suspension aided by the action of the conveyors, and overflows at G. The inventor states that he has devised the weir so that the overflow from a feed of six tons per hour will all pass a 150-mesh screen. The action of the conveyors results in building a bed of sand as shown in Fig. 2, and the slime and water draining from the sand bed finds its way into the side launders K' and runs back to the settling portion of the trough. The filtering portion consists

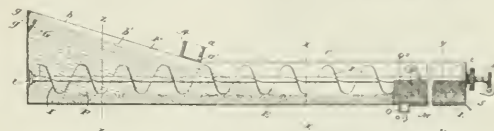


FIG. 1.—SAND-SLIME CLASSIFIER.

of angular slats L arranged transversely of the trough. If desired, a spray of water may be made to play on the sand as it passes over the filtering portion. In a modified form of classifier, the settling portion is circular, thus giving a larger weir. (1,044,844-5, Nov. 19, 1912.)

Slime Filter.—Various applications of a porous mineral septum in agitating and filtering mineral slime have been proposed by the advocates of the Just process, silica sponge being the material used. In patents recently granted to MR. W. W. RONAHER, of Rochester, N. Y., assigned to the Just Process Co., Syracuse, N. Y., it is proposed to use silica sponge brick as a facing on a revolving vacuum filter, as shown in Figs. 3 and 4 (p. 1001). The device consists of a drum 2 revolving in a tank 11 containing the slime to be filtered. The drum is covered on its sides with a porous mineral septum 6. The filter revolves on a hollow shaft actuated by gearing 12, 13. Vacuum is applied to the inner portion of the drum through pipe c, and causes a cake of slime to form on the faces of the drum, while cyanide solution is drawn through the septum and removed for precipitation. As the drum revolves and the cake is removed from the tank, it is removed from the filtering surface by means of scrapers 15 and transferred to a mixer 17, in which it is agitated with more solution or water and sent to a second similar filter. (1,051,160-1, Jan. 21, 1913.)

Copper, Lead and Zinc

Dry Chlorination of Mixed Sulphide Ores.—Among the recent efforts made to evolve a suitable process for treating complex sulphide ores, is one disclosed in a patent granted to MR. JOHN L. MAUM, of Denver, Colo. The principal steps

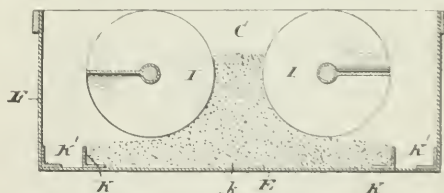


FIG. 2.—CROSS-SECTIONAL VIEW OF SAND-SLIME CLASSIFIER.

outlined in the present patent are the following. Fully crushed ore is fed into one end of a tube mill where it comes in contact with dry chlorine gas admitted into the tube from the opposite end. The reaction between the sulphides and chlorine evolves heat, but the temperature is controlled either by varying the quantities of ore and chlorine admitted, or by admitting the necessary quantity of air from the outside. In either case the temperature is maintained below the point at which sulphur monochloride or plastic sulphur would be formed as products of the reaction, thus preventing the ballooning or caking of the ore, and permitting every particle of it to be

acted upon by the chlorine. This step in the process affects only a partial chloridization of the ore, so that some of the metals exist as chlorides in the lower state of oxidation.

The ore is then transferred to an agitator, where it is treated with water or mill solution containing free chlorine, whereby the chloridization of the metals is completed and all exist as chlorides in the high state of oxidation. These chlorides are soluble and pass into solution, while the gangue and sulphur remain in a solid, granular state. The mixture is then filtered, and the metals precipitated from solution in any suitable manner.

It may be added that the procedure disclosed in this patent does not represent the finished state of Mr. Malm's process, other features being contained in pending applications. (1,049,746, Jan. 7, 1913.)

Hydrometallurgical Process for Copper Ores.—A process patented by Mr. JOSEPH IRVING, of Salt Lake City, Utah, has for its object the treatment of copper ores containing precious metals. It consists essentially of the following steps: Treating a low-grade carbonate ore of copper with dilute sulphuric acid solution, injecting steam and mechanically agitating the mixture, drawing off the resulting solution, passing it through a filter of sand and pyrites, and precipitating it on iron. The precipitated solution, containing at this time ferrous sulphate, is oxidized by agitation with air and injection of steam, and used for the treatment of a sulphide ore of copper containing the precious metals. Sodium chloride is added also, and the mass agitated mechanically and with steam. The resulting solution, containing copper, gold and silver, is then filtered and passed over iron and zinc to precipitate the contained copper and precious metals. On reoxidizing the precipitated solution and adding the requisite quantity of sulphuric acid, it is ready to be used on a fresh lot of ore with addition of salt. (1,048,541, Dec. 31, 1912.)

Pyrometallurgy of Zinc-Lead Sulphides.—A method for treating mixed zinc-lead sulphide ores has been patented by Mr. C. A. L. W. WITTER, of Hamburg, Germany. The ore is first finely ground and roasted until it is practically free from sulphur. It is then briquetted with a reducing agent and flux, as coal and lime, and fed into one end of a reverberatory furnace. The furnace is fired with gas, and in such a manner that the feed end contains a neutral or preferably re-

with lead fume by reason of the conditions maintained in the furnace. (1,047,360, Dec. 17, 1912.)

Nickel-Copper Mattes.—A process for the separation of copper and nickel in high-grade or concentrated mattes, patented by Messrs. ALEXANDER McKECHNIE, of Birmingham, and FREDERIC G. BEASLEY, of Smethwick, Birmingham, England, is claimed to be simple and effective. The raw or partly roasted matter is digested with sulphuric or hydrochloric acid at a

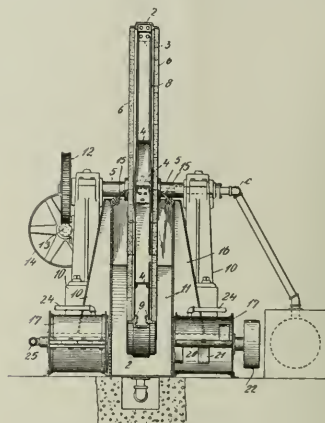


FIG. 4.—VACUUM SLIME FILTER

temperature of 100° C. in a closed air-tight vessel. Under these conditions of temperature and pressure the nickel is readily dissolved as nickel sulphate, when sulphuric acid is used, with evolution of hydrogen sulphide. The latter gas is utilized to the highest degree, under the conditions, for the precipitation of copper from solution as sulphide. Thus the single operation yields nickel in solution as sulphate and copper as a solid precipitate. Although the process can be applied to raw mattes, the inventors find that the action is accelerated by preliminary roasting. In the latter case, however, enough sulphide should be left to generate hydrogen sulphide for precipitation of copper. (1,047,825, Dec. 17, 1912.)

Rare Metals

Separating Uranium from Vanadium.—Solutions resulting from the treatment of carnotite contain both uranium and vanadium. The separation of these metals offers some obstacles, but may be accomplished readily according to a process patented by Mr. WARREN F. BLEECKER, of Canonsburg, Pa., and assigned by him to the Standard Chemical Co., of the same place.

The solution is first heated to about 90° C., and treated with heated sodium hydroxide in sufficient quantity to precipitate uranium as a mixture of sodium uranate and uranyl hydroxide. This precipitate carries some vanadium which must be separated by a further process. After filtering and washing the precipitate, it is dissolved in sulphuric acid. The solution is made slightly alkaline with sodium carbonate, and then electrolyzed with anodes of any desired metal, such as copper, nickel or iron. The vanadium will be precipitated as the vanadate of the anode metal, while the uranium will remain in solution and can be separated from the precipitate and converted into any desired form. (1,050,706, Jan. 21, 1913.)

Treatment of Radium-Bearing Ores.—A process supposed to be applicable to all known complex radium-bearing ores has been patented by Mr. SIDNEY RADCLIFF, of Bairnsdale, Victoria, Australia. It has for its object the separate recovery of the following products: (a) radium as radium and barium sulphates; (b) uranium as oxide or uranate; (c) acid earths such as tantalum, niobium, titanium, as oxides; and (d) rare earths such as cerium, thorium, lanthanum, didymium, etc., as oxides.

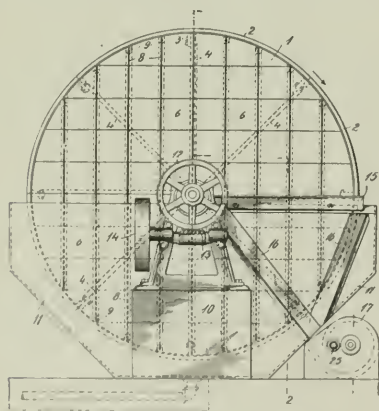


FIG. 3.—VACUUM SLIME FILTER

ducing atmosphere, while the opposite end is strongly oxidizing. The furnace contains a bath of slag on which the briquets fall. The metals in the briquets are reduced, the lead sinking through the slag bath, while the zinc vapor passes toward the opposite end of the furnace to the outlet. Here it is oxidized and subsequently caught as zinc oxide in flues and bags. The lead and slag may be tapped from the furnace periodically. The inventor claims that the zinc oxide will not be contaminated

The ore or concentrate is crushed to pass a 30 or 40-mesh screen. It is then fused in a reverberatory furnace with about $2\frac{1}{2}$ times its weight of acid sodium sulphate. When this charge has been fused, and while still fluid, sodium chloride to the extent of 10% to 15% of the weight of the ore is added and well rabbled. This causes complete decomposition and oxidation. The fused product is cooled and pulverized, after which it is agitated with warm water. Uranium, iron, the rare earths, and part of the acid earths will go into solution. Insoluble sulphates of radium, barium, lead and lime will remain in suspension. This turbid liquid is rapidly siphoned into a settler from which two products are obtained, namely, solution and fine slime.

The solution is treated with sodium carbonate, avoiding excess, which will precipitate iron, aluminium, chromium, uranium and the rare and acid earths. These are filtered from the solution and boiled with an excess of sodium carbonate which will cause the uranium to dissolve. The balance of the precipitate is treated with dilute sulphuric acid, whereupon all the slime except the acid earths will dissolve. These latter are filtered and ignited. The solution is treated with oxalic acid, which will precipitate the rare earths as oxalates, which are washed and ignited.

The fine slime contains most of the radium and is treated on the usual lines, and the crude radium and barium sulphates obtained in the usual way. (1,049,145, Dec. 31, 1912.)

Metallurgical Furnaces

Reverberatory for Copper Refining.—To prevent the disintegration and "floating up" of the bed of a reverberatory furnace, and to provide a furnace bed through which molten copper will not leak during the process of refining, are objects of improved furnace construction patented by Dr. J. B. F. HERRESHOFF, of New York City. In reverberatory furnaces hitherto used for copper refining the bed is composed of iron plates covered with fire brick on which is placed a covering of sand partly vitrified. A space beneath the plates constitutes a vault through which air may pass for the purpose of cooling the bed. It has been found that at times the molten copper will work its way down through this sand bed and cause the brick bottom to float, thus necessitating the temporary abandonment and repair of the furnace. The inventor has found that this disintegration of the bed is due to inefficient and imperfect cooling. As a consequence he proposes a construction whereby cooling may be accomplished uniformly, under suitable control, so that all parts of the bed may be kept cool enough to avoid the accumulation of molten copper below the brick bottom, and thus avoid disintegration and floating. In general, the invention may be said to consist of a series of pipes laid closely together in the bed of the furnace, both transversely and longitudinally, to carry air for the purpose of removing any desired amount of heat and thus keep the bed cool. Under this construction, any copper which may find its way through the bed will be cooled and solidified, and thus prevent further leakage. The heated air issuing from the cooling pipes may be used as preheated air at the combustion chamber of the furnace. (1,047,521, Dec. 17, 1912.)

Sintering Furnace.—In a patent issued to Mr. MILTON H. KAUFFMAN, of Denver, Colorado, the inventor describes the method of sintering or agglomerating fine mineral substances illustrated in Fig. 5. He makes use of removable pans 1 and 2, having perforated bottoms, which may be placed over an air box 6, which has a conduit 7 leading to an exhaust fan. His method of operation is to place a layer of material in the lower pan, with a layer of live coals in the upper pan, and set both over the air-box as shown. The exhaust fan draws air through the pans, as shown by the arrows, and ignites the layer of material in the lower pan. When ignition has proceeded far enough to continue without the aid of the live coals, the pans are removed and a third pan filled with mineral material is placed below the one previously ignited, and the pan containing burning fuel is removed. The exhaust continues to draw air through both pans with the result that ignition progresses from the upper pan to the lower, and the

material in the upper pan becomes completely sintered. The operation is then repeated, placing new material in a lower pan, while the partly sintered material is placed above, and the completely sintered material is discharged. (1,050,079, Jan. 7, 1913.)

Zinc Retort Furnace.—Novel construction for introducing gas and air into gas-fired furnaces, is the basis of a patent granted to Mr. ALEXANDRE FOLLIET, of Brussels, Belgium. In

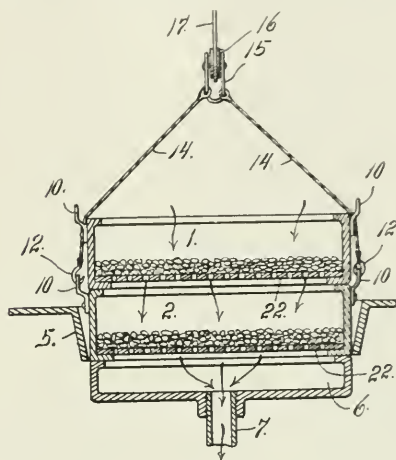


FIG. 5.—SINTERING FURNACE

furnaces of the type mentioned, it is essential that the distribution of heat be uniform both vertically and horizontally, and the object of the present invention is to improve the manner of mixing the gas and air used in combustion. Accordingly, the gas and air are introduced through separate, narrow, adjacent slits, constructed and arranged to cause the air and gas to enter the combustion chamber in the form of thin, vertically-directed streams or "slices," parallel and adjacent to each other. These slices will thus travel side by side, so that the contact is progressive, permitting of the maximum development of flame and radiation of heat, since the broadest sides of the slices are in contact. No opportunity is afforded for the mixture of air and gas prior to their introduction into the combustion chamber, thus avoiding improper mixtures and consequent formation of darts of flame even when air is forced by a blower. Each group of slits consists of one gas port between two air ports; or five ports or any other odd number may be used, the only requirement being that each gas stream should issue between two air streams. These groups of ports are disposed at suitable intervals along each half of the furnace. (1,049,569-70, Jan. 7, 1913.)

Chemical Industry

Manufacture of Cyanide.—The synthesis of hydrogen cyanide was accomplished by Berthelot by electrically heating a mixture of nitrogen, gaseous hydrocarbon and hydrogen, the latter being present in such excess that the quantity of hydrocarbon present did not exceed 10%. In this process the output was small, and not commercially profitable. Recently IGNAZ MOSICKI and CASIMIR JABLONSKI, of Freiburg, Switzerland, have discovered that in the synthesis of hydrogen cyanide from the above constituents, the output increases with an increase of the percentage of nitrogen in the mixture, the proportion of hydrocarbon remaining the same. The dilution of the hydrocarbon with hydrogen, instead of with nitrogen, greatly reduces the output.

A specific example of the application of the improved process is given as follows: The basic substance is a gas made from oil having the following approximate composition: Methane, 50.3%; heavy hydrocarbons, such as acetylene, 22.5%; hydrogen,

23.2°C., and nitrogen, 4%. This gas is mixed with pure nitrogen and hydrogen so that the mixture will contain 74 parts nitrogen, 10 parts hydrocarbon and 16 parts hydrogen. The mixture is preheated and treated in an electric furnace having a revolving high-tension flame. In this a part of the gas is heated to a temperature between 2500 and 3500 degrees Centigrade, and the elements form hydrogen cyanide. That part of the gas which was not highly heated in the flame is not decomposed, and issues with the hydrogen cyanide, cooling the latter and preventing its decomposition. That part of the gas which did not enter into the reaction is now replenished with the necessary ingredients and again passed through the furnace. The hydrogen cyanide formed may be dissolved in caustic solution to form alkali cyanide. An intermediate reaction in the process causes some of the saturated hydrocarbons to change into unsaturated hydrocarbons, thus liberating a quantity of hydrogen which, if not removed, would accumulate to such an extent as to greatly reduce the output of hydrogen cyanide. This removal may be accomplished by any one of several methods, namely, by diffusion, by oxidation, by catalysis with air, as a result of which additional nitrogen will be added; by burning the hydrogen with a deficient quantity of oxygen, nitrogen being again added to the mixture; by oxidation with copper oxide at a low red heat. (1,050,078, Jan. 21, 1913.)

Manufacture of Phosphorus.—A process for producing phosphorus from mineral phosphates has been patented by Mr. FRANK S. WASHBURN, of Nashville, Tenn. The work of the inventor was based on the idea that a less volatile acid than phosphoric should displace the latter from its compounds, in a manner analogous to that in which sulphuric acid will displace hydrochloric and nitric from their compounds. After many experiments he found that it is necessary to mix a reducing agent with the phosphate and silica, and that the temperature of the electric furnace is required to cause the reaction to take place. When finely divided tricalcium phosphate, silica and carbon are subjected to the temperature of the electric furnace, the phosphorus is driven off rapidly and up to 90% of the whole. Sufficient air is brought in contact with the evolved phosphorus to form the pentoxide, which may be absorbed in any desired medium. (1,047,864, Dec. 17, 1912.)

Electric Storage Batteries

Preparation of Films or Flakes of Nickel.—In storage batteries of the Edison type, films or flakes of metallic nickel are used in the makeup of the positive electrodes. Mr. THOMAS A. EDISON, of Llewellyn Park, West Orange, New Jersey, has patented a method for producing these films, consisting in first making composite sheets of alternating thin layers of copper and nickel electrolytically deposited, cutting these sheets into strips and flakes of suitable size, and then dissolving the copper with a suitable solvent which will not affect the nickel. Such solvents patented by Mr. Edison are (1) a 25% solution of copper sulphate containing about 15 grams cupric chloride per liter. Other reducible haloids may be substituted for cupric chloride but they are not as desirable. (2) a solution containing 285 grams ammonium sulphate and 10 grams cupric chloride per liter. When the metallic films of copper and nickel are agitated with these solutions, preferably warm, the copper separates as basic sulphate and may be decanted with the solution, leaving the nickel flakes to be suitably washed. This process is preferable to a former method of Mr. Edison's, in which he used an ammoniacal solution of copper sulphate, with attendant loss of ammonia, and the difficulty of recovering the copper in a desirable condition. (1,050,629-30, Jan. 14, 1913.)

Gage numbers of wire have no significance unless the gage is named. Thus No. 5 gage varies in diameter from 0.181 to 0.220 in. in six different standard gages. A far better way of specifying sizes of wire is to give diameter in decimals of an inch. This method is accurate and eliminate possibilities of error in filling orders.

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver

Air-Lifts vs. Centrifugal Pumps.—At the all-sliming cyanide mill of the Cia. Minera Jesus Maria y Anexas, San Jose de Gracia, Sinaloa, Mexico, centrifugal pumps were used to elevate the mill discharge and the final slime product. In the *Colorado School of Mines Magazine*, January, 1913, Mr. W. B. RHOODES describes the troubles encountered with the pumps, and their replacement with air-lifts. The pumps wore badly and gave constant trouble and expense. Of the two air-lifts installed, one had a 2½-in. lift pipe, with 65 per cent submergence, giving an effective lift of 18 ft. The other had a 3½-in. pipe, 54 per cent submergence and 38-ft. lift. The former was used for the sandy tube mill discharge and the latter for slime. The sand lift has plugged at times, but by the use of high-pressure air it has been cleaned without serious difficulty. Air is used at a pressure of 25 lb. per square inch. The total cost of the air-lifts, including the sinking of a shaft, 5 ft. square by 40 ft. in depth, was only \$300. The cost of installing an old compressor was \$150. The saving which has been effected by the air-lifts is as follows:

Maintenance of centrifugal pumps, including cost of pumps, liners, shafts and mechanics' wages, per month	\$200.00
Saving of 20 per cent in horsepower, or 5 hp at \$1.25 per day, 30 days.....	187.50
Saving 12 hours per month, equivalent to 40 tons ore at \$25 per ton.....	1,000.00
Total monthly saving.....	\$1,387.50
Total annual saving	\$16,650.00

These two lifts have been working steadily for two years without interruption and have saved approximately \$30,000. The air consumption varies from 2½ to 3 cu. ft. for each cubic foot of sand or slime lifted, depending on the dilution of the pulp. For the slime lift of 38 ft. a minimum dilution of 5:1 was found necessary to give good results. The sand is lifted at a dilution of 3:1.

Cyaniding at the San Poil Mill, Republic, Washington.—Preliminary cyanide tests on the ore of the San Poil mine indicated possible extraction as high as 98 per cent when the ore was crushed to 200-mesh and treated with a 5-lb. cyanide solution. The strong solution was required on account of the silver in the ore. Mr. EDWARD C. MORSE describes the operation of the mill in the January, 1913, *Bulletin of the Amer. Inst. Min. Eng.* One of the interesting features of the work is the use of a Williams hammer pulverizer for crushing the ore to 3/16 or 1/4-in. size. This machine receives the run-of-mine ore. It is similar in operation to the Quenner dry-placer machine used in Mexico for breaking cement-gravel; it has been used by the government for making sand for concrete purposes at the Cello locks on the Columbia River. Its use in the San Poil mill is its first application to ore crushing.

The Williams hammer pulverizer consists of a central shaft carrying thirty-two 16-lb. hammers suspended by chains. The shaft revolves at 600 r.p.m. within a trommel built of grizzly bars 2 in. deep, with a 1-in. face, spaced 3/16 in. apart. The machine has shown a capacity of 15 tons per hour of run-of-mine ore. Sizing tests of the product have shown as high as 40 per cent passing 20-mesh, of which 65 per cent was finer than 10-mesh and 6 per cent as fine as 200-mesh. When the ore fed is wet the machine does about twice as much fine crushing as with dry material, but has lower capacity.

The crushed ore is sampled and run through a set of finishing rolls provided with carborundum blocks to prevent grooving of the rolls. The product is as fine as 8 or 10-mesh, and is sent to a tube mill, the discharge of which is classified by a Dorr classifier, and the sand returned for further grinding. The slime is thickened and agitated in a series of nine air lift agitators, 12 ft. diameter by 18 ft. high. The tanks are connected in series, and the material passes from one to the

TABLE I

Per cent.

Insoluble	55.35
Lead	14.61
Zinc	8.65
Iron	5.17
Sulphur	7.63
Peroxide of manganese	5.70
Lime	1.03
Alumina	0.57
Carbonic acid	1.05
99.76	
Silver	10.3 oz. per ton.

TABLE II

	Tons	ASSAY VALUE			PER CENT. OF CRUDE		
		Pb %	Ag oz.	Zn %	Pb %	Ag oz.	Zn %
Concentrate	529	68.4	35.1	4.0	71.1	50.5	7.5
Tailing	2052	3.9	5.4	9.6	15.7	29.4	69.8
Van tailing	292	10.0	12.4	13.8	5.7	9.8	14.11
Slime to dam	185	19.0	19.0	12.4	6.98	9.6	8.13
Slime to storage tank, 8 % H ₂ O	8	33.8	23.2	11.6	0.52	0.5	0.32
Mine Assay	3225	15.4	11.4	8.7	100.00	99.8	99.86

vision between quartz and rhodonite treatment also was adopted.

Power is supplied by a 175-hp high-speed engine and partly by a 250-kw generator. The consumption on the different machines is given below:

STEAM

	Speed	Tons per Hour	HORSEPOWER	
			Each	Total
2 Rolls 33 in. by 18 in.	24 r.p.m.	10.5	30	60
4 Elevators 12 in. by 6 in., 40 ft. centers	365 "	10.5	9	36
17 Wilfley tables	260 "	0.75	1	7
14 Vanners 3-ft. 4-ft.	240 "	0.5	1	7
1 Elevator 20 ft. centers	400 "	5.5	6	6
1 3-in. Centrifugal pump for lead	1200 "	0.5	6	6
1 5-in. Circulating water pump	1000 "	0.75	6	6
16 Revolving screens	10 "	5.25	1	2
1 jig	200 "	10.5	2	2
Lead conveyors	54 ft. p.m.	16	2	2
Tailings draining belt	35 "	2	2	2
Small elevator 6 ft. centers	350 "	2	2	2
4 Feeders		1	2	2
Total steam power				150

ELECTRIC

	HORSEPOWER		Tons	Total H.P.
	Max.	Av.		
1 SD Gates crusher	35	25	48	25
1 16-in. by 10-in. Blake crusher	25	15	40	15
1 Crude ore conveyor	15	12	48	12
2 Tube mills	35	25	10	50
8 Grinding pans	35	7	1	56
1 Vanner section	25	22	22	22
2 Pumps	16	16	32	32
1 14-in. conveyor	4	2.5	16	2.5
Total electric power				214.5

The average cost of steam power is 2 cents per hp-hour, and of electric power $1\frac{1}{2}$ cents per hp-hour. The mill is treating well over 3000 tons of crude ore per week of 144 hours.

Some points of interest in the mill are the following: All Wilfley tables are fitted with riffles ending in a special curve at the delivery end, determined by experiment, to separate the fine lead from the rhodonite. All launders are of $\frac{1}{4}$ -in. steel, parabolic in section. A 3-in. centrifugal pump with flooded suction and 2-in. delivery pipe, gives satisfaction in raising lead

concentrates at the rate of 1000 lb. per hour. The draining belt used to carry tailings from the mill is 24 in. wide and 50 ft. long. The first 20-ft. section is in the form of a semi-circular trough, with slight slope so that water will drain back. The belt then rises sharply at an angle of about 15 deg., and this, with the addition of a bumper, dewater the tailings sufficiently for use in underground filling. The moisture can thus be reduced to 5 per cent.

A minerals separation plant is now being erected to treat the ore after the jig lead has been extracted. The mill circuit will be broken at the table screens in both quartz and rhodonite sections, and from these all the undersize will run to the flotation plant. Old dump material also will be treated by flotation.

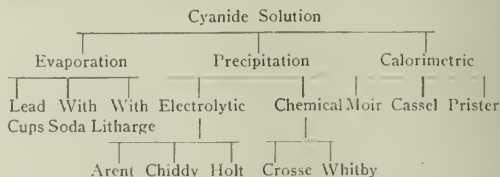
Chemical Analysis and Assaying

Variations in Assaying Gold Ore.—Tests were made at the Akaska-Treadwell mine to determine the error in assaying gold ores, and the degree of accuracy which would be considered satisfactory for mill-feed, concentrate and tailing. Mr. W. P. LASS gives the details of the experiment in the *Mining Magazine* for January, 1913. To summarize the results, the average variation of one assay from another on mine and mill samples, averaging \$3.46 per ton, was 41.3 per cent. The average variation on tailing samples, averaging 32.7 cents per ton, was 20.6 per cent. The average variation on concentrate samples, averaging \$68.79 per ton, was 7.6 per cent.

The author accounts for these variations by the fact that the gold occurs in Treadwell ores in free gold particles. There are at least 280,000 particles in one assay-ton of 40-mesh sand. If one of these particles be fine gold it would increase the gold content of the ore \$21.52 per ton. In one assay-ton of cubic particles of the same dimensions as the openings in a 100-mesh screen there are at least 5,348,811 particles; if one of them be fine gold it would increase the assay result \$1.12 per ton.

In ten different samples of concentrates the average first assay was \$68.81; the average second assay was \$68.77; the average variation was \$5.21, or 7.6 per cent figured on the average of twenty assays and the average variation. Twenty-eight mine and mill samples gave an average first assay of \$3.50; average second assay \$3.41; average variation, \$1.43, or 41.3 per cent figured on the average of fifty-six assays and the average variation. A point of particular interest is that though the average variation may be great, the average value of the samples remains approximately the same. It is a compensating variation. In ten tailing samples the average first assay was 32.5 cents per ton; the average second assay, 32.9 cents; the average variation being 6.7 cents, or 20.6 per cent figured on the average of the twenty assays and the average variation.

Assay of Cyanide Solutions.—The different schemes proposed for the assay of gold-bearing cyanide solutions are reviewed by Mr. L. J. WILMOTH in the February, 1913, *Merican Mining Journal*. The following diagram shows the classification of the different methods:



Of the evaporation methods, it is the opinion of the author that the first is antiquated and no longer in use; the second is seldom used, and is inferior to the third, which he regards as undoubtedly the best, but undesirable on account of the time needed.

Of electrolytic precipitation methods, there are those in which the gold is deposited by means of an extraneous current, and those in which the gold is deposited by means of an electric couple formed by two metals brought into the solution to be assayed. The first method is unimportant and has never been brought into commercial use. The second class of methods includes several which are of interest and importance. In

all these methods a given quantity of cyanide solution is acidified with sulphuric or hydrochloric acid; a solution of copper sulphate or lead acetate is then added, and the gold precipitated by introducing metallic zinc or aluminium to form the electric couple. Arant uses sulphuric acid, copper sulphate and aluminium. Chiddy's method requires hydrochloric acid, lead acetate and zinc shavings; and Holt proposes the use of hydrochloric acid, lead acetate and aluminium. The author believes the method of Chiddy to be the best; it is most extensively used. The use of copper solution is undesirable on account of the difficulty of eliminating the copper in the assay. Zinc is preferable to aluminium as it passes into solution more readily, is cheaper, more easily procured and just as effective.

In the second division of precipitation methods we find those of Crosse and Whitby. The former uses an excess of silver nitrate to precipitate a double gold-silver cyanide, while the latter relies on an acidulated solution of copper sulphate to throw down a double gold-copper cyanide. Crosse's method is regarded as costly, and has been superseded by others. Whitby's method is regarded as being in the same class with Chiddy's and the evaporation with litharge is being most extensively used.

The calorimetric methods have not been widely adopted. They all depend on the precipitation of gold from solution, redissolving it and then adding stannous chloride to form purple of Cassius. Moir first destroys the cyanide by addition of sodium peroxide. A few drops of lead acetate solution is added and a very small quantity of aluminium stirred in. This is not unlike Holt's electrolytic method. Prister precipitates the gold with a solution of copper sulphate and an alkaline sulphide, while Cassel uses potassium bromide and sulphuric acid.

Of all the methods proposed there are only three that are widely used. The evaporation method may be said to be in universal use; in America Chiddy's method is widely adopted, while Whitby's method is used in most of the mines of South Africa.

Assay of Zinc-Dust Precipitate.—In a paper presented before the American Metallurgical Society in June, 1912, Mr. H. R. LAYNG gives his procedure in the assay of zinc precipitate. The sample is ground on a bucking board to pass a 60-mesh screen. It is then thoroughly mixed and finally spread to a uniform depth of about $\frac{1}{2}$ in. By the use of a square-end spatula a sample is taken by dipping from various parts of the layer of dust. This is dried, cooled, and then mixed and spread again. Three samples of 0.1 A.T. each are then dipped by means of a spatula $1/16$ in. wide. Each sample is placed in a 10-gm. Battersea crucible, previously dried and containing 10 gm. test lead and $1\frac{1}{2}$ gm. of a mixture composed of one part each of potassium carbonate and sodium bicarbonate and two parts horax glass. The precipitate is mixed with this charge and the crucible is tapped to settle the mixture evenly.

Three fluxes are used in assaying the original charge, the resulting slag and the cupels. They are shown below:

	No. 1.	No. 2.	No. 3.
	gm.	gm.	gm.
Sodium bicarbonate	400	600	1 part
Potassium carbonate	400		60 gm.
Litharge	800	1200	$5\frac{1}{2}$ parts
Silica	250	200	$8\frac{1}{2}$ gm.
Charcoal	30	$12\frac{1}{2}$	1 gm.
Borax glass			3 gm.

After mixing the sample with lead as described above, 20 gm. of flux No. 1 is added, allowing it to wash around the sides of the crucible. This is leveled and covered with 20 gm. of flux No. 2. The general effect of these fluxes is to catch any gold or silver which might be carried to the top of the crucible. It will be noted that the charge is of such a character that lead is reduced in successive stages, and not all at once as in the ordinary assay. The crucibles are subject to low red heat until fused, then finished at a higher temperature for 10 minutes, and poured.

The buttons are cupelled carefully. The slag is assayed in the original crucibles, using 20 gm. of flux No. 2 and 10 gm. charcoal. The button resulting is cupelled separately, and the weight added to that of the original.

The cupels are assayed for absorption loss by grinding to 150-mesh, and assaying 10 gm. with 70 gm. of flux No. 3, together with 1 gm. charcoal and a borax cover. The author finds that his method gives higher results than other fusion methods, and compares favorably with the volumetric methods for silver. Assays check within 5 or 10 oz. on precipitate carrying 20,000 to 22,000 oz. silver.

Electrolysis

Chlorination of Benzene by Electrolysis.—A paper by R. G. Van Name and Carlton H. Maryott, in the February, 1913, issue of the *American Journal of Science* discusses the mechanism of the chlorination of benzene in the electrolytic cell. The investigation was carried out in the Kent Chemical Laboratory of Yale University. The chief results are as follows: (1) Electrolysis of benzene in a solution of lithium chloride in glacial acetic acid gave chlorinated benzenes with current yields, under favorable conditions, of 50 to 70 per cent. Both addition and substitution chlorine were present in the product, the latter predominating. There was, however, nothing to show that the effects were not due to the secondary action of chlorine previously set free by the current and dissolved in the liquid.

(2) Benzene dissolved in the same solution is really chlorinated by direct treatment with chlorine gas in the dark, yielding addition and substitution compounds in proportions which vary with the conditions.

(3) The addition of benzene to the acetic acid-lithium chloride solution during electrolysis raised the anode potential, and by an amount equal to or greater than that produced by a similar amount of carbon tetrachloride. This indicates that benzene has little or no depolarizing power toward chlorine, at least in this case. Phenol, added under similar conditions, produced a decided lowering of the anode potential.

(4) The rate of chlorination of benzene, when dissolved in the same medium saturated with chlorine, was measured at 10.2 deg. Simultaneous electrolysis did not accelerate the chlorination perceptibly under conditions where a 10 per cent current yield should have been evident. The current yield, if appreciable, was therefore below 10 per cent.

(5) The products of 4 contained addition and substitution chlorine in the ratio of about 2 to 3, this ratio increasing slowly as the reaction progressed. The reaction velocity, calculated on the assumption that monochlorobenzene and benzene hexachloride were the sole products, showed a steady rise, but the rate of chlorination of monochlorobenzene, separately determined, appeared to be sufficient to account for this effect.

(6) No positive evidence of strictly electrolytic (i. e., anodic) chlorination of benzene was obtained.

Non-spinning wire rope, the invention of Olof Tangring, an engineer of the American Steel & Wire Company, is made by first winding one series of wire strands over a core in one direction, and an outer series over the inner series as a core in the opposite direction. The wires composing the inner strands are twisted in the same direction with them, but in the outer strands the wires are laid in the opposite direction to them. The construction results in the counteraction of the tendency of the rope to spin when used in hoisting or lowering loads.

The melting point of cupric oxide has been investigated by R. E. Slade and F. D. Farrow (*Zeit. Electrochem.*, XVIII, 817), who find that the pure salt does not melt below 1148 deg. C. Decomposition occurs at this temperature, even with a pressure of 2.5 atmospheres, producing cuprous oxide which forms a solution with the cupric oxide solidifying at about 1081 deg. C. Under a pressure of one atmosphere the decomposition begins at 1000 deg. C., and the solution, containing over 50 per cent cuprous oxide, solidifies at about 1064 deg. C., which is the value given by Wöhler for the true melting point of copper oxide.

Chemistry in Russia in 1912

Russia as a chemical producer is a comparatively new comer. But there are some branches of the industry which have acquired what the Russian calls "citizen rights" in the country and which have developed in a very satisfactory manner. These are particularly soda and acids.

As far as the year 1912 is concerned the year has been a very satisfactory one, although more from the importers' point of view even than from the manufacturers'. The Russian field for foreign chemical products is an immense and a growing one, and will be worth more attention for many years to come, because the industry, as suggested above, is only partially developed in the country. Anyway during the year under review the factories were kept working all the time, and some of them were at times in straits to comply with the orders they received.

The market value of chemicals in the country did not vary very much last year, but there are some leading items that suffered changes in price, such as sulphuric and nitric acids, borax, prussiate of potash, and the like. Prussiates rose rather briskly, because of the stocks becoming exhausted both in Russia and Germany. There was also a substantial advance in lead, zinc, and copper compounds on account of the increase in the value of the respective metals, and the advance was maintained right to the end of the year, or nearly so. But the important Russian chemical products, as stated, are sodas and the value thereof had slightly declined in comparison with 1911 both for calcined and caustic. But it is objected and regretted that these particular goods should be so high in prices when they enter so largely into the economic life of the country. For example, the average cost of calcined soda last year was 1 rouble 16 copecks per pood; and the average cost of caustic soda was 2r. 47c. per pood.

Chemicals as a rule are protected by a tariff, and it is only when an agitation grows pretty strong that the quasi monopolists consent to make a slight reduction in order to avoid further attention being called to the fine business they are doing. But perhaps the most interesting development in the Russian chemical industry is the most recent one, namely, the alacrity with which the South Russian metallurgical works are adopting by-product coke ovens; the most interesting for the moment among the by-products is ammonia, because the Russian agriculturist is becoming a larger consumer every year of fertilizers—both home-made and imported.

But although this business has been catered to generously in the South, the first cost of the soda appears to the agriculturist to be so high as to discourage him from adopting it on a large scale; and the consequence has been that the production of sulphate of ammonia has not made the progress that was expected of it. It will be some time evidently before the Russian farmer will learn to appreciate the value of ammonium sulphate as a fertilizer. On the other hand, he is becoming a very large consumer of superphosphates. These were imported last year in far greater quantities than ever before—in fact, the imports increased at a much greater rate than the production at home, which is not unimportant. It is not that phosphate rock does not abound in Russia that causes him to buy the superphosphate abroad. Phosphate rock is plentiful enough in the country, but the difficulty is to get sulphuric acid cheap enough and the phosphate rock is not found in comfortable proximity to either native sulphur, of which there is a very little in the country, or pyrites.

In the Ural region there are large deposits of phosphate rock close enough to the pyrites, but unfortunately these particular deposits are poor and it becomes a question whether it will pay to work them even with convenient supply of acid. On the other hand, the Zerkovo (that is, district council) of Perm has decided to erect a superphosphate factory at public cost in order to supply the agricultural needs of the Perm and Viatka governments. The richest phosphate rock deposits in Russia are in the Southwest (Podolia) and they are largely exported into Austria unmanufactured. There does not ap-

pear to be sufficient enterprise amongst the industrial chemists of Russia to turn these deposits to account. The trouble is always, they will tell you, the question of acid.

The importation of fertilizers into Russia in 1912 during the first 10 months amounted to close on 24,000,000 poods, which compares with a little over 19,000,000p. in 1911 and close on 10,000,000p. in 1910, always during the first ten months.

It is hoped that in the course of time Russian industrial chemistry will be able to deal with the other byproducts of the coke oven, such as benzol and so on, for the purpose of establishing an independent color industry, but that time is probably still far off.

Besides superphosphates, the Russian farmer consumes large quantities of Chile saltpetre and German potash salts, both of which were imported last year in larger quantities than in any preceding year.

While Russia manufactures largely for her own use under the protection of a high tariff wall in several branches of chemistry, she can hardly be said to have an export trade in any of them, an exception being the products of wood distillation. This industry in Russia has entered into a very interesting phase. For many years back Russia has exported a turpentine of deplorably inferior quality which only gets a chance, say, on the English market when American turpentine becomes very high in price. A large customer for it is Germany where it is hardly ever turned to any account, however, until it has passed through the refineries there. But it is used in other countries, and perhaps in Germany, too, for mixing; but probably not to any large extent.

During the last year or two systematic experiments have been made for distilling the resin and it is claimed that this has been accomplished with great success, the prospect held in view being that in a very few years French, and particularly American, turpentine will have lost their significance for the Russian market; because the Russians have the raw material in abundance, there will be no difficulty in supplying their own needs independent of supplies from either America or France to which countries they have had to look in the past for turpentine of anything like quality. This applies also to rosin.

There is a point, however, that has to be considered in connection with this feature of Russian chemistry, namely, the question whether the Russian climate will be favorable to tapping the trees, as is done in France. Should it prove that the severity at times of the Russian weather does not discourage this preliminary operation in connection with the production of turpentine and rosin, it is quite likely that American and French goods under these headings will soon cease to be quoted on the Russian market.

As hinted in the first few lines although Russia has made enormous strides in the manufacture of a limited number of chemicals, if we take the whole field she is still so far behind in the race that chemical manufacturers in other countries may well cultivate the market in full confidence that it will be available for their goods for many years to come. Germany knows this and is doing an immense business with her neighbor over the border, not only in colors, in respect to which, of course, she leads the world, but also in other chemicals when other countries may easily become serious competitors.

St. Petersburg, Russia.

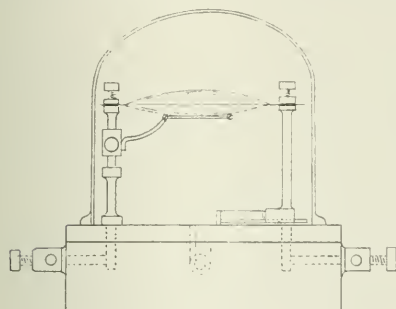
Six Chapman Rotary Gas Producers, built by the Chapman Engineering Company, of Mount Vernon, Ohio, are being installed at the open-hearth plant of the Bethlehem Steel Company at South Bethlehem, Pa. Three Chapman gas producers of the same type have been ordered by the Tremont Nail Company for its open hearth at West Wareham, Mass., and one by the Crucible Steel Company, of America for its plant at Syracuse, N. Y.

The consulting engineering firm of Mr. E. S. Lincoln, of Waltham, Mass., has been incorporated as E. S. Lincoln, Inc., and has moved its offices to the new laboratory now nearing completion at 129 Bacon Street, Waltham.

The Apophorometer

A paper by Prof. J. Joly in *Philosophical Magazine*, February, 1913, describes the "apophorometer" which is an instrument for enabling sublimates obtained from substances at high temperatures to be collected in their entirety and weighed on the chemical balance for purposes of chemical analysis, identification or research.

The apophorometer is a very simple instrument. A ribbon of this platinum, about 6 cm. in length and 4 or 5 mm. in width is stretched between two forceps which are provided with binding-screws so that a current can be sent through the ribbon, raising its temperature to any desired degree up to the melting point of platinum. One of the forceps is movable parallel to itself, and is so acted on by a light spring that the ribbon is kept stretched. The substance to be volatilized is placed in the form of powder upon the ribbon or hob. Beneath the ribbon a watch-glass is held in position by a support which can be raised or lowered or rotated to one side. When this watch-glass is in contact with the ribbon a second watch-glass is placed on the lower one. The ribbon is thus enclosed between the glasses for nearly its entire length. Overall a receiver fits enabling



APOPHOROMETER

an indifferent gas to be introduced around the heated substance or a vacuum to be established.

The procedure is as follows: The lower watch-glass is raised till it is in contact with the ribbon. From 5 to 30 milligrams of the substance are distributed over the ribbon so as to extend nearly over its entire length. The upper glass is then laid on. All is now ready for the experiment. In some cases it is desirable to weigh the platinum ribbon before placing it in position. And again in some cases it is advisable to make a preliminary experiment on a portion (not weighed) of the powder in order to observe the behavior of the substance.

A current is now passed through the ribbon and gradually increased. It may be that more than one sublimate is obtainable from the substance by careful regulation of the temperature; or it may be that only the one is to be expected, and the residue being non-volatile very great care in raising the temperature is unnecessary. When the sublimate is coming off freely the high temperature is maintained unaltered for from 10 to 15 minutes or until the sublimation is seen to be completed. The current is then cut off and the glasses let cool; but while still considerably above air temperature the upper glass is lifted off, the lower one let down and removed, and both glasses are clipped together for weighing. This is important, as some sublimates are hygroscopic. It is well to use watch-glasses which are ground to meet accurately.

It has been assumed that the presence of oxygen is not injurious. In many cases it is not. If it is, the receiver must be taken into use and the air excluded.

If there is any uncertainty as to the completion of the process, a second pair of watch-glasses should be placed around the ribbon and the current again turned on. We can thus make sure that the volatile constituent has been completely removed.

The whole process is one of ease and simplicity. It enables the nature of the sublimate to be anticipated, and its character enables it to be determined, and of course analytical tests may be applied to it after its weight has been ascertained. Knowing the nature of the sublimate the amount of oxygen may be calculated out.

The weight of residue left upon the ribbon is of no instances of importance. But it is necessary to bear in mind that the residual element may be oxidized in these cases. A re-excess of oxygen has been permitted. It will sometimes be found that from this cause the residue may actually weigh more than the original sample of the substance.

Certain substances may alloy with the platinum, and owing to the extreme thinness of these wide ribbons, cause it to fuse. These may be dealt with either on heavier platinum ribbon or on carbon. Molded strips of carbon would, doubtless, be obtainable. In point of temperature-range carbon has also advantages, but this is of minor importance, for there are few sublimates not obtainable by the use of platinum. It should be remembered that ordinary carbon may contain volatile impurities. If we desire to deal with very high temperatures the watch-glasses should be of transparent vitreous silica. The results obtained by this method with various substances are described.

The Non-Ferrous Metal Market

In general the tone of the non-ferrous metal market is weaker than reported a month ago. In view of the unsettled conditions, buyers are not anxious to enter the market, and as a consequence concessions have been made by the sellers. Special brands have commanded special prices, but ordinary requirements have been met at lower figures.

Copper.—Concessions have been made to induce buyers to purchase, and the market has fallen to a point that makes buying attractive. Few transactions are reported in Lake copper, and the quotations are nominal. The latest quotations are for Lake, 158¢ to 158½¢ cents; and for electrolytic, 152½¢ to 153¢ cents.

Tin.—The market is generally low in view of the decline in London. Domestic purchasers have entered the market freely and the metal is again becoming scarce. February tin in the domestic market is quoted at about 40½¢ cents.

Lead.—This market is quiet, and prices have not changed materially. The foreign market is slightly firmer. The latest quotations are 4.15 to 4.20 cents, St. Louis, and 4.30 to 4.35 cents, New York.

Spelter.—This market is weak with a slight decline. The price has declined twice during the month and at last reports was 5.05 to 5.15 cents, St. Louis, and 5.20 to 5.30 cents, New York.

Other Metals.—The aluminum market has been rather dull, with metal selling at 23½¢ to 24¢ cents, New York. Prices for antimony also have declined slightly, different brands bringing from 8.25 to 9.00 cents per lb. The quicksilver market is quiet but prices are unchanged at \$40 for domestic orders and \$37.50 for export.

Portland cement manufactured in the United States in 1912 is estimated by the U. S. Geological Survey at 8,049,088 barrels, which estimate is believed to be correct within 1.3 per cent. The production showed an increase of 4.3 per cent over that of 1911. The curve of production of Portland cement is still rising and has yet to take its last downward turn.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines of November, 1912, was 62. The total quantity of ore milled during that period was 2,213,000 tons. There were 9000 stamps in operation with an average duty of 8.58 tons per 24 hours. The mills in commission numbered 280. The yield for the month was 757.33 fine ounces gold. The report shows a steadily decreasing number of stamps and increasing number of fine mills in use.

Standard Screens for Screen Analyses

By Robert H. Richards

Great interest has been shown in recent years, and rightly so, in the importance of having standard sizes of screens, or sieves, in laboratories where sampling and screen-sizing has to be done. The need of such a standard is occasioned by the fact that without it the worker on mill or furnace products in one part of the country can not read the article written by the worker in another part of the country, and translate the figures obtained into the measures that he is used to in making his own experiments, unless he knows the sizes of grain that are dealt with in the experiments described.

Some efforts have been made in this direction up to the present time, and the writer will refer to two. The sieve scale adopted by Rittinger, in his work on ore dressing, started with 1 mm. and laid out the different sizes for the screen holes, for sizes above 1 mm., by multiplying each lower size by 1.414, or the square root of 2, in order to obtain the size next larger; going up in this way until he reached the largest size that in all probability would be needed, namely 64 mm., which is about $2\frac{1}{2}$ in. in size, and going down to sizes below 1 mm., by dividing the size of the hole in each screen by 1.414, in order to obtain the size of the hole in the screen next smaller, and so on down the series until the smallest possible size of screen hole was reached. In bringing out this sieve scale so far as we know Rittinger simply worked it out as a very desirable plan, but so far as we know no manufacturer took up the making of the screens, and no standard set of screens were to be had.

The second instance to which the writer will refer is the standard set of sizing screens adopted by a commission appointed by the Institution of Mining and Metallurgy of London, and presented at their twentieth session 1910-11. This commission took the ground that miners, mill men, and metallurgists are so wedded to the use of the word "mesh" as description of the size of a sieve that it was unwise to try to introduce any other mode of designation except "mesh."

It is known to all who have given any thought at all to the matter that the word "mesh" does not define the size of the sieve hole, and therefore the size of the product which the sieve makes; for example, the writer has before him the catalog of a manufacturer of screens, and in it he sees that a 14-mesh screen, with the coarsest wire that is ever used by this company would have a hole 0.030 of an inch in diameter, while the 14-mesh screen, with the finest wire that is ever put into that size of screen would have a hole 0.061 of an inch in diameter; and there are sieves made by this manufacturer that have fourteen different sizes of wire, ranging between the above two extremes; 14-mesh sieve therefore has sixteen different sizes of hole according to the sixteen different sizes of wire used. Therefore the writer who speaks of a 14-mesh sieve in his descriptive article may mean to his reader that he is talking about grains anywhere from 0.030 of an inch to 0.061 of an inch, failing utterly to convey a scientific fact to his reader.

The commission, aforesaid, in making their screen set, known as the I. M. M. standard screens, fully realized this difficulty, and met it by arranging to use wires of the same diameter as the width of the holes. This gives the reader definite information as to the size of the grain that is meant when a particular mesh of the sieve is used. These screens, however, do not have a constant ratio between two adjacent sizes; in fact the ratio between the width of 100-mesh holes and 90-mesh holes is 1.095, while the ratio between 30-mesh and 20-mesh is 1.508. Other lesser variations occur all the way along the series, and in fact there is no attempt at a uniform ratio.

There is another difficulty which this method runs against, and that is, that in order to have screens of uniform size of opening, we must use double-crip wire cloth, and a prominent manufacturer of screens informs the writer that it is quite difficult to make double-crip wire cloth where the wire is as

large as the size of the holes, particularly when one gets to the coarser sizes, and if single-crip wire cloth is used, the wires are not fixed, and the sizes of holes will vary to a ruinous extent by the slipping of the wires.

In regard to the use of a constant ratio between the sizes of holes of the adjacent sieves, the writer believes there are very strong reasons in favor of using a constant ratio. If one is making a series of sizing tests from a crushing machine, one wishes to know the proportional amount of the sizes at different points down the series, and will probably wish to draw a curve of the same. If one pair of sieves has a very small ratio of sizes of holes, and another has a very large ratio, then no regular statement of quantities of products made by the oversize of the successive sieves will give its full meaning, nor will it be possible to make a curve which will truly represent the result of that crushing operation.

Also if the series of products obtained are to be subjected to any concentrating operation, with a view to ascertain the ability of the ore to be concentrated, there will be no fair comparison between two products, using the above illustration, where in one instance the coarsest grain in the product is 1.5 times the size of the finest, while in the second case the coarsest grain in the product is only 1.09 times the size of the finest grain in the product. The work of concentrating on two products differing so completely in their limits of size as those two products would differ, would fail entirely of being comparable one with the other.

Again, the very fact of using "mesh" as the standard of size of sieve invites the man who has not obtained the I. M. M. standard sieve to talk about his products by use of the word "mesh," when his products are made by sieves of different make, and therefore have different sizes of holes from the I. M. M. standard screens.

On these three accounts (1) the difficulty of making double-crip wire cloth of such coarse wire; (2) the difficulty of drawing deductions from curves, and of comparing results; and (3) the difficulty of having others not provided with the I. M. M. standard entering into the discussion and using the word "mesh," it seems to me that a standard set of screens, which is standardized according to the size of its hole, instead of according to the number of meshes, and that has a constant ratio from one end to the other, for example, Rittinger's ratio of 1.414 or the square root of 2, and having the liberty to use wires that are smaller than the size of the hole, admitting therefore of the use of the most perfect kind of double-crip wire cloth, can not fail to have great advantages over the series of standard screens based upon mesh.

Such a set of standard screens has just now been prepared by the W. S. Tyler Company of Cleveland. This set of screens begins with 200 mesh as its starting point, which is a size of screen that has been standardized at Washington by the U. S. Government, and uses the ratio upwards from that point of 1.414, running all the way to the maximum size. These screens are made with the greatest care in order that they shall always be standard in size of hole, and in order that the various experimenters in the different parts of the country who make screen analyses of their products can read each other's results, and know exactly what is meant by the results.

Since the ratio between the width of the hole between the next small screen and the next larger is always the same, namely, 1.414, a sizing test made by this set of screens will give the proportional quantity of material that belongs between each two sieves according to a perfectly definite law, that is to say the largest grain in any product will always be 1.414 times the size of the smallest grain in that product. As a consequence of this the series of products obtained will have their quantities, or weights, proportional to a perfectly definite law, with the variable ratio entirely eliminated. They will yield curves which truly represent those quantities according to that definite law, and if concentrating tests are desired upon these products, the results of concentrating will be comparable all the way down the line of products, ranging from the coarsest to the finest.

The writer warmly commends this effort to make a valuable standard set of screens for use in screen-sizing tests, and hopes that they will generally be adopted by the mining engineers, the mill men, and the metallurgists of the country, in order that the papers written on these various subjects may have the greatest uniformity in the description of the products entering into such discussions.

*Massachusetts Institute of Technology,
Boston, Mass.*

An American Oil Engine Test

The Busch-Sulzer Bros.-Diesel Engine Company of St. Louis, Mo., have just issued a well illustrated booklet, entitled "The Diesel Engine—A Test," which seems to deserve some comment.

The booklet gives the report of an exhaustive test of a 225-hp, 4-stroke cycle, Busch-Sulzer-Diesel heavy oil engine. This is the engine invented by Dr. Rudolf Diesel, of Munich, Germany, and introduced into the United States under patents and licenses obtained from Dr. Diesel by Mr. Adolphus Busch, of St. Louis, and built by the Busch-Sulzer Bros.-Diesel Engine Company, of St. Louis, in the shops of the Power & Mining Machinery Company, Cudahy, Wis., and installed a little more than a year ago in the plant of the Hugo (Okla.) Ice & Light Company.

Since most that has been written in this country about performances of the Diesel engine has been based upon the showing of European tests, the results of this American test of an American built Diesel heavy oil engine, designed on lines to meet the best ideals of American practice, are interesting.

The test was made on the 225-hp Diesel engine mentioned, by Prof. A. C. Scott, of the Scott Engineering Company, Dallas, Tex., while the engine was about its daily work in the operation of the power plant of the Hugo (Okla.) Ice & Light Company.

The engine had been in service something over six months when the test was made and there was no tuning up, or special preparation made therefor. The test extended over a period of three days and records were taken at all stages from no load to 10 per cent overload.

These records show a fuel consumption of 10.8 gal. of oil at $\frac{1}{4}$ load; 6.8 at $\frac{1}{2}$ load; easing down to 6.2 gal. at full load for 100 net brake hp-hours, or, 0.441 lb. of oil per net brake hp-hour, and a shade less at the 10 per cent overload.

The thermodynamic records taken, based on net useful output, show 17.4 per cent at $\frac{1}{4}$ load; 27.8 per cent at $\frac{1}{2}$ load; 29.5 per cent at $\frac{3}{4}$ load; 30.3 per cent at full load, and 30.2 per cent at 10 per cent overload.

There are no European records known to us which show better, and this speaks well for the American practice.

There have been installed so far over 60,000 hp Busch-Sulzer-Diesel oil engines in various lines of industry in twenty-five states and the Busch-Sulzer Bros.-Diesel Engine Company has now determined on building a large manufacturing plant at St. Louis, to add to the output of the American & British Manufacturing Company, at Providence, R. I., and the Power & Mining Machinery Company, at Cudahy, Wis., who have been building these prime movers for the Diesel Engine Company for some years.

This plant is now nearly completed, and will begin operations about June 1 next. It will, when finished and equipped, represent an outlay of over \$1,000,000, and will be one of the most up-to-date, and the largest plant in the United States devoted exclusively to the building of one type of engine.

Gould, Free and Ash is the name of a new firm of chemical engineers with offices in the Monadnock Building, San Francisco, Cal. Mr. Ralph A. Gould was formerly chief of the San Francisco Food and Drug Inspection Laboratory of the United States Bureau of Chemistry. Mr. Edward F. Free was formerly with the United States Bureau of Soils in Field Potash Investigations. Mr. Charles S. Ash was formerly chemist and assistant superintendent of the California Wine Association.

Evaporators for Export

One practice of the Swenson Evaporator Company, of Chicago, which deserves notice has to do with the shipment of goods to countries far distant from the place of manufacture. Two Swenson evaporators which were sent recently to South America on the order of Liebig's Extract of Meat Company, of London, England, were designed for the greatest possible simplicity in construction and operation, thus resulting in a minimum of attention and repairs.

At the same time, with each equipment were sent ten extra tubes, an extra vacuum gage, a full extra set of tube gaskets, extra peep-hole glasses, and additional special gaskets of all kinds. It is a practice of the Swenson Company to furnish these spare parts for foreign shipments because of the distance from the factory and the long time which would ordinarily be consumed in obtaining spares when for any reason they might be needed.

With the pumps were also sent a number of spares, for the same reason.

German Competition on the Smoke Problem

Since the results of the prize competition of Aug. 7, 1908, of the Department of Finance of Saxony as to the prevention of damage by smoke to agriculture and forestry has not been satisfactory, the department has decided to grant rewards for inventions which make it possible to render harmless to plants the gases coming off from furnaces and chemical works without in any way limiting the operation of the works.

Methods and apparatus which are only intended to produce a soot-free combustion will not be considered.

All applications will be investigated and an opinion given on them by the commission appointed by the Department of Finance for the investigation of the smoke evil.

Applications, which must be in German and which must contain any necessary drawings and analyses, should be addressed to the Königlich Sächsische Finanz-Ministerium, II. Abteilung, Dresden, Germany.

Rewards may also be granted for literary work intended to promote the solution of this problem.

The Smoke Abatement Problem

The Committee for the Investigation of Atmosphere Pollution, appointed at the International Smoke Abatement Conference and Exhibition held in London last March, has held three meetings in London and has just published what may be regarded as an "Interim Report."

This report states that after careful consideration of all the various methods that have been suggested or tried for measurement of the impurities of the atmosphere, that employed for the *Lancet* investigation of the soot and dust fall of London in 1911, has been selected as the simplest, and the one most likely to yield satisfactory results, under the conditions which will govern the observations that are to be made. The method is based upon the use of an apparatus resembling an enlarged rain-gauge, with a catchment area of 4 sq. ft. This gauge receives all the dust and soot that falls by its own weight, or is carried down by the rainfall during the period of its exposure, and on examination of the water which collects in the bottle attached to the apparatus, the amount of total suspended matter, tarry oils, soot, etc., can be determined.¹ A circular letter has been sent out by the committee to all the more important city and local authorities in the United Kingdom asking for their co-operation in the application of this method of observation in districts over which they have administrative powers.

This circular has met with a most gratifying response. The authorities of a large number of important cities have already signified their intention of commencing observations on the lines suggested by the committee, and many other authorities are only waiting for further details, before promising their

¹We intend to describe the apparatus more in detail in a future issue.

support to the movement, and also co-operation in the work.

Birmingham, Bradford, Leicester and Newcastle are the most important of the cities that have definitely promised their support; but there is no doubt that Glasgow, Liverpool, Manchester and London will join in these observations.

The new movement that has been initiated by the committee for studying and recording the character of the soot fall in various industrial centers of the United Kingdom, is therefore meeting with considerable support, and there is little doubt that the observations and records will prove of considerable value to all interested in the progress of smoke abatement.

Dr. W. N. Shaw, F. R. S., of the Meteorological office, is chairman of the committee; and its secretary is Dr. J. S. Owens, 47 Victoria Street, S. W., from whom any further particulars regarding the work of the committee can be obtained.

Annealing Furnace and Electric Steel Furnace

The Midvale Steel Company of Philadelphia, Pa., has ordered a battery of six large Stobie annealing furnaces for the heat treatment of tool steel bars. The installation will be the largest in the world for this class of work, having a total capacity of 7500 tons of steel bars per year.

It is about six years since Mr. Victor Stobie, of Sheffield, erected in that town the first of the then, newly invented furnace for annealing tool steel bars, which had been designed by him. Since that date, in addition to the many furnaces working in Sheffield, a considerable number of furnaces has been erected in steelworks in all parts of Germany and Sweden, including the biggest Continental steel concerns such as Krupp, of Essen, Baidonhutte, of Kattowitz, Lindenburg, of Remscheid, Bismarckhütte, of Schwientochlowitz, Oberschlesische Eisen, A. G., of Gleiwitz, Hofors, A. B., of Sweden, etc.

At the plant of the Midvale company the furnaces are producer-gas fired, the gas being manufactured in plant attached to the main structure. Such gas costs about 1.5 cents per 1000 cubic feet and offers the advantage of being under as good control as a gas kitchen range. The furnaces are built underground, and can be regulated in fifty to one hundred sections according to the length of each furnace.

A few notes should be added on some recent work by Mr. Victor Stobie on electric steel. As has already been noticed in this journal, a big all-electric steel works is being erected on Tyneside by Mr. Stobie which will be all British from foundation to roof. The electric furnace has been designed by Mr. Stobie to "embrace all the features which are looked upon as essential by the expert Sheffield steel maker." No detail of construction are yet available. But it is stated that a 5-ton Stobie electric steel furnace is shortly to be erected in a German steel works.

Pulverizing Coal

Elsewhere in this issue will be found a very interesting article by Mr. H. G. Barnhurst, of the Fuller Engineering Company, of Allentown, Pa., on the proper methods of using powdered coal as fuel.

This subject is attracting very considerable attention at present as a result of the rise of the price of crude oil, and in this connection the matter of the cost of pulverizing coal becomes important. Any figures from practice on the cost of pulverizing coal should, therefore be welcome.

We are obliged to the Bradley Pulverizer Company of Boston, Mass., for the following figures obtained with Griffin mills in cement plants.

The 30-in. Griffin mill has been used for the past 24 years as a coal pulverizer. The "Giant Mill" has been on the market but three years, but in the short period has demonstrated that it is a very economical mill. It is really an enlarged 30-in. mill designed with the view of giving a larger output and has many improvements which past experience has advised. The Giant mill is now used in many important cement plants for grinding cement rock, slag, cement clinker, and coal.

At the plant of the Penn Allen Cement Company, Penn Al-

len, Nazareth, Pa., where 30-inch Griffin mills are used, the horsepower consumed is from 18 to 20 hp, the fineness 95 per cent through 100-mesh screen, the output approximately 1½ tons per hour, the cost of up-keep about ¼ cent per ton.

At the plant of the Cape Girardeau Portland Cement Company, Cape Girardeau, Mo., where 30-in. Griffin mills are used, the horsepower consumed is 25 to 30 hp, the fineness 93.2 per cent through 100-mesh, the output 2.67 tons per hour.

At the plant of the Iola Portland Cement Company, Iola, Kan., where 30-in. Griffin mills are used, the horsepower consumed is 25 hp, the fineness 95 per cent through 100-mesh, the output better than 1½ tons per hour.

At the plant of the Knickerbocker Portland Cement Company, Hudson, N. Y., where Giant Griffin mills are used, the horsepower consumed is from 40 to 50, the fineness 94 per cent through 100-mesh, the output over 3 tons per hour, the up-keep "practically nothing."

One of the largest users of Giant Griffin mills for pulverizing coal states that they are procuring an output of from 3 to 5 tons per hour to a fineness of from 90 to 95 per cent through a 100-mesh screen depending on dryness and texture of coal, since, of course, the output of any mill is regulated by dryness and character of material.

Calendars, Diaries, Directories

Calendar.—We have received an attractive calendar from the Lead Lined Iron Pipe Company, which shows several pictures of their lead-lined iron pipe and lead-lined iron valves. This calendar also gives some interesting information about a few of the many installations of their pipe and valves. It will be mailed to interested parties.

American Cement Directory.—We have received from the Bradley Pulverizer Company, Boston, Mass., their very useful American Cement Directory for 1913, compiled by the Technical Press of Boston, Mass. Besides illustrated notes on the Griffin mills, made by the Bradley Pulverizer Company, it contains an alphabetical list of Portland cement plants in operation in this country with the names of officers.

The Badger Chemical Diary is a new annual published by the E. B. Badger & Sons Company, of Boston, Mass., which should prove exceedingly welcome to chemical engineers. Besides illustrated notes on the various chemical apparatus made by this company (evaporation, vacuum pans, stills, dryers, autoclaves and digestors, lead-and-silver-lined apparatus, etc.), it contains useful engineering data on specific heats, expansion coefficient of liquids, properties of various chemicals, strength of materials, properties of saturated steam, heating data on various fuels, conversion tables, etc. It also contains a set of maps and a diary. The intention is to enlarge the engineering data in future issues. But as it is, this first edition marks an excellent beginning.

Personal

Mr. M. W. Atwater, of Butte, Mont., has recently resigned his position as general manager for the Butte & Superior Copper Company, to accept the management of the Davis-Daly Copper Company. He succeeds Mr. William B. Fisher, who has resigned.

Mr. James L. Bruce, general manager for the Continental Zinc Company, Joplin, Mo., is assisting Mr. J. R. Finlay in the examination of the St. Joseph and Doe Run lead mines in southeastern Missouri.

Mr. C. F. Buck has been appointed chief engineer and superintendent of construction for the Dominion Nickel Copper Company, and will have charge of the design and construction of a new smelter at Sudbury, Ont. Mr. Buck built the Hayden smelter of the American Smelting & Refining Company, and rebuilt the Tacoma plant of the same company.

The Colorado Chapter of the American Mining Congress, at its annual meeting, elected the following officers and three

tors for 1913: Governor, S. D. Nicholson, Leadville; first lieutenant governor, J. M. McClave, Denver; second lieutenant governor, A. L. Burris, Cripple Creek; third lieutenant governor, Bulkeley Wells, Telluride. Executive Committee: V. C. Alderson, Golden; R. L. Martin, Central City; Thomas Tonge, Denver; secretary and treasurer, E. L. Wolcott, 725 Majestic Building, Denver. Directors: D. W. Brunton, Denver; E. W. Bosco, Creede; E. A. Collum, Denver; R. D. George, State Geologist, Boulder; A. W. Harrison, Silverton; R. M. Henderson, Breckenridge; Irving Howbert, Colorado Springs; W. E. Renshaw, Idaho Springs; John T. Roberts, Jr., Ouray; G. S. Wood, Cripple Creek.

Mr. Karl Eilers is making a tour of inspection of the smelters of the American Smelting & Refining Company, and expects to return to New York about the last of March.

Mr. Robert M. Keeney has been appointed electro-metallurgist of the Bureau of Mines, and will be stationed at Pittsburgh. He will be engaged for the present on experimental work in the treatment of non-ferrous ores in the electric furnace.

Mr. Frederick Laist has been appointed general superintendent of the Anaconda smelter of the Anaconda Copper Mining Company, succeeding Mr. William Wraith.

Mr. Richard K. Meade has opened an office and laboratory at 202 North Calvert Street, Baltimore, Md. Special attention will be given to lime and cement.

Mr. Uley Wedge recently made a trip to the cities on the the Pacific coast.

Mr. William Wraith, who has been general superintendent of the Washoe plant of the Anaconda Copper Mining Company for many years, has been made general manager of the International Smelting & Refining Company, and will reside in Salt Lake City. He assumed his position on the first of this year.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

669,752, March 12, 1901, Paul W. Knauf, of Philadelphia, Pa., assignor to The Electrical Lead Reduction Company, of same place.

Relates to apparatus for electrical reduction of lead sulphide, etc., and consists of a series of preferably cone-shaped pans of antimonial lead, superimposed, so that a gas-tight and insulated joint is made between the several pans. Within each pan is a removable insulating lining, covering the sides, and restricting electrolytic action to the bottom of the pan. The pans are supplied with a suitable quantity of lead sulphide and 10 per cent sulphuric acid, the top pan connected to the positive pole, and the bottom to the negative pole. When a series of such pans are thus connected, the bottom of an upper pan constitutes the anode, and the lead sulphide the cathode; hydrogen liberated by electrolysis reduces the lead sulphide to metallic lead, and forms hydrogen-sulphide which passes out through a vent to a stack.

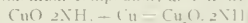
669,926, March 12, 1901, Carl Hoepfner, of Frankfort-on-the-Main, Germany.

Relates to a method of depositing zinc from a solution in a three-compartment cell. The cell consists of a plurality of U-shaped sections, separated by diaphragms, all bolted together. Every anode and cathode compartment is connected in series by passages, so that the anode electrolyte circulates through anode compartments, and the cathode electrolyte through cathode compartments. The intermediate compartments are also connected by passages, either in series or in parallel. The level of the solutions is so regulated that the catholyte is at

a higher level than that of the intermediate cell and still higher than that of the anolyte. There is therefore a constant diffusion of solution from the cathode compartment toward the anode compartment, which prevents contamination of the catholyte by the anolyte. The anode is a metal soluble in the anolyte, and electronegative to zinc, as lead or copper. On dissolving, it forms nitrates, acetates, etc., according to the electrolyte used, and after circulating through the several compartments, is precipitated in a suitable vessel outside of the cell, by a zinc salt as zinc sulphate, zinc carbonate, etc. The solution freed of lead, etc., is now used as the catholyte. The intermediate compartment is supplied with a solution preferably identical with the catholyte, and contains a separator which is electrically connected to the cathode through a resistance. As anodes, impure lead or copper ingots may be used, and the metal subsequently precipitated is highly pure and may be suitably reduced or otherwise treated. Any lead or copper that diffuses into the intermediate compartment will be precipitated by electrolysis; or the intermediate electrolyte and catholyte may be further purified by zinc dust. All of the electrodes are constantly vibrated to assist stirring and depolarization.

677,089, June 25, 1901, John C. Kessler, of Milwaukee, Wis., assignor of one-half to Christian Wahl, of same place.

Relates to the extraction of native copper from precious metal tailings by a leaching solution containing a cupric ammonium compound, such as cupric oxide, cupric sulphate, etc., dissolved in ammonia. The tailings or pulverized ore is placed in a leaching tank and covered with the cupric ammonium solution; this is then covered with about 4 in. of water which prevents loss of ammonia by evaporation. After 24 or 48 hours, the lower ammonia solution is run off, and the ore covered with water to wash it. The cupric ammonium solution dissolves copper forming cuprous-ammonium compounds, as follows:



This solution thus obtained is electrolyzed, precipitating half of the copper, the remainder being oxidized to the cupric state and reused.

678,526, July 16, 1901, Charles P. Stewart, of Oakland, Cal.

Relates to an apparatus in which gold, etc., is electrolytically deposited from a flowing cyanid solution upon a quiescent body of mercury. The apparatus consists of a long shallow trough or tank through which a cyanid solution flows. The bottom of the trough is covered with a layer of mercury electrically connected as a cathode; a number of separate anodes are adjustably suspended in the solution, the adjustment providing for any irregularities in wear. When the mercury is saturated it is removed and the precious metals separated in any suitable manner.

682,155, Sept. 3, 1901, Charles P. Tattno and George Delms, of Seattle, Wash.

Relates to an apparatus for extracting precious metals, and refers to their prior patents 640,718 and 653,325. This apparatus consists of a suitable acid-proof tank having a copper pan covering the bottom, and a mercury cathode in the copper pan. Above the mercury are rotating paddles which keep the pulp and solution well stirred. Above the paddles are a number of depending carbon anodes, supported by a lid. The lid, anodes and paddles are capable of being raised vertically from the tank to facilitate charging, etc. A supplementary anode projects through the side of the tank and serves to keep current continuously electrolytically protecting the mercury and copper from corrosion when the main current is cut off. When the operation is completed, the tank is emptied through a spout in the bottom.

682,018, Dec. 17, 1901, William Orr, of Salt Lake City, Utah, assignor to The Gold and Silver Extraction Company, of America, Ltd., of Denver, Col., a joint stock company in Great Britain.

Relates to a method of regenerating cyanid solutions, particularly those containing copper and zinc cyanids, and an apparatus to electrolytically precipitate the copper. The apparatus consists of a tank having a plurality of pairs of electrodes, alternate ores of zinc connected as anodes, and the intermediate

ones of copper, connected as cathodes. The solution flows between the electrodes, the copper being deposited and an equivalent of zinc going into solution. To the solution now containing zinc cyanid is added an alkaline or alkaline earth hydroxid, forming, for example, sodium zincate. To this solution is added just enough sodium sulphide to form zinc sulphide which separates, and may, if desired, be filtered off and recovered. The solution now consists of the original cyanid plus some alkaline or alkaline earth hydroxid, and is used in the usual manner for treating ores.

689,070, Dec. 17, 1901, Alexander Stanley Elmore, of London, England.

Relates to the separation of mineral substances by the selective action of oil, and states that in the presence of an acid the separation is more effective. The apparatus used consists of a mixer, a settling tank, and a centrifugal separator. The ore is suitably powdered and is supplied to the mixer with five to ten times its weight of water, forming a thin freely flowing pulp. To the mixer is also added a thin stream of oil, and an acid soluble in oil but insoluble in water, such as oleic acid which together with the oil coats the metallic particles, such as metal dust, tellurids, sulphides, graphite and elemental sulphur, with a film of oil, thereby causing them to float. Instead of oleic acid, or a similar acid, an acid soluble in water but insoluble in oil, such as sulphuric acid, may be used. The quantity of acid added, in either case, will vary with the materials being separated; in some cases it may be as little as 1/500 part of the oil or water used.

689,674, Dec. 24, 1901, Albert Isaiah Irwin, of Cripple Creek, Col., assignor of one-third to Caleb F. Bryant, of Cripple Creek Mining District, Teller County, Col.

Relates to apparatus for electrolytically dissolving and precipitating the metallic content of an ore. The electrolytic apparatus consists of a traveling endless belt, of metal, on which are cross-pieces of insulating material. The movement of the belt conveys the ore through a suitable solution, and discharges it from the machine. The belt is connected as an anode, and adjacent thereto are cathodes. With gold ores, a cyanid solution may be used as electrolyte, the dissolved gold and silver being electrolytically deposited. If the ore gives an acid reaction to water, sufficient alkali is added to maintain the alkalinity of the electrolyte.

689,959, Dec. 31, 1901, Edward Leslie Graham, of Upper Warrlingham, England.

Relates to the disintegration and comminution of ores, etc., by the action of gases generated by passing an electric current through a suitable acid solution, such as sulphuric, hydrofluoric, or mixed sulphuric and hydrofluoric. Other acids and salts may also be added, such as ammonium chlorid or carbonate, barium chlorid, fluorspar, etc. With free-milling gold ores, 2% per cent of sulphuric acid and 1 per cent of hydrofluoric acid are said to give good results; other ores, such as those of copper, silver, and antimony, require different percentages. The current strength will vary with the ore and size of the vat, and should have a voltage of about fifty volts per vat.

Book Reviews

Die Spezialstähle. Ihre Geschichte, Eigenschaften, Behandlung und Herstellung. By G. Mars, Dipl.-Ing. Octavo (15½ x 24 cm.), 517 pages, 143 illustrations; price 17 marks, bound 18.40 marks. Stuttgart: Ferdinand Enke.

A monumental treatise by the director of the research department of the Rhenish Metalware & Machinery Works, in Düsseldorf. It is far more than a mere compilation, for the author discusses in a very instructive manner the relations of the steels to each other, the bearing of the periodic system of the elements on the effects produced by different metals, and even reflects upon the somewhat neglected question of the improvement of these steels by proper treatment of the fluid metal.

After chapters on history, metallography, the iron-carbon diagram, heat treatment, forging and cold-working of steels, the author takes up in distinct chapters the carbon, Si, Mn, Cr, Mo, W, V, Ti, Al, and Ni steels, and more briefly, in two chapters, the V, Ta, B, Sn, Co and Cu steels. There are also separate chapters on construction steel, high-speed tool steel, and (76 pages) on the manufacture of the special steels.

The whole is worked over with great accuracy and completeness. Friends of the electric furnace steel will read with satisfaction the verdict of this expert upon the rôle of the electric furnace in the manufacture of these special steels: "The electric steel smelting process appears to be undoubtedly the highest stage in the development of steel manufacturing processes."

* * *

Metallurgische und technologische Studien über das Ausglühen von Metallen und Legierungen. Von Dr. Ing. Max Weidig. Large quarto (22½ x 30 cm.), 121 pages, 48 illustrations; price 6 marks. Berlin: Leonhard Simion Nf.

The author is privat-docent at the Bergakademie, Freiberg. In one-half of the book he reviews the results of others on the influence of annealing upon the structure and mechanical properties of metals and alloys; in the second half he relates his own experiments upon nine materials: copper, nickel, copper-nickel, brass, aluminium bronze, tin bronze, tin-zinc bronze, german silver, and aluminium. Each of these materials were tested hard and annealed, and the effects on color, luster, structure, hardness, tensile strength, ductility, and electrical conductivity carefully determined.

An interesting observation is the relative quantities of heat energy absorbed in annealing slowly at a low temperature, and quickly at a higher temperature. Using an electric furnace for heating, it is usually more economical of power to anneal quickly at the higher temperature. This is only one of many useful items of information in this monograph.

* * *

Transactions of the American Foundrymen's Association: Vol. xx. Edited by Richard Moldenke, Secretary. Octavo (15 x 23 cm.), 684 pages, numerous illustrations. Watchung, N. J.: The Association.

A fine collection of mostly short papers, dealing with details of foundry practise up to the theory of alloys. An 18-page paper, finely illustrated, on the use of the induction furnace for making steel castings, by Mr. C. H. Vom Baur, is a notable contribution to that subject; a paper on pyrometry, by Mr. S. H. Stupakoff, also deserves mention. Many of the others are well worth reading, although the English and the manner of presentation are frequently faulty. The printing and binding are excellent.

* * *

Die Praxis des Eisenhüttenchemikers. By Dr. Carl Krug. 15 x 23½ cm., 226 pages, 31 illustrations; price 6 marks. Retail price New York, \$2.00. Berlin: Julius Springer.

Chemische Untersuchungsmethoden für Eisenhütten und deren Nebenbetriebe. By Ing. Albert Vita and Dr. Carl Massenez. 13 x 21 cm., 175 pages. Price 4 marks. Retail price New York, \$1.35. Berlin: Julius Springer.

Dr. Krug is "Dozent" in the "Bergakademie" in Berlin. His book is the product of 15 years' teaching experience, and is very carefully and painstakingly written. Engineer Vita is chief chemist for the Friedenshütte in upper Silesia, and Dr. Massenez teaches analysis in the Technical High School of Breslau. Their book is the product of practical experience, omitting all methods which have been superseded in recent years.

Both works are commended to the iron and steel works' chemist. Science is the most truly international factor in modern life, and the scientific man has the widest opportunity to become cosmopolitan in his profession if he will only avail himself of such assistance and information as are at his disposal. Thus, a mere bagatelle of ten marks will put him abreast of the best European practice in iron and steel works' laboratories.

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The Haber Synthetic Ammonia Process

At last year's International Congress of Applied Chemistry Dr. Bernthsen made the important announcement that the problems involved in the Haber synthetic ammonia process have been "solved fully on a manufacturing scale and that the walls of the first factory for synthetic ammonia are already rising above the ground at Oppau, near Ludwigshafen on the Rhine." The report which Professor Haber made to the Badische Company three years ago and which led to the development of the process by this company on a large scale, has now been published and a rather full account of it is given elsewhere in this issue. It is a historic document and calls for comment on the commercial, technical, and scientific aspects of the process.

From a commercial standpoint the process has various very attractive features. As mentioned in the beginning of Prof. Haber's report, there is a large margin between the value of the ammonia and the value of the raw materials, hydrogen and nitrogen. In the form of 25 per cent ammonium sulphate the ammonia is estimated to be worth 22.25 cents per kilogram, while the cost of the nitrogen and hydrogen in it is estimated at $1.125 + 4.375 = 5.5$ cents. In view of the comparative simplicity of the process and the low power requirements it is hardly risky to say off-hand that this margin is large enough to pay manufacturing cost and a good profit.

A second attractive commercial feature is the fact that with the rapid increase in the consumption of fertilizers in recent years over-production and a drop of prices due to over-production seems a practical impossibility for years to come. Taking nitrogen fertilizers alone and taking nitrate and ammonia together, Bernthsen finds that there has been an annual increase in consumption of 38,000 tons of nitrogen, corresponding to nearly 185,000 tons of ammonium sulphate. Clearly enormous quantities of synthetic ammonia must be produced to affect the nitrogen fertilizer market. This explains why Dr. Bernthsen was emphasizing that the development of the new industry would not take place at the cost of other branches of industry and commerce. It may also explain why the Badische Company has withdrawn from the Norwegian electric-furnace nitrate field. Synthetic ammonia is a big enough proposition even for a very big company.

From a technical viewpoint it is interesting to compare the synthetic ammonia process with the electric-furnace processes of making nitrate and of making calcium cyanamide. The latter two employ a very high temperature and their power requirements are high. The ammonia process employs a comparatively moderate temperature—between 500 and 700 deg. C.—and its power requirements are low. From a technical viewpoint the most interesting and novel features are the use of a high atmospheric pressure and the circulation of the high-pressure nitrogen—

hydrogen gas mixture between a furnace (where the ammonia is formed) and a refrigerator (where the ammonia is liquefied and withdrawn).

Even more interesting, perhaps, is the process from a scientific viewpoint. The process will go into history as one of the biggest achievements of modern physical chemistry. Without modern physical chemistry, based on thermodynamics, the process could never have been developed. The one fundamental problem was, indeed, of a thermodynamical nature; it was to find the best temperature and the best pressure for the reaction. How keenly Professor Haber attacked the thermodynamic equilibrium problem and how systematically he solved it, is appreciated by all who know his classical *Vorlesungen über technische Gasreaktionen*. But there was another fundamental problem to be solved which is intimately connected with the first problem. This was to find the best catalyst to get a sufficiently high reaction velocity. How the thermodynamic equilibrium and the reaction velocity influence the choice of temperature and pressure is admirably brought out in the report.

Admirable, too, is the systematic perseverant research of Professor Haber and his co-workers on the effectiveness of various catalytic agents. But the successful and clever way in which the periodic system of the elements is made use of in this research, makes the mystery of the nature of catalytic action only more mysterious and more interesting. Why should certain elements accelerate the reaction? Why should other materials (Bernthsen's "poisons") retard it? Why should the effective catalytic agents be found in certain groups of the periodic system of elements? Why should, in these groups, the elements with highest atomic weights be the best catalytic agents? There is no end of why's and apparently there is no answer. Thus, while the process emphasizes the importance of the advances made in physical chemistry in recent years, it also emphasizes the limitations of our theoretical knowledge.

Professor Haber is no stranger to American chemists. Especially among American electrochemists he has many friends and admirers. Ten years ago he attended the first Niagara Falls meeting of the American Electrochemical Society, and his able report to the Bunsen Society on the American electrochemical industries is unforgotten. In his hour of triumph his American friends sympathize with him and extend to him their hearty, sincere congratulations.

Engineers, Practical Men, and Theories

In his presidential address before the Colorado Scientific Society, Mr. George E. Collins recently made some pertinent remarks about the practical application of hypotheses and theories. Although Mr. Collins was speaking particularly of mining and not of metallurgy, it would require only a simple substitution of terms to make his comments equally pertinent to ore treatment. Comparing the so-called "practical man" with the professional engineer, Mr. Collins showed that while the former often scoffs at the ideas of the latter, regarding him as a theorist, nevertheless the practical man "also plans his work along the lines suggested by some hypothesis. The trouble is, as has been pointed out by others, there is no theorist so wild or so inveterate as your practical man; none that rides his hobby so hard or so far." This must call to the memory of every

engineer some experience with a "practical theorist"; as, for example, the man who knows that the fire assay for gold is unreliable for *his* ore; or the one who confided to us that the presence of "telluride of quick" in his ore was the principal metallurgical obstacle to its successful treatment. More serious in their consequences are the theories of subordinates in metallurgical works, who frustrate the plans and tests of their superiors because of an abiding faith in their own ideas.

Following Mr. Collins by analogy: What is needed for progress in ore treatment is the trained mind, acquainted with the literature of the subject, able to observe and to correlate and interpret results, and familiar with all the ideas of others which might have a bearing on his problem. Thus equipped, the individual will be able to form rational hypotheses on which to base experimentation. He will know when to pursue and when to abandon a line of investigation, and will be able to balance the probability of failure with the possibility of success. With all justice and consideration for the practical man, the need of the day is the trained mind.

Grinding Machines

The respective merits of the four principal types of grinding machines, rolls, and Huntington, Chilean and tube mills, have been the subject of many general discussions and some comparative tests. But despite the work that has been done in this line, the fact remains that no adequate comparative test of all the machines has ever been made, whereby the various factors of efficiency, such as power consumption, cost of operation and suitability of product, could be compared. Competitive tests in individual instances have determined the choice of machines for those particular cases, which is perhaps the best way of eliminating the less desirable machines and determining which is fittest to survive. In this way we have secured some good data, and have arrived at certain practical conclusions.

The increasing popularity of the tube mill as a secondary grinding machine, directs attention toward the deficiencies of the other types mentioned. Generally speaking, the objection to rolls is their failure to deliver a finished product in one operation, with the result that oversize in considerable quantity must be returned to the same machine. This, in turn, entails poor sizing in the trommels, and builds up feed at this point in the mill system to a tonnage which usually is considerably greater than the original feed to the mill. This condition could be avoided by a screen-faced machine with fixed maximum size of discharge, such as a Chilean mill; but a new factor would be introduced in the form of excessive slime production, which might be decidedly detrimental to the subsequent treatment of the ore. Rolls give a more granular product for concentration, the middling from which can be further ground to liberate the mineral; and for this reason rolls have been widely adopted in concentrating mills. On the other hand, the Chilean mill has been successfully used in copper concentration, for grinding an ore which must be finely divided in order to free the disseminated mineral from the gangue. But in these cases, excessive sliming has been avoided by grinding in stages, using coarse screens on the mills, and regrinding the coarser sizes of sand.

The Huntington mill works well on soft ore, but is not properly constructed to be of much effect on hard ore. It lacks the positive action of the Chilean mill or rigid rolls, and hence has little effect on an ore which is hard enough to resist crushing by the centrifugal force of the rollers against the die. On the whole, it is a rather complicated machine, which requires careful attention and the exercise of good judgment in its operation. One operator at Park City, Utah, has secured better results from this mill by connecting the upper ends of supports of opposite rollers by means of rods and a turnbuckle, thereby holding the rollers positively in position against the die.

The tube mill, and its close relative, the ball mill, are likely to displace the Huntington mill altogether, as well as make inroads on the former domain of the Chilean mill and rolls. These two machines, especially the Hardinge type, have been widely adopted as secondary grinders in concentration. They are attractive on account of their simplicity, absence of the wearing parts so common to other grinders, and the ease of replacing the grinding medium. Even for small mills the combination of a coarse crusher, ball mill and tube mill, each adjusted to produce a definite crushing effect, and thereby eliminating troublesome screens, offers such a simple and efficient means of preparing ore for different forms of treatment that it should receive attention.

The Steel Corporation's Report

The United States Steel Corporation's report for the year 1912 shows very substantial progress toward the full realization upon the improvements and extensions it has been making. The output in steel products, weighed in the form in which sold to the public, was 12,506,619 tons, an increase of no less than 1,772,624 tons over the best previous year, which was 1910. While the corporation's capacity was not fully engaged in 1912, the output was approximately only about a million tons less than capacity.

The corporation is able to meet its interest and dividend charges with very much smaller profits per ton than when it was formed, for its capacity when formed April 1, 1901, was only 8,000,000 tons, over which its actual output in 1912 was a gain of 56 per cent., while its present capacity represents a gain of between 70 and 75 per cent. Concurrent with this increase in capacity and in actual output there has been a relatively small increase in capital obligations. At the start the bonds, both of subsidiary companies and of the steel corporation itself, together with the common and preferred shares outstanding, made a total face value of \$1,400,000,000. To-day the securities outstanding amount to \$1,513,000,000. The sinking fund, interest and dividend charges have increased in hardly as great ratio. At the outset the corporation paid 4 per cent. dividends on its common stock, but assuming 5 per cent., to make a comparison with the present dividend at that rate, the sinking fund, interest and dividend charges were approximately \$85,000,000 a year; to-day they are \$91,500,000. This latter sum includes the interest paid on bonds which are carried in sinking funds, but the face value of such bonds is not included in the \$1,513,000,000 of outstanding securities already mentioned. The charges of 1901 represented \$10.60 per ton of output at capacity, while

the charges to-day represent \$6.65 per ton of output at capacity. The total earnings in 1912 represented \$9.45 per ton of actual output. The earnings covered all charges and left a slight balance for surplus, \$3,605,247.

Wall Street studies the annual reports of the Steel Corporation with much interest and pays attention to summaries of the amounts which the corporation has invested in plant betterments and extensions, for the purpose of illustrating the extent to which the corporation has "squeezed the water" out of its capitalization. The practical steel manufacturer realizes such summaries are inconclusive without adequate study also with respect to the depreciation charges which have been allowed, and on account of the vicissitudes in the steel industry it is impossible to be certain at any time how much depreciation should be allowed. While the view is apparently very superficial, it is probably much more accurate, if not so precise, to summarize the Corporation's position in this way, that it undoubtedly requires a larger capital investment to manufacture a ton of finished steel economically to-day than it did 12 years ago, when the Corporation began business, yet we find the actual showing is that in this period the Corporation has increased its capital obligations and its annual charges only 8 per cent., while it has increased its productive capacity between 70 and 75 per cent.

Since its early days the Steel Corporation has made an annual offering of its shares to employees, naming a price somewhat under the current market figure, to be paid upon the installment plan, but to bear dividends from the date of subscription. The offering of January, 1913, resulted in subscriptions by 36,119 employees, for 34,551 shares of preferred and 25,793 shares of common stock, representing respectively 0.96 and 0.49 per cent. of the total shares outstanding. Under the latest subscription, subscribing employees receive a bonus of \$5 a year on preferred shares and \$3.50 a year on common shares, as long as they retain the stock and remain in the employ of the corporation, up to a period of five years, at the end of which time the bonus which would have inured to those who dropped out is divided among the survivors. It would be interesting to know the situation at present as to participants in previous distributions, how much stock is retained by the original subscribers, the classes respectively whose bonuses have expired and whose bonuses continue. An accurate statement as to the former would, of course, be impossible, as shares sold are not always transferred on the books.

The Corporation's years of largest output were 1910, with 10,733,995 tons, and 1912, with 12,506,619 tons, showing a gain for 1912 of 16.5 per cent. over the record. There were great changes in the distribution of output, for some products actually lost in tonnage, while others made great gains. Rails lost more than 12 per cent. In minor railroad products there was a large increase in axles, but almost as large a loss in steel car wheels. The most important gain was in sheets and tin plates, 39 per cent., while tubular goods showed an increase of 28 per cent. On the whole it is readily observed that the Corporation carried its tonnage to a higher degree of finish, on the average, in 1912 than in 1910, so that the physical value of its output increased more than the 10.5 per cent. which the tonnage increase represented.

Readers' Views and Comments

Simple Approximate Rules for Temperature Conversions

To the Editor of *Metallurgical & Chemical Engineering*:

Sir.—The following simple, easily remembered, approximate rules for mentally converting high temperatures in Fahrenheit degrees into one in Centigrade degrees and the reverse, may be of interest to those dealing with furnaces; the troublesome factor 5/9 is thereby evaded. Whether they are already generally known or not in this form, the writer does not know.

Double the Centigrade temperature and deduct one-tenth of that figure, (or deduct to per cent, then double it). Thus 1000 deg. C. doubled is 2000, less a tenth (200) leaves 1800 deg. Fahr. For all ranges the error is that this value is always 32 deg. too small; hence it may be made accurate by adding 32.

Halve the Fahrenheit temperature and add one-tenth of that figure (or add 10 per cent, then halve it). Thus, 2000 deg. Fahr. halved is 1000, plus a tenth (100) gives 1100 deg. C. Accurately it is 1093.3+; hence the approximate value is only about 1/2 of 1 per cent too high. Again 3000 deg. Fahr. halved is 1500, plus a tenth (150) gives 1650 deg. C. Accurately it is 1648.9—; hence the approximate value is less than a tenth of 1 per cent too high. For the lower temperatures the errors become greater.

If 32 is first deducted from the Fahrenheit values, then the values given by the latter rule are always almost exactly 1 per cent too low for all ranges, hence could be made nearly absolutely correct if desired by adding 1 per cent.

These rules are easily remembered as it is well known that the Fahrenheit values are *not quite* double the Centigrade ones, hence something must be *deducted* after doubling the Centigrade values, or must be *added* after halving the Fahrenheit values; and that this correction factor is 10 per cent is easily remembered as this is the simplest possible of all factors, being merely a shift of the decimal point.

The mathematical proofs of these rules are very simple, as the correct factors are 5/9 and its reciprocal, beside the 32

Philadelphia, Pa.

CARL HERING.

Determination of Oxygen in Iron and Steel.

To the Editor of *Metallurgical and Chemical Engineering*:

Sir.—In the February number of *METALLURGICAL AND CHEMICAL ENGINEERING* appears an article by Mr. R. H. McMillen, on the application of the electric resistance furnace to the determination of oxygen in iron and steel.

The writer's¹ modification of the Ledebur method (*Jour. Ind. & Eng. Chem.*, vol. 3, p. 372), is referred to, and the double electric furnace as well as certain other modifications are proposed as improvements.

The laboratories under the writer's supervision have had probably a larger and wider experience with the determination of oxygen in iron and steel than any other in this country. In our opinion the modifications proposed by McMillen present no improvements over the process used by us, and are open to criticism in respect to several important points. In a routine laboratory where a large number of oxygen determinations have to be made, the time element is of great importance. With the gas blast furnace using a silica tube it is possible to cool down immediately by turning on the cold air blast, so that accurate results can be obtained in one-half hour. The double electric furnace modification runs the test for a full hour, and if the analysis is started with a cold furnace, it will require an hour to reach the specified temperature of 950 deg. C., whereas the gas blast furnace can be brought to temperature in a few minutes. In our practice we often exceed this temperature and have never found that we obtain higher results with the higher temperature.

McMillen proposes the use of a porcelain boat instead of the nickel boat used by us. We especially avoided the use of porcelain, owing to the tendency of that material to adsorb moisture from the air.

The most important criticism of McMillen's modification that we have to offer is with respect to weighing the phosphoric acid absorption tube filled with hydrogen. We believe this to be bad practice, and numerous tests made in this manner have shown discordant results owing to the tendency of hydrogen to diffuse through "air-tight" stoppers.

The McMillen method is almost identical with the one published by the writer, with the exception of the points discussed above, and for the reasons stated it does not appear that any contribution has been made to the older method. McMillen again points out the well-known fact that the Ledebur method will not determine total oxygen, owing to the fact that certain oxides and silicates are not reducible by hydrogen. Walker and Patrick have proposed a method for determination of total oxygen, by making use of a vacuum electric furnace of the Arsem type. (Paper before Eighth Intern. Congr. of Appl. Chem., *Met. & Chem. Eng'g.*, vol. X, p. 629.) The installation for this method is very expensive and at best it could hardly become a routine determination in any chemical laboratory. On the other hand, it should not be forgotten that in most cases it is only the determination of oxygen combined with iron that yields the data desired, and it is this information which can be readily obtained by the Ledebur method. The amount of oxygen combined with silica and other oxides unreducible by hydrogen has no bearing on the question as to whether iron or steel is properly deoxidized in the process of manufacture.

ALLERTON S. CUSHMAN

Institute of Industrial Research,
Washington, D. C.

* * *

To the Editor of *Metallurgical & Chemical Engineering*

Sir:—In the criticism of Dr. Allerton S. Cushman upon my article, "An Application of the Electric Resistance Furnace to the Determination of Oxygen in Iron and Steel," published in your February issue, he objects to the use of the electric resistance furnaces in my modification on account of the time required to bring the furnaces to a full heat and to cool them after the combustion is completed, thereby curtailing the number of daily determinations.

I consider that the advantages offered by this method more than compensate for the time lost, when the convenience of this means of heating is taken into account, and also the little attention the furnaces require. After the operation is started and until the combustion is completed, the operator is practically free to attend to other duties, so that very little of his time is counted against the oxygen determination. When the number of daily determinations to be made requires it, more furnaces can readily be added, using the one preheating furnace for all.

Dr. Cushman next objects to the use of the porcelain boat on account of the moisture absorbing tendency of this material. In making use of the porcelain boat this property was not lost sight of. No opportunity should be given for it to absorb any appreciable amount of moisture, as the dried boat need not be exposed to the moisture of the laboratory but a very short time while receiving the sample and being placed into the tube. Since the empty boat, which has been exposed to the air under conditions similar to those in the actual determination, is used in running the blank, a correction for the moisture which may be added this way is made when the blank is subtracted.

In the Cushman method (*Jour. Ind. & Eng. Chem.*, Vol. 3, p. 372) the statement is made that either platinum or silica

boats are used, and no reference whatever is made to the nickel boat which he now states is used.

Dr. Cushman's third and most important criticism is to the practice of weighing the phosphoric acid absorption tube filled with hydrogen. I consider the displacement of hydrogen by dry air as in the Cushman method unnecessary, since in drying the hydrogen filled absorption tube the same time is given and the same conditions maintained before weighing both before and after the combustion.

At various times experiments were conducted in order to determine whether or not the diffusion of hydrogen through the stoppers makes any variations in the weights which would materially affect the results. A phosphoric acid absorption tube was filled with dry hydrogen and dried over concentrated sulphuric acid for fifteen minutes before weighing, this procedure being repeated many times, and in no case was there any appreciable variation in weight. A second weight was made thirty minutes after the first in each case, and in no instance did the weight vary more than 0.0002 gram from the first, showing that, in a reasonable time limit, no appreciable error is introduced from this source.

R. H. McMILLEN.

*Crescent Laboratory, Crucible Steel Co., of America,
Pittsburgh, Pa.*

Dr. Struthers' Reply

To the Editor of Metallurgical & Chemical Engineering:

Sir:—Enclosed please find a copy of a letter to me from Mr. Charles F. Rand, president of the American Institute of Mining Engineers, which is a conclusive reply to the direct and inferred criticisms, so far as they refer to me, appearing in the letter of Messrs. Corning and Stone in your issue of September, 1912.

I respectfully request, as a matter of justice, that you will publish this present communication and the letter from Mr. Rand in an early issue of your paper.

JOSEPH STRUTHERS.

New York City.

Mr. Rand's letter is as follows:

"Dr. Joseph Struthers, New York.

"Dear Sir:—Your letters of February 21 and 24 have been received, together with copies of the METALLURGICAL AND CHEMICAL ENGINEERING of September, 1912, the *Mining and Scientific Press* of January 25 and February 8, 1913, and the *Engineering and Mining Journal* of February 8, 1913, in which latter the suggestion is made that you demand an investigation; with the request on your part that the criticisms appearing in these publications as far as they relate to you should be considered by the Board of Directors or the Executive Committee of the Institute.

"Your letters and the accompanying newspapers have been carefully considered by the Executive Committee.

"We are very glad to state that no charges of any kind against you are in existence before the Board.

"Your accounts as Assistant Treasurer have been audited annually by a firm of certified public accountants and always found correct.

"As to the excursions which are specifically referred to by the *Mining and Scientific Press*, you are entitled to the statement which we very gladly make that you did not originate the system by which such excursions were conducted. The system had prevailed for many years, and before you became connected with the administrative work of the Institute.

"With regard to the reference in the *Mining and Scientific Press* to the New Haven and Spokane meetings, we now find that there was no excursion connected with the New Haven meeting, and understand that you had no financial relation beyond the receipt of salary and expenses to the excursion connected with the Spokane meeting. Both of these meetings were held while you were Assistant Secretary and before you became Secretary of the Institute.

"A committee of the Board and Council has already inquired

into the conduct of the excursion to Japan which was under your charge, and exonerated you, as shown in a report which has already been published.

"As to the letter of Messrs. Corning and Stone, published in METALLURGICAL AND CHEMICAL ENGINEERING last September, we appreciate the position taken by Dr. Raymond and you at the urgency of your friends, to abstain from any newspaper controversy which might have been harmful to the Institute. At the same time we are pleased to say, concerning the general charge which might be construed to include you, that you have not received an excessive salary and that you have not been inefficient in the office work.

"The Executive Committee consists of Charles F. Rand, Benj. B. Thayer, A. R. Ledoux, Joseph W. Richards, and James F. Kemp.

"Yours very truly,

"(Signed) CHARLES F. RAND,

"President American Institute of Mining Engineers."

Early American Mining Schools

To the Editor of Chemical & Metallurgical Engineering:

Sir:—On page 120 of the March number of your journal is a note on mining schools. In this the name of the School of Mining and Practical Geology of Harvard University is omitted. This school was founded in 1865 by the appointment of Prof. J. D. Whitney to the chair of geology in 1865. In 1866 Prof. R. Pumpelly was appointed to the chair of mining, and in 1871 William H. Pettee, who had been acting as instructor, was appointed Assistant Professor of Mining.

The first class of four members was graduated in 1870. Two of these Gannett and Marvin, entered the United States Geological Survey, where Gannett still remains. In 1870 the class was taken to Michigan by Professor Pumpelly to see the iron and copper mines, and afterwards J. D. Whitney took them to the mountains of Colorado. During this part of the trip Mt. Harvard and Mt. Yale were ascended and named.

This party, so far as I know, was the first college party to make a summer visit to the Rocky Mountains. The party consisted of Prof. J. D. Whitney, of Harvard; Prof. William Brewer, of Yale; Instructors Pettee and Sharples, of Harvard, and four students, William M. Davis, who has succeeded Professor Whitney as geologist at Harvard; Joseph Bridge, Henry Gannett, now chief geographer of the United States, and A. Marvin, who was on the United States Survey, but died in 1876. Mr. Bowles, son of the editor of the *Springfield Republican* was also with the party. They were also accompanied by Charles Frederick Hoffman as topographical engineer. The school was discontinued in 1875, being merged with the Lawrence Scientific School.

Boston, Mass.

S. P. SHARPLES

The Solubility of Alumina in a Bath of Fused Fluorides.

To the Editor of Metallurgical & Chemical Engineering

Sir:—In the March number of your journal, page 121, Dr. Charles A. Doremus discloses from a paper published by St. Claire Deville in 1857, that alumina like silica was known to be soluble in a mixture of potassium and sodium fluorides fused in an externally heated crucible; and that while Deville succeeded in electrolytically depositing silicium by feeding silica into such a bath, he failed to deposit aluminium under the same conditions. Deville's conclusion was that alumina unlike silica, is not decomposed by the alkali metals.

Dr. Doremus stated that had Deville's proportions been more propitious the alumina should have been decomposed by the electric current in the same manner as silica. Deville's bath contained no aluminium fluoride. This latter deduction made by Dr. Doremus would be natural if he considers alumina when fused an electrolyte whose molecules are directly decomposed without a secondary reaction. The old literature of electrolysis teaches that pure oxides like water are not directly decomposed by the current. Deville was expecting that the liberation of

potassium or sodium on the cathode would react on alumina in the bath with the formation of aluminium, while at the same time oxygen or carbonic oxide would form against this carbon anode and fluorine would have an opportunity to combine with the alkali metal of the alkali oxide that would be forming in the bath. He knew that an alkali metal would reduce silica.

Dr. Paul Héroult in his patents and the experts for Mr. Charles M. Hall in litigation that was never decided all held that alumina in itself was capable of being directly decomposed by electrolysis. In the latter case the experts for the Cowles Electric Smelting & Aluminum Company held that it was well known that alumina was soluble in cryolite, but that alumina was not a direct electrolyte.

The following experiment performed by the writer about twenty years ago must have some scientific bearing on this subject:

A round hard carbon block, 3 in. in diameter and about 3 in. in height, was taken. Three holes were bored into one of the flat surfaces of this block to a depth of about an inch and of nearly the same diameter. One of these holes was filled with pure sodium fluoride NaF, another was filled with cryolite $(\text{NaF})_2\text{Al}_2\text{F}_6$, and the third was filled with cryolite containing a little added aluminium fluoride. The block was now uniformly heated until the material in all three holes came to quiet fusion. Three small pieces of aluminium, all having the same weight, had been previously prepared each about the size of a small bean. One of these pieces was now dropped into each of the three fused baths. The bath of sodium fluoride gave off a great abundance of sodium vapor which burned with its characteristic yellow flame. The bath formed of cryolite gave off some yellow flashes of sodium vapor. The bath containing a slight excess of aluminium fluoride, above that of cryolite, gave off no sodium vapor. Upon allowing the block to cool and breaking it, it was found that all the aluminium had been consumed with the formation of aluminium fluoride in the bath formed of sodium fluoride. On weighing the button from the cryolite bath, it was found to have lost a very small per cent of its weight, while the bath containing a slight excess of aluminium fluoride gave up a button having its original weight.

Dr. Percy in the year 1855, and Rose independently in 1856 discovered that cryolite could be directly reduced and aluminium produced by using sodium as a reducing agent, but they found the efficiency very low. This was due to too great a quantity of sodium fluoride accumulating in virtue of the reduction of the aluminium fluoride with sodium.

All three baths were heated to the same temperature in the above described experiment, which was about the temperature employed in the present commercial manufacture of aluminium. It will be noted that when aluminium fluoride is in excess to that contained in cryolite $(\text{NaF})_2\text{Al}_2\text{F}_6$, aluminium does not reduce sodium fluoride, and on the other hand, when sodium fluoride is in excess, aluminium does reduce sodium fluoride, and sodium acts in just the reverse manner as to aluminium fluoride in the composition.

Déville might have said at his time that aluminium and sodium reverse their positions in the electropositive and negative series in these different mixtures, but modern chemists, I think, would explain this as a phenomenon of mass action, but it still leaves open the question as to whether alumina in a pure condition is or is not like water a non-electrolyte.

ALFRED H. COWLES

Seawaren, V. J.

New Mill at Manhattan, Nevada

The amalgamation mill of the Big Four company, at Manhattan, Nevada, was put in operation early in March. While this is a small mill, it is well equipped with modern machinery, and its operation is regarded as an important event in the history of Manhattan. The ore is free-milling. It is first crushed in a gyratory crusher, then in stamps. The stamp product is classified into sand and slime by Akins classifiers, the sand being reground in tube mills. Amalgamation follows tube milling. The capacity will be 120 tons per day.

Atlantic City Meeting of American Electrochemical Society

The spring meeting of the American Electrochemical Society will be held in Atlantic City from April 3 to 5 (Thursday to Saturday of the first week of April). Headquarters will be at the Hotel Traymore, and the professional sessions for the reading of papers will be held at the Solarium at the hotel.

Thursday and Saturday will be completely devoted to reading and discussion of papers, while on Friday, April 4, an excursion will be made to Philadelphia, leaving Atlantic City on the Pennsylvania Railroad in a special car attached to the 7:45 a. m. train and arriving at Broad Street station at 9:10 o'clock. The works of the Crucible Steel Castings Company at Lansdowne, Pa., will be visited in the morning where the making of crucible steel will be seen and a 2-ton Roechling-Rodenhauser induction steel furnace will be inspected.

At noon the party will return to Philadelphia and a visit will be paid to the University of Pennsylvania, where Dr. Edgar F. Smith, the provost of the University, will address the society.

After lunch visits will be paid to the Harrison Bros. Company on the Grays Ferry Road (leads, paint, lithophone, contact sulphuric acid) and to the United Gas Company Works at Point Breeze (water gas, illuminating gas, and gas testing laboratories).

The party will return to Atlantic City, leaving Broad Street station at 4:14, arriving at Atlantic City at 5:39 p. m. On the evening of Friday, after the lecture by Professor Kenrick, a smoker will be held at a place to be announced at the meeting.

The program of papers is as follows:

Thursday, April 3, 1913

10:30 a. m. Annual business meeting of the society, in the Solarium, of the Hotel Traymore; reports of the Board of Directors; announcement of the annual election; miscellaneous business.

The following papers will then be read and discussed:

Conduction and radiation of heat, by *J. Langmuir*.

Experiments with furnace electrodes, by *F. A. J. Fitzgerald* and *A. T. Hinckley*.

Aluminium nitride, by *J. W. Richards*.

Some tests of the Edison Storage Battery, by *C. W. Bennett* and *H. N. Gilbert*.

Concentration cells containing organic liquids immiscible with water, by *R. Beutner*.

Concentration changes in Copper Sulphate Electrolysis, by *C. W. Bennett* and *C. O. Brown*.

The 3-phase, 2-phase induction furnace, by *A. E. Greene*.

Making Electric Steel without Slag, by *A. E. Greene*.

On Friday, April 4, 1913, a lecture will be given in the Solarium, Hotel Traymore, by Prof. Frank B. Kenrick, of Toronto, on the subject of hyperphasia.

Saturday, April 5, 1913

10:30 a. m. Presidential address by President *W. Lash Miller*, introducing the symposium on the electrodeposition of metals.

Symposium on Electrodeposition of Metals:

Electrodeposition of Gold and Silver, by *Francis C. Frary*.

Electrodeposition of Copper, by *C. W. Bennett*.

Electrodeposition of Brass and Bronze, by *C. W. Bennett*.

2:30 p. m.

Electrodeposition of Cobalt and Nickel, by *O. P. Watts*.

Electrodeposition of Lead, by *F. C. Mathers*.

Solid thick lead cathodes from lead lactate solutions, by *F. C. Mathers* and *W. B. Cockrum*.

Registration will begin on the evening of Wednesday, April 2, in the lobby of the Hotel Traymore, and a meeting of the Board of Directors will be held on the same evening at 8:30 p. m.

Dr. W. Lash Miller of the University of Toronto, is the president of the American Electrochemical Society. Dr. J. W. Richards of Lehigh University, South Bethlehem, Pa., is the secretary.

Annual Meeting of the British Institute of Metals

Abstracts of Papers Presented

The annual general meeting of the (British) Institute of Metals was held in London on March 11 and 12.

The report of the Council states that the year 1912 has marked a period of material development in the activities of the Institute. The number of members is 614.

Corrosion Committee

From the report of the corrosion committee we note that considerable progress has been made in the investigations of the causes of corrosion of tubes of the four types of alloy selected by the committee, namely, 70:30 brass, Admiralty mixture, lead-bearing brass, and Muntz metal. Twelve tubes of each alloy have been tested in the special condenser plant for nine months, *i. e.*, from April to December, 1912. Three tubes of each composition have been withdrawn for detailed examination; this, however, has not yet been completed. A preliminary examination has shown that a small amount of corrosion such as that usually met with in tubes that have failed in practice, has occurred in some of these tubes, but has not penetrated to any considerable depth. All the tubes have been found to be covered with a scale of composition similar to that typical of those used in the mercantile marine. Investigation on these tubes is still proceeding.

The plant itself was closed down temporarily on Dec. 31, 1912, pending the supply of further funds for working expenses which amount to about £100 per annum. It is highly desirable that the experiments with this plant should be continued at the earliest opportunity, and this will be done as soon as the necessary funds are forthcoming.

An extensive scheme of laboratory experiments has been devised with the object of elucidating the nature of reactions which take place during the processes of corrosion and scale formation. It has been found necessary to continue a number of experiments for periods of several months, as the collection of data is necessarily slow. Hitherto it has not been possible to devise any satisfactory "acceleration" test, so work in this direction has been abandoned for the present.

A considerable number of badly corroded tubes have been sent in for examination by various shipping firms and manufacturers, and afforded useful information. The investigator is now desirous of obtaining a number of tubes that have endured exceptionally long service in marine or land condensers.

In the following we give abstracts of the papers presented at the meeting:

Metallic Filament Lamps

A paper on "Metal Filament Lamps," by Mr. Alexander Siemens, summarizes the history of the use of metallic filaments in glowlamps.

Owing to the uneconomical working of carbon-filament lamps, endeavors were made to replace carbon by one of the rarer metals, which can sustain a higher temperature than carbon without disintegrating.

Auer was the first to utilize osmium, prepared by a squirting process, and his methods were followed by several other inventors experimenting with various metals.

The first lamp to have actually drawn metal wire as its filament was the tantalum lamp manufactured by Siemens & Halske, Berlin. They also succeeded in drawing an alloy of tungsten and nickel, but before that process was perfected the General Electric Company, of Schenectady, patented a process to make pure tungsten ductile, which is described in the paper.

Corrosion of Condensed Tubes

A paper by Mr. Arnold Philips deals with "The Corrosion of Distilling Condenser Tubes."

Main and auxiliary condenser tubes become corroded on the sea water side; corrosion on the steam side of the tubes practically never occurs.

In distilling condensers the reverse of this is the case. No

corrosion occurs as a rule on the sea water side of the tubes but frequently occurs on the surfaces exposed to the steam.

The absence of corrosion of the sea water side in distilling condensers is apparently due to the facts that (1) the sea water flows outside the tubes, which tubes are generally placed vertically; on this account no particles of carbon, ash, etc., can find a lodgment on the surface of the tubes on the sea water side and thus what the author has shown to be one of the most important causes of corrosion is absent. (2) The tubes are usually in direct metallic connection with the casing, being expanded directly into the tube plates; this permits the full galvanic protective effect of any protector bars and of the casing itself when this is wholly or partially of iron or steel.

The corrosion on the steam side of distilling condenser tubes is due to hydrochloric acid in the steam.

The origin of the hydrochloric acid is the dissociation of salts contained in sea water, probably magnesium chloride. This dissociation only takes place to any noticeable extent when the primary steam tubes in the evaporators project above the surface of the brine which is being boiled.

The same distilling condenser used with two evaporators, one of which has steam coils projecting above the brine surface and the other having drowned steam coils, produces distilled water containing hydrochloric acid in the former case, but free from this acid in the latter case.

The effect of the acid-containing steam from the evaporator is to corrode the surface of the distilling condenser tubes and the distilled water obtained contains not only free acid but also chlorides of copper, zinc, and (if tin-coated tubes are used) also tin.

The presence of copper in the distilled water is particularly objectionable both for potable uses and also because when passing into the boilers it tends to set up further corrosion and pitting on the steel surfaces.

The cure to avoid these troubles is to use distiller evaporators which contain completely drowned coils. A palliative to prevent copper passing into the boilers with the distilled make-up water, if this contains copper due to a faulty form of evaporator being used, is to pass the distilled water through a zinc scrubber which removes the copper.

Chemical methods for detecting and measuring minute amounts of free hydrochloric acid in the distilled water obtained from distilling condensers are described.

Corrosion of Aluminium

A paper by Dr. G. H. Bailey deals essentially with the action of water and salt solutions on aluminium.

In the course of the investigation the effects of variation in the nature of the waters used and in the concentrations of the salt solutions have been examined. A large number of results are given illustrating also the dependence of corrosion on temperature and on the quality of the aluminium employed.

The author also gives a method by which reliable measurements of the rate of corrosion may be made, and from an examination of the experimental evidence he draws the following conclusions:

That aluminium of high purity is less readily acted upon than that of lower purity and that the presence of sodium and copper in the metal increase the rapidity of corrosion. Well-annealed metal is also more resistant to corrosion than unannealed metal. He finds that in general the corrosion of aluminium is a process of oxidation, and that as a matter of fact metal exposed for several months to water or salt solutions from which the dissolved air has been expelled underwent no corrosion whatever. The normal course of corrosion (excluding the action of acids and alkalis) is thus a transformation of aluminium into alumina, which separates out as a flocculent precipitate without any of the aluminium passing into solution.

Microstructure of German Silver

A paper by Mr. O. F. Hudson deals with the microstructure of German silver. Samples of German silver which had been annealed for varying lengths of time at a temperature of

7000 deg. C. were examined microscopically. The alloys which consist of a single solid solution, require many hours annealing at this temperature to bring them to a condition of true equilibrium when the solid solution appears to be perfectly homogeneous. German silvers when annealed thus commonly possess a characteristic "cored" structure which is independent of the normal twinned crystalline structure also seen.

The rather coarsely crystalline structure which results from the long annealing necessary to remove the "cores" is not in itself a sign of deterioration in rolling qualities. The results of tests on samples annealed for various times at temperatures up to 1000 deg. C. showed that, under laboratory conditions, the rolling qualities of good German silver are not impaired by severe annealing.

Hardness tests, using the scleroscope, showed that prolonged annealing, accompanied by pronounced crystal growth, does not lead to any decrease in hardness beyond that due to the normal annealing operation.

Effect of Rapid Cooling on Constitution

A paper by Mr. G. H. Gulliver deals with "the quantitative effect of rapid cooling upon the constitution of binary alloys."

The author shows how to calculate the proportions of the constituents of a rapidly cooled binary alloy by means of the information obtained in the usual manner from very slowly cooled alloys. The importance of the subject is due to the fact that the constitution of a cast alloy cooled at ordinary rates lies between that of the very slowly cooled and that of the very quickly cooled mixture, so that its limits of variation with change in the rate of cooling can be now specified.

The method of calculation is applied to the lead-rich alloys of the lead-tin series. The proportion of eutectic in a just solid alloy, the proportion of liquid and of solid in an alloy at a temperature between its freezing and melting points, and the rate of solidification of an alloy at a given temperature, when slowly and when rapidly cooled, are found numerically. The apparent form of solidus obtained from a rapidly cooled alloy is given, together with the variation in the apparent position of the saturation point which accompanies variation in the rate of cooling. It is shown that the position of the apparent saturation point depends not only upon the rate of cooling, but still more upon the composition of the alloy, and further that the proportion of eutectic present in a rapidly cooled alloy is sensibly independent of the curvature of liquidus or solidus.

The paper is accompanied by a number of diagrams which clearly show the differences between the conditions which prevail during slow and rapid cooling respectively.

Heat Treatment of Gun Metal

A paper by Messrs. H. S. Primrose and J. S. Glen Primrose deals with the "practical heat treatment of Admiralty gun metal."

As difficulties often arise in producing gun-metal castings to fulfil specified physical tests and withstand hydraulic pressure, the authors describe their experimental work with the object of finding a simple and reliable method of improving this commonly used industrial alloy so as to obtain the highest possible strength and elongation accompanied by greater soundness and homogeneity of the metal. They found that even in the absence of blowholes, which constitute the commonest source of unsoundness, cast gun metal behaves unsatisfactorily under hydraulic tests due to the presence of microscopical pores formed between two constituents of widely different properties.

A series of tests was carried out to investigate the practical value of various kinds of heat treatment, such as reheating, quenching, and annealing test bars of gun metal conforming to the Admiralty specification of 88-10-2, which had been cast in sand and chill molds and then cooled at different rates. The sets of normal and treated bars were tested physically and examined microscopically to determine exactly the effect of quenching from different temperatures and also of annealing for various periods of time at different temperatures.

The first set of tests made showed no improvement was

effected by reheating and quenching the bars, as the strength and elongation fell off rapidly as the temperature was raised. This result was unexpected as it is known that pure bronze is increased in strength by similar treatment.

More satisfactory results were obtained by simply annealing the metal at various temperatures, followed by moderately slow cooling, as this was found to considerably increase both the tensile strength and the elongation. The most marked improvement was found to be produced by annealing the bars for half an hour at a temperature of 700 deg. C., as the physical tests showed lower results both above and below this critical point. The time factor was also investigated and found to be critical for the $\frac{5}{16}$ in. thickness of metal used at thirty minutes, although only a slight improvement resulted after annealing for twenty minutes, and a slight diminution in strength occurred if annealing was too long prolonged.

The practical applications were discussed and examples quoted from practice in which material not otherwise defective failed under the hydraulic test due to water sweating through the microscopic pores, but after suitable annealing conformed to the most rigid tests. The increased homogeneity of the metal after the heat treatment was shown to be due to the absence of eutectic segregations from the microstructure, leaving only the strongly interlocking crystals of alpha solid solution of the tin and zinc in the copper. It was anticipated that this heat treatment would minimize corrosion troubles.

The headquarters of the Institute of Metals are at the Caxton House, Westminster, S. W., London. Mr. G. Shaw Scott is the secretary.

The Non-Ferrous Metal Market

No marked changes have occurred since our last report. Business has been fairly quiet, and the quotations were not realized on all transactions. At the latest reports the different markets have assumed a firmer tone, and it is expected that this condition will continue.

Copper.—The domestic market has been quiet and few purchases have been made. Most of the business has been effected at concessions under the quoted price. The bulk of the sales have been made to foreign buyers who have been attracted by the prevailing low prices. Little Lake Copper has been sold, and the market has not yet been firmly established on a 15-cent basis. The last quotations are 14 $\frac{3}{4}$ @15 cents for Lake, and 14.70@14.80 cents for electrolytic.

Tin.—Wide fluctuations have been noticeable in the London market, but have not been reflected in this market. The favorable February statistics gave a firmer tone to the March market, and liberal purchases were made, especially for future delivery. The latest quotation is about 48 $\frac{3}{4}$ cents, New York.

Lead.—A firmer undertone has prevailed in this market and sales have been heavier. The market is active at 4 17 $\frac{1}{2}$ @4.20 cents, St. Louis, and 4.32 $\frac{1}{2}$ @4.35 cents, New York.

Spelter.—Prices have advanced a little under the effect of a better inquiry for metal. Quotations have a wide range according to delivery, the last being 6.05@6.15 cents, St. Louis, and 6.20@6.30 cents, New York.

Other Metals.—Aluminum has shown a firmer tone since our last report, being quoted at 26@27 cents, New York. No material change is noted in the antimony market, prices ranging from 8 $\frac{1}{4}$ to 9 $\frac{1}{2}$ cents for various brands. The quicksilver market is quiet; the New York quotations being \$40 per flask of 75 lb. The San Francisco price is the same, except for export, the latter being \$37.50.

Nissen stamps have become popular machines for crushing ore in Rhodesia, where three new mills are being equipped with these machines. Roasting also is a feature in several of the new Rhodesian mills.

Tilting Crucible Melting Furnaces for aluminium, brass, bronze, copper, gold, silver and other non-ferrous metals are the subject of illustrated catalog 44 of the W. S. Rockwell Company, of New York City.

Evolution of Methods of Handling Slime—I

By H. N. Spicer

Modern methods of handling ore-slime, either in concentration or cyanidation, are comparatively recent developments in the history of metallurgy. It is necessary only to review the literature on this subject for the past decade to find that slime has been the source of much trouble, and that its successful treatment has been the subject of many investigations and inventions. The result is a comparatively high state of develop-



FIG. 1.—REMOVING SAND FROM COLLECTORS AT NUNDYDROOG MINE

ment in the art of handling this once troublesome material, and the successful extraction of valuable metals from it.

The methods employed in classifying, thickening, agitating and filtering slime probably are matters of common knowledge to well informed engineers; but it may be surprising to many to learn of the crude yet profitable methods employed in those countries where the latest mechanical appliances are not yet in use. The writer visited last year some of the important mining districts of the world, and being specially interested in the slime problem, took many notes and interesting photographs on the prevailing methods in the different countries. In India, as in many other foreign countries where nature labor is employed, the methods of operation may appear crude to the uninitiated; but it must be borne in mind that these methods have been determined by the prevailing labor conditions, and that those in authority deserve much credit for determining the economic limit to which this class of labor can be used.

Methods in Vogue in India

India at this time presents a particularly interesting phase of the slime problem, because the industry is in the midst of a period of transition from the old to the new. Prevailing methods of handling slime probably are the crudest to be seen anywhere in the world; and yet it will be but a short time until they are wholly abandoned in favor of modern methods. Already there is some evidence that the old order of things has



FIG. 2.—REMOVAL OF SLIME FROM COLLECTING TANK AT NUNDYDROOG MINE

passed, and that the strange operations shown in some of the accompanying illustrations will no longer be seen.

Gold mining has been a profitable industry in India for many years, but it is only within the last year that the first slime-treatment plant was erected. Prior to that time some sand had been leached with cyanide solution, but only such slime was treated as might remain with the sand and still give a

leachable product. The consequence is that large tonnages of slime await treatment at the different mines. It is probable also that a large portion of the leached sand is sufficiently valuable to be reground in tube mills and treated as slime.

The principal reason why so little attention has been paid to cyanidation in India is that the gold recovery by amalgamation has been so efficient. In the Mysore district, for example, from 82 per cent to 85 per cent of the gold in the ore is recovered by stamp-amalgamation. The sand and slime represent a value worth recovering, however, and they are being saved with that end in view; but now that cyanidation is about to be widely adopted, the slime problem is complicated by the fact that the ore is crushed in water, necessitating the dewatering of both sand and slime before further treatment.

The mill at the Balaghat mine, on the Kolar gold field, Mysore, is a very old one. The pulp from the stamps runs into concrete pits in the ground; the slime and water overflow, and when the pit is full of sand it is excavated by hand and stacked for future treatment. The slime flows to other concrete tanks, where the pulp is thickened by free settling to about 50 per cent moisture and then excavated and stacked by hand.

Sand and Slime at Nundydroog

Methods of dewatering at the Nundydroog mine represent a step in advance of those at the Balaghat; they are shown in



FIG. 3.—ELFVING SLIME FROM THICKENING TANKS AT NUNDYDROOG

Figs. 1, 2, 3 and 4. The stamp-mill pulp flows to steel collecting-tanks, which are equipped with Butters distributors, as shown in Fig. 1. When a tank has been filled, the flow of pulp is diverted and the sand allowed to drain. It is then dug out by hand, the work being done for the most part by women using a kind of hoe and a small wicker basket. Stacks of old tailings may be seen in the background.

The slime overflowing these sand collecting tanks passes through launders to a series of concrete settling basins, shown in Fig. 2. Here the slime settles to a consistency of about 55 per cent solids, after which it is elevated and stacked by means of a native contrivance known as a "deckool." Its operation will be better understood by reference to Figs. 2, 3 and 4. In Fig. 2 the bucket is shown lowered to the woman in the slime pond, who fills it, after which it is elevated as shown in Fig. 3 to a man who dumps it into a launder, Fig. 4. As most of the slime is too thick to run in any launder, it is pushed or scraped along to the dumping place by women provided with implements for the purpose.

The Ooregaum 80-stamp mill is shown in Fig. 5. At this

point reference may be made to the construction of mill and other mine buildings in the Mysore district, which are built of dressed granite, with walls 3 ft. thick. To one unfamiliar with local conditions, this must appear to be an extravagance; but as a matter of fact, the cost of this type of construction is less than for frame and corrugated iron.

The stamp batteries are arranged in two sets, facing each



FIG. 4.—DUMPING SLIME INTO LAUNDER, NUNDYDROOG MINE

other, and the combined pulp flows out at one end of the mill. Formerly the sand was allowed to settle in pits with walls of masonry, as shown in Fig. 6, the slime overflowing to settling basins. Later this was abandoned and the pulp conveyed in an open launder to a series of steel sand collecting-tanks, shown in the middle background of Fig. 7. At the head of the open launder are two spitzluten, by means of which from 10 per cent to 12 per cent of the coarsest sand is eliminated for future



FIG. 5.—OOREGAUM 80-STAMP MILL

regrinding. It falls to the ground and is taken to a dump by women, who fill the cars as shown in Fig. 6.

The remainder of the sand and slime passes on, as above indicated, to steel sand-collectors, Fig. 7, the slime overflowing to the concrete slime pits in the foreground. These pits have pyramidal bottoms, and are approximately 35 ft. square at the top, and 16 ft. deep in the center. Formerly the thickened slime

was pumped to the dump, but since the installation of a 30-leaf Butters filter the procedure is as follows: The slime is allowed to settle as much as possible and the supernatant water is pumped off by means of a portable pump shown on a truck at the right of the settlers. Cyanide solution is now added in quantity to give a pulp containing about two parts of solution to one of solids. Compressed air is turned into the pulp from the pipes shown extending from the left side to the center of the settlers. This agitation keeps the solids in suspension while the pulp is being pumped to a series of four 19 ft. by 10



FIG. 6.—ELIMINATION OF THE COARSE SAND AT OOREGAUM MILL

ft. Pauchuca agitators. After agitation the pulp is filtered in a Butters filter, shown in Fig. 8. This illustration is particularly interesting as affording a view of the first vacuum filtration plant erected in India. The filter builds a cake of about 1 1/16 in.; the cycle of filtration, washing and discharge occupies from 50 to 60 minutes.

Sand and Slime Separation at the Mysore Mine

At the Mysore gold mine, sand and slime are separated by the system shown in Fig. 9. The stamp mills are out of view at the right. The battery pulp flows through the launder shown supported on a trestle at the farther end of the row of tanks, and thence into two transverse launders built of masonry. Pipes leading from these transverse launders deliver the pulp



FIG. 7.—THICKENING SLIME AT OOREGAUM MILL

to Butters distributors placed over two rows of filter-bottom steel tanks. Only one row of tanks is shown in the view, the other being at the left. Each tank is provided with an overflow launder for slime, which passes to a series of concrete settling basins where it is handled in the same manner as at Nundydroog, as shown in Figs. 2, 3 and 4.

When a tank has been filled with sand, the feed is discontinued and the charge drained as dry as possible by vacuum applied under the filter-bottom. Naturally this system of separating sand and slime is not the most efficient, particularly in the be-

ginning of the operation when the tank is empty. The consequence is that the lower part of the charge contains more or less slime which affects the uniformity of drainage and leaves a variable quantity of moisture in the sand. After draining the charge as dry as possible it is discharged by hand through doors in the bottom of the tank, falling onto belt conveyors, which transport it to the dump shown in the hack-



FIG. 8.—PACHUCA AGITATORS AND BUTTERS FILTER AT OOREGUM MILL

ground. Extensive experiments are now being carried out to determine the best type of cyanide plant to build for the treatment of this ore.

First Complete Slime Plant in India

The first complete slime-treatment plant in India has been built at the Champion Reef. Fig. 10 shows the Champion Reef 180-stamp mill, with slime settlers and Pachuca tanks at the right. The battery pulp flows over plates and then to cone classifiers, the slimy overflow from which gravitates to two 50 ft. by 8 ft. slime-collecting tanks, which have 4-ft. conical bottoms, and in which the slime is thickened to a consistency of about 55 per cent solids. After decanting the water the slime is sluiced out with cyanide solution, and pumped to three 33 ft. by 10 ft. Pachuca tanks connected in series. Vacuum filtration completes the treatment.

Transition to Modern Methods

Turning now from past operations to present and future



FIG. 9.—SAND COLLECTORS AND CONVEYORS AT MYSORE GOLD MINE

plans, we find many instances of the transition to modern methods of handling slime. At Nundydroog a slime plant was designed last year and is now being erected. It comprises eight Dorr thickener classifiers, four tube mills, three 30 ft. by 10 ft. Dorr thickeners, four 38 ft. by 10 ft. Pachuca agitators and a 120-leaf Butters filter. At Ooregum the old clumsy method

of thickening the slime is to be replaced by Dorr thickeners. At Mysore experimental work is under way to ascertain what proportion of the ore should be treated as slime in an up-to-date plant. In all probability tube mills will be used, as well as the latest developments in mechanical classifiers, thickeners, and vacuum filters.

At the Jubtil mines of Anantapur, in the Madras Presidency, 60 miles north of the Kolar field, a 20-stamp mill is about completed which will be the first in India to adopt crushing in cyanide solution. This will greatly simplify the slime problem by eliminating the necessity for dewatering. The equipment includes stamps, amalgamating-plates, tube mills, Dorr thickeners, Pachuca agitating-tanks and Butters filter.

Near Raichur, in the native state of Hyderabad in the Deccan, is an old gold mine known as the Hutti (Nizam). It has



FIG. 10.—CHAMPION REEF 180-STAMP MILL, PACHUCA TANKS AT RIGHT

been operated intermittently for many years, the sand and slime being handled in as crude a way as in the old plants of the Mysore district. A modern mill is now being erected, to treat about 70 tons of ore per day, using a tube mill, Dorr thickener, and small Butters filter.

Throughout India the process of evolution in gold metallurgy, particularly in the treatment of sand and slime by cyanidation is very noticeable, and within a year or two the old ideas and methods will have been completely abandoned.

Denver, Colo.

Advantages of Small High Speed Electric Furnaces.

By Carl Hering

The question is often asked how much electric energy will be required to melt a specified amount of metal in a foundry for casting purposes. It is not generally understood, however, that to answer this question intelligently and to the satisfaction of the proposed user, there are other factors which one should also know as they may have an important bearing on the result. For instance, a small furnace which melts many small charges per day, and a large one which melts only a few large charges per day, can have the same daily output, and yet may require quite different amounts of electrical energy, hence different amounts per pound. The amount of energy required may also be materially different depending upon whether a given quantity of metal is melted rapidly or slowly.

The purpose of the present article is to analyze and discuss these two features of electrical furnaces, to show their significance and importance, and to illustrate the results with some estimated approximate quantitative data.

The user of a commercial furnace is not interested in theoretical efficiencies or in the so-called radiation losses, electrode losses, etc.; the efficiency which he is interested in is the amount of product produced per dollar; this may be represented by the simple formula $P/\$$, in which P is the amount of the product produced, and $\$$ is the total cost of producing that amount; the greater this "efficiency" the better he is pleased. Hence the duty of the engineer is to do everything and anything to increase this commercial efficiency, even though this may include some things which by themselves might appear to be bad practice because they are inefficient; if they increase this commercial efficiency they become good practice.

Some years ago the writer made the statement that the most efficient electric furnace is the electric fuse for protecting electric circuits when it "blows out" on a short circuit. Its efficiency, as a furnace, for sudden short-circuit currents, is probably high in the nineties, that is, very nearly perfect. The reason for this is that the heat is generated so rapidly that the metal is melted, and the intended process is completed, before any appreciable amount of the heat has had time to get lost.

This little "furnace," therefore, teaches us the importance of proceeding as quickly as possible not only in generating the heat, but also in doing whatever is to be done with the metal while it is melted, such as casting it, purifying it, mixing alloys, etc.; in other words, *to reduce as much as practicable the time during which each pound of metal must be kept hot*. During the whole of that time it is losing heat, and this "stand-by loss" increases in proportion to this time; hence if the melted metal can be melted and handled twice as quickly, this loss will be approximately halved. And as these stand-by losses are by no means small and insignificant, their reduction becomes important.

There are in general two ways in which these "time" losses in electrical furnaces can be reduced in practice. The first is to get the heat into the metal as quickly as practicable, that is, to force the furnace to the greatest practicable extent. This means to increase the power delivered to the furnace (the kilowatts, not the kilowatt hours) as much as practicable, and to get this energy into the metal as quickly as possible; in those resistance furnaces in which the heat is set free in the metal itself, absolute perfection has been reached in not losing any time in getting the heat into the metal after the heat has been generated, as it gets there instantaneously; in arc and other furnaces depending chiefly upon radiation and conduction it will necessarily take some time for the heat to get disseminated in the bath, and it is difficult to see how that time can be reduced except by very high temperatures and by rapid circulation. Furnaces which can be forced are, therefore, better in this respect than those which cannot.

The second way to reduce these stand-by "time" losses is to reduce the quantity of metal which forms one charge, to the smallest amount that the particular conditions warrant, so that the heated charge may be disposed of as quickly as practicable, thereby reducing the time that each pound of metal is losing heat. For instance, if the purpose is to cast into molder's flasks taking, say, 50 pounds each, then the ideal furnace would be one which melted a charge of about 50 pounds at a time; the losses in such a furnace would be less than in one which contained a charge of 100 pounds, as the latter has to hold the charge twice as long. Hence the size of the charge must be decided by the user, but his attention should be called to the possible economy of the smaller charges, as it would probably not suggest itself to the average foundryman; in fact, he would probably have difficulty in understanding it even after it was explained to him, as the best electric furnace practice differs from the customary fuel furnace practice.

In some cases it is important to have large charges, as, for instance, when an alloy of very definite proportions is required; it may be too troublesome to mix small batches, especially when samples have to be analyzed. Also when slag must be changed. A certain amount of time may also be necessary in certain refining operations. In all such cases the conditions govern what should be done. But even in such cases one should bear the above in mind and make the charges as small and the time of heating as rapid as the particular conditions permit, if economy of energy and the reduction in the size of the furnace are important.

By thus reducing the charges to the smallest which are practicable, and melting as quickly as possible, the size of the furnace may be greatly reduced, so much so that it may be possible in some cases to make it portable either on rails or suspended from an overhead track, thus enabling it to be brought to the flasks or other molds in a foundry, thereby permitting casting directly from the furnace into the molds; this not only

saves some of the superheat, probably a very appreciable amount, but it also saves labor, pouring ladles, heating the latter, etc. It may also make it practicable to enclose the whole furnace in a closed tank for heating in a vacuum or under pressure, for refining.

The ideal plant for a foundry, for instance, where regular work is done, would be a furnace which contained only about as many pounds of metal as are cast into one flask, and which melted this much metal in the time it takes to move the furnace from one flask to another and to do the pouring. For small brass castings, such a furnace would be small enough to be readily handled on a track or suspended from a trolley rail, and could be tilted so as to cast directly into the flasks. Such a furnace is now under construction for a brass foundry, though it was not considered wise to go to the extreme of the ideal; in the present case, for instance, the furnace will contain enough metal for several flasks.

It is by making use of every possible advantage of the properties of the electric furnace in ways like these that the greater cost of the source of heat may be more than overbalanced by other gains of money value.

Elsewhere the writer has shown¹ that the losses through the walls are in general greater per pound of metal, for small than for large furnaces, and that, therefore, furnaces should better be built of large capacity; yet the arguments given above seem to lead to opposite conclusions; the facts are that both conclusions are right when properly interpreted (see Figs. 1, 2 and 3 in the concluding summary).

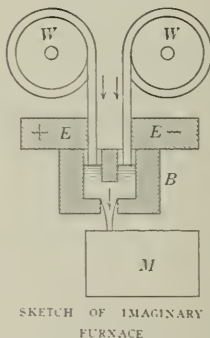
In general small machines are not as efficient as large ones, and yet from what was said above the contrary is true about furnaces. Again, a machine which is forced or overloaded is in general running less efficiently, and yet the contrary is here claimed to be the case for furnaces.

These apparent inconsistencies are due to the fact that with furnaces there are so many different factors involved which affect the result in different and sometimes contrary ways, that the problem sometimes becomes quite confusing. The only satisfactory way to study them is to determine the laws governing each

factor alone and then in various combinations. The writer has published at various times analyses and discussions of the laws of the losses through the walls, and those through the electrodes; in the present article the effect which the element "time," or speed of operation, has upon the losses is discussed with the other two sources of losses.

When problems become complicated because there are so many variable factors, it is often instructive and helpful to study extreme cases. In the present instance time or speed is the chief element under discussion; it appears that the greater the operating "speed" of a furnace the more economical it will be; imagine therefore a furnace in which this factor is carried to its extreme limit. Such an imaginary furnace is shown in outline in the adjoining sketch. Let B be a small block of refractory material with a very small U-shaped hole through it, and with the electrodes EE attached to a source of current. Let M be the mold into which the melted metal is to be cast and let WW be reels of wire of that metal capable of being fed rapidly into the holes. The current through the metal in the holes heats and melts it, and the melted metal immediately runs into the mold. Suppose the wires to be fed in as fast as the melted metal can flow out, and suppose the electric current supplies energy fast enough to do the melting at this rate and that the "pinch effect" did not interfere.

¹Heat Conductances Through Walls of Furnaces. Trans. Amer. Electrochem. Soc., Vol. 14, p. 225 (1908).



It will be seen that this would be an extremely small and highly efficient furnace; it would probably have the highest possible efficiency of any electric furnace, the reason being that the metal is melted and discharged into the mold so rapidly, and the quantity of melted metal in the furnace and therefore its surface, is so small, that there would be very little time or opportunity for the heat to get lost. This furnace is based on the principle of the electric fuse which, as described above, is the most efficient form of electric furnace.

This imaginary extreme case points out the directions in which practical furnaces should tend to be developed, namely, rapidity of melting and smallness of charge.

In order to get some idea of the possible economy gained by applying the above principles, a series of furnaces for melting brass have been calculated and the results are given in the accompanying tables and curves. Actual figures have here been assumed for the melting energy and for the rate of losses, the total being equal to their sum. But as these basic data are not yet known: to the desired degree of precision these actual figures are only roughly approximate and no great weight should be placed on them; it is to the *relative* and not to the *absolute* results that attention is here called; on the other hand, these relative results would probably not be greatly different for more accurate basic data.

The basic data which have been assumed here are as follows: When there are no losses of heat it is assumed to take about 8.25 kwh. to melt 100 lb. of brass, including about 10 per cent. more energy to superheat it. This figure is not yet known definitely; it may be and probably is somewhat less; it is, moreover, different for different brasses; if too high it is on the safe side, hence is conservative. The weight of brass is taken to be about 540 lb. per cu. ft. It is thought that the loss through the walls may for brass temperatures be taken as about 1 kwh. per sq. ft. of surface of the charge;² this is also only approximate, but is probably not far wrong; it would probably require careful designing to bring the loss down to this figure. The electrode loss is assumed to be 20 per cent. of the total; this is believed to be liberal. The hearth is assumed to be

TABLE I.

SAME CHARGES—DIFFERENT RATES OF MELTING

Note.—Brass. Volume of charge 3.70 cu. ft. Diameter of hemispherical hearth 2.42 ft. Loss from charge 1 kw. per sq. ft. Electrode loss 20%. Theoretical consumption for melting and 10% superheating of 100 lbs. brass, 8.25 kwh.

Charge in Lbs.	Time of Melting in Hours	Rate of Melting in Pounds per Hour	Kilowatt-Hours for Melting	Kilowatts for Melting	Loss Through Walls in Kilowatts	Loss in Electrode in Kilowatts	Total Power in Kilowatts	Efficiency in Percentage	Kilowatt-hours per 100 Pounds
2000	4	500	165	41.3	13.7	11	69	60	13.8
2000	3	667	165	55.0	13.7	17	86	64	12.9
2000	2	1000	165	82.5	13.7	24	120	69	12.0
2000	1	2000	165	165	13.7	45	224	74	11.2
2000	0.5	4000	165	330	13.7	86	430	77	10.7
2000	0.25	8000	165	660	13.7	168	842	79	10.5
2000	0*	∞	165	∞	0	0	∞	100	8.25

*Imaginary theoretically perfect furnace in which there is instantaneous melting, like in an electrical protecting fuse

hemispherical in order to reduce its surface, and therefore the wall loss, to the least possible. In most furnaces these losses would probably be greater, hence the relative differences pointed out below would in most cases be still greater.

In Table I and Plate I, the data are given for a series of furnaces for melting brass each of which has the same charge, namely, one short ton of 2000 lb., hence all have the same size of hearth, but in each of them the charge is melted at a different rate; the slowest rate, 500 lb. per hour, would be considered slow in a fuel furnace of that size, while the most rapid rate, 8000 lb. per hour, would probably be quite impossible in a one ton fuel furnace. The last one in the table is an imaginary theoretically perfect furnace, given merely for the purpose of showing the absolutely extreme case.

²Possible Reduction of the Power Consumption in Electric Steel Refining Furnaces. Met. & Chem. Engineering, November, 1911, p. 590.

The total required power in kilowatts is made up of that required for melting, plus that lost through the walls (a constant in this case), plus that lost through the electrodes, these being the only two unavoidable losses. These component parts are shown by dotted lines in the plate, the total sum of all three being shown by a thick line.

It will be seen that as the speed of melting is increased the power in kilowatts, of course, becomes greater and quite rapidly so at the higher speeds, but the losses become less in proportion, hence the economy becomes considerably better for the higher speeds, as is seen by the efficiencies and by the kilowatt-hours required per 100 lb.; the latter is the crucial figure of interest to the user as it represents his power bill. It will be noticed from the curves that in this case the latter varies by a straight

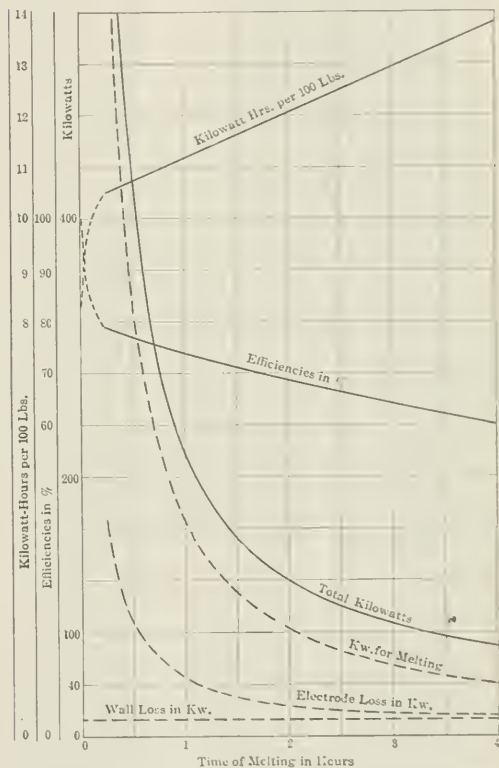


PLATE I.—RESULTS FOR A SERIES OF BRASS FURNACES WITH THE SAME CHARGE, BUT OPERATED AT DIFFERENT RATES OF MELTING.

The relation. It will also be seen that the reason why the efficiency is so poor for the slowest speed is because the losses approach in amount to the power required for melting.

Table I and Plate I therefore show that, having a furnace of a given capacity of hearth, it pays well to force it as much as other conditions warrant. When the user generates his own power, greater kilowatts (greater speed of melting) means a greater cost of generating plant, which therefore tends to offset the advantages, hence the usual "happy medium" must be determined.

Comparing a slow rate of 500 lb. per hour with one of 4000 lb. it will be seen that although the latter is eight times as fast, the power is not increased in that proportion, 60 kw. compared with 430, but is only about 6.4 times as great. The efficiencies are 60 and 77 per cent., hence considerably more favorable for the higher rate. The power bills will be as 13.8 to 1.07 for the same quantity of metal; hence almost 30 per cent.

greater per pound for the slower rate. Hence the melting capacity of the plant may be increased eight fold with only about 6¼ times as much power, and the cost of power per pound of metal will be much less. Large foundries could, therefore, be operated cheaper than small ones, and the latter might to advantage do their casting in the shortest practicable time. Another illustration is given in Figs. 3 and 4 in the concluding summary.

Having now shown from Table I and Plate I, that with a given furnace of any stated capacity of hearth, it pays well to force it as much as conditions permit, the next feature to be discussed is the question of the amount of the charge, hence the size of the furnace, as the "ideal" furnace indicates that the smaller it is the better. This may sometimes be decided by other considerations than economy of power, as for instance by the weight of a desired large ingot or casting for which the charge is intended, and in such a case the only rule is to make the charge as small as one of these ingots or castings and not large enough for several, provided this is not impracticable. But assuming that one is otherwise free to choose the size of the charge an illustration of the relative results is given in Table II and Plate II.

In this second set it is assumed that the rate of melting has been fixed, and it is desired to know whether this can be done best with many smaller charges or with a few large ones. Here again the data has been estimated for a series of different furnaces, besides the imaginary perfect one. It is here assumed that the required rate is 500 lb. per hour. The data then shows the losses, power, efficiency, etc., of furnaces of different sizes, any of which will melt at this same rate of 500 lb. per hour. The same basic data is used as before, and the hearth is again assumed to be hemispherical. In the curves the kilowatts for melting, for supplying the losses through the walls, and in the electrodes, are shown by dotted lines, and their sum, the total power, is shown by a heavy line.

As the melting is now assumed to be done at the same rate in all, in pounds per hour, the power used (the kilowatts, not the kilowatt hours) for melting is the same for all, and its curve is therefore a horizontal line. The loss through the walls (the total, not that per pound of metal) is now the factor which increases rapidly as the furnace gets larger; this is due to the fact that the metal is subjected to this loss for a greater length of time.

It will be seen from the curves how greatly the efficiency improves as the furnaces are made smaller, and how fast the

TABLE II.
DIFFERENT CHARGES.—SAME RATE OF MELTING

NOTE.—BRASS, Rate 500 lbs. per hour. Hemispherical hearth. Loss from charge 1 kw. per sq. ft. Electrode loss 20%. Theoretical consumption for melting and 10% superheating of 100 lbs. brass, 8.25 kwh.

Charge in Pounds	Time of Melting in Hours	Volume in Cubic Feet	Diameter of Hearth in Feet	Kilowatts for Melting	Loss Thru Walls in Kilowatts	Loss in Electrodes in Kilowatts	Total Power in Kilowatts	Efficiency in Per cent	Kilowatt-hours per 100 lb.
2000	4	3.70	2.42	41.3	13.7	13.8	68.8	60	13.8
1500	3	2.78	2.20	41.3	11.4	13.2	65.8	63	13.2
1000	2	1.35	1.92	41.3	8.7	12.5	62.4	66	12.5
500	1	.926	1.52	41.3	5.5	11.7	58.4	71	11.7
250	0.5	.463	1.21	41.3	3.5	11.2	55.9	74	11.2
125	0.25	.231	.96	41.3	2.2	10.9	54.3	76	10.9
		0	0	41.3	0	0	41.3	100	8.25

* Imaginary theoretically perfect furnace of infinitely small size, in which 2,000 lbs. are melted in 4 hours with continuous feeding and instantaneous melting.

kilowatt-hours per 100 lb. diminish as the furnaces are reduced in size; this is important to the user because that factor represents the power bill. (The curve for the latter happens to coincide with that for the kilowatts due to the scales here used; in all cases, however, the two quantities will be proportional under the assumptions here made.)

The conclusion to be drawn from Table II and Plate II is that for the same rate of melting, the smaller the furnace the more

economical it will be in power consumption; besides this the first cost will also be less. Hence the rule is to make the furnace as small as the desired charge will permit. If a certain ingot is desired make the charge equal to that one ingot and not to two or more. If the metal is to be cast into molds, make it equal to that for one mold; but as this may not always be practicable, then make it equivalent to as few molds as are.

As a concrete example taken from Table II, a 2000 lb. furnace may here be compared with a much smaller one for 125 lb., hence one-sixteenth. The large one will require about 69 kw. for 4 hours, the smaller one will require only about 54 kw. for the same 4 hours, and will deliver in that time sixteen charges of 125 lb. each, one every quarter of an hour, the rate of melting being the same in both. Hence there will be a saving of

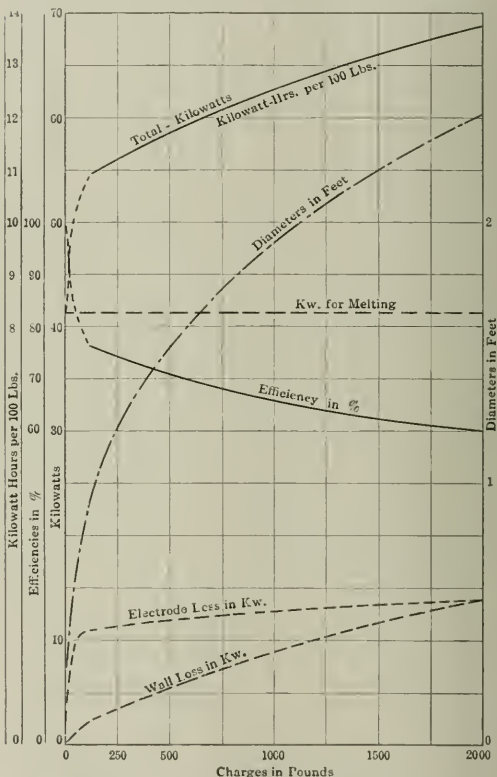


PLATE 2—RESULTS FOR A SERIES OF BRASS FURNACES OPERATED WITH THE SAME RATE, BUT WITH DIFFERENT CHARGES

about 15 kw. in favor of the smaller one. The energy consumption, and therefore the power bill, will be at the rate of 13.8 kwh. per 100 lb. for the large one, as compared with 10.9 for the small one, or about 27 per cent. better, in favor of the smaller one. Another comparison is given in Figs. 2 and 3 in the concluding summary.

As stated above, there will often be other conditions which will determine the limit to which one can go in increasing the rate of melting and decreasing the charge, and therefore the size of the furnace. The above discussion merely shows the directions in which one should tend to go in designing a furnace, other conditions permitting.

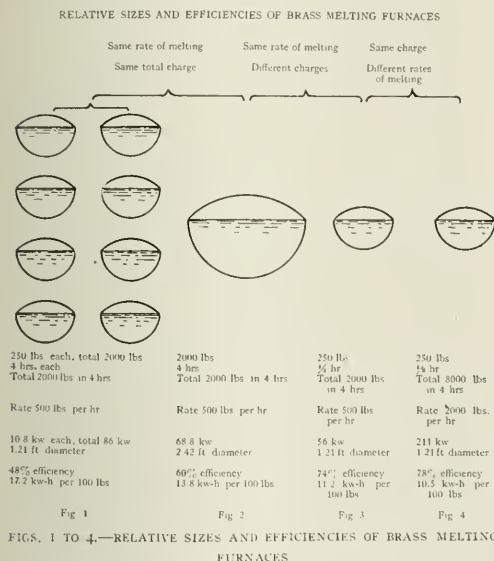
There are also some other minor factors which, for the sake of simplicity in an analysis which is already somewhat involved, are not considered here and which in some cases may affect the conclusion more or less; their importance should be examined in each case. It is true, for instance, that in a large slow-speed furnace the temperature is not at the maximum at the start,

hence the wall losses will at first be less. The extra labor of filling and emptying a small furnace frequently instead of a large one less frequently, should also be considered. On the other hand the very appreciable reduction in first cost of a smaller furnace for the same daily output, has not been considered here; nor the advantages of a reserve unit which may become practicable when the furnaces are small and cheap. Also the saving of labor, superheat, crucibles, their preheating, etc., when the furnace becomes small enough to be transported to the molds. The engineer must, of course, consider these and many other factors in his final decision.

The academician may find entertainment in expressing the above deductions in algebraic form. The writer believes that a series of concrete examples like those in these tables and curves, gives a clearer insight to the relations than abstract figures and letters do. Moreover, algebraic analyses are often burdened with complicated factors which in practice turn out to be negligible quantities.

Summary.—The general conclusions of this analytical investigation may be briefly summarized as follows:

The total consumption of energy per pound of metal, and



therefore the power bill, will decrease quite appreciably: (a) the more rapidly the melting is done, and (b) the smaller the charge, and therefore the furnace; both might be combined by saying: the less the time that each pound of metal is held in the furnace. This is based on the energy required for melting and the losses in the walls and electrodes. Such reductions in the size of the furnace also diminish its first cost and may in some cases make it portable so that it may be taken to the molds; the ideal melting furnace for small castings is one which melts as fast as it can be moved from mold to mold, and poured; for small brass castings such a design is not impracticable; the same may be true for small steel castings also.

No great importance should be attached to the approximate actual figures for brass melting furnaces given in the tables, although they are probably not far wrong; attention is here intended to be called chiefly to the relative values.

Pictorially these deductions concerning the relative values may be represented by the following diagrams Figs. 1-4, which are based in part on the tables and are drawn to scale. Another feature which was brought out in a previous paper is here included because it at first sight appears to be contradictory, but in fact is not.

Figs. 1 and 2 show that it is much more economical in power and in plant cost to use one large furnace instead of eight smaller ones of the same combined capacity of metal when all the smaller ones are being operated at the same time. This was described in a previous paper. Fig. 1 represents usual foundry practice in that a large number of small furnaces is used, all operating together. The rate of melting here assumed would be considered a moderately slow rate for fuel furnaces.

Figs. 2 and 3 (illustrating Table II and Plate II) show that it is much more economical in power and in plant cost to melt smaller charges at a time and to melt them in succession in the same furnace, though at the same rate of melting; the furnace will then also be considerably smaller. The smaller furnace of Fig. 3, though melting no faster than the large one, namely, 500 lb. per hour, will have the same daily output and will be more economical in power consumption.

In electrical parlance, Figs. 1 and 3 show that it is far more economical both in power and plant cost to melt the charges "in series" than "in multiple."

Figs. 3 and 4 (illustrating Table I and Plate I) show that it is much more economical in power to melt a given charge quickly, four times as rapidly in this particular case. The same is shown to a more striking degree by comparing the furnace of Fig. 3 with one of the eight furnaces of Fig. 1. The rate of melting assumed in Fig. 4 is about fifteen to twenty times as fast as the usual rate with fuel furnaces of that small size, and is approximately the rate used in a large 20-ton reverberatory furnace.

Comparing the two extreme cases with each other, namely, Fig. 1 and Fig. 4, it will be seen that Fig. 1 represents in general the usual foundry practice of to-day in so far as a large number of small furnaces are operated together; while Fig. 4 represents what would seem to be the best practice with electrical furnaces, namely, a single small furnace melting rapidly a large number of small charges in succession. A comparison of the data of Figs. 1 and 4 will give an idea of the very marked increase in economy obtained by the small, quick-acting furnace.

The furnaces of Figs. 1, 2 and 3 all have the same daily output, while that of Fig. 4 has four times as great an output per day, although it is no larger than that of Fig. 3.

Philadelphia, Pa.

Reform of Procedure in Patent Suits by the New Rules

By Dyer Smith

It is perhaps not fully known among manufacturers and inventors that a quite new method of procedure in the trial of suits for the infringement of patents went into effect Feb. 1. This change results from the new rules of practice for the Courts of Equity of the United States, promulgated by the United States Supreme Court.

New Rules of Practice for the Courts of Equity, Promulgated by the Supreme Court

The new practice will undoubtedly, in most cases, cause greater expedition and lessened expense to litigants, because of a general simplification of procedure, removal of causes for delay, and shortening up of times within which action may be taken. Especially, it is believed, will these beneficial results spring from the requirements of the rules that testimony shall be taken, in all usual cases, orally in open court, before the judge.

Under this new practice, the testimony is taken in one continuous session, in court, followed by the arguments of the lawyers, and the parties and the judge give their attention entirely to the matter at hand until its conclusion.

The rules now in force take the place of those originally promulgated in 1822 and 1842 and not changed greatly since the latter date. The present revision is the work of a committee of Supreme Court justices, aided by committees of lawyers residing in the nine circuits into which the United States

is divided. The changes which have gone into effect as the results of their labors are considerable, and, while some of the innovations will have to be "worked out," the new practice is clearly a greater boon to the owners of patents.

Under the old practice testimony was almost always taken out of court, in patent suits, and reduced to writing to be read by the judge after the hearing by him of the lawyers' arguments. It usually ran to a considerable length, and was generally more expensive and unsatisfactory, it is believed, than the testimony in open court of the new practice. The former methods were a continual cause of complaint to inventors and manufacturers, so that it was commonly said that a poor litigant had no chance in a patent suit against one well supplied with funds. The aim of the new rules is to remedy this condition as far as possible.

The first point to be noted in the new rules is that the technical forms of pleadings in equity, generally speaking, are abolished. Demurrers and pleas, always strongholds of delay, are done away with, the defenses formerly presented by these forms of pleading now being made in a simple way by motion to dismiss the bill or being incorporated in the answer. The answer of the defendant now merely sets out in short and simple terms the defense to each claim asserted by the bill brought by the plaintiff against him, avoiding general denials, but specifically admitting or denying or explaining the facts upon which the complaining party relies in his bill.

The answer must also, briefly and simply, state any counter-claims or set-off which the defendant may have, arising out of the same transaction as that upon which the bill of complainant was brought, or which might be the subject of an independent suit in equity against the opposing party, thus permitting various matters to be settled in one suit which might otherwise require the filing of cross-bills or bringing of separate suits by the defendant against the plaintiff. The bill of complaint, according to the provisions of the new rules, is to be framed in as simple and concise a manner as possible.

Not only are the pleadings simplified, but the times within which they can be filed are considerably shortened. The subpoena issued when the bill of complaint is filed is returnable in twenty days, to the office of the clerk of the court, and the defendant is required to file his answer or other defense in the clerk's office within twenty days after the subpoena has been served on him. Otherwise the bill may be taken *pro confesso*, unless the time be enlarged by a judge, for cause shown. When twenty days is not enough for the preparation of a defense, as in a suit on a patent in a technical art, involving the search for and study of many prior patents and publications, more time will, of course, be given, but the United States judges are all disposed to enforce the spirit of the new rules, and delay cannot be obtained against the opposition of the other side unless some good reason therefor is shown.

The pleadings will be concluded immediately upon the filing of the answer, and the controversy then deemed to be at issue, without the subsequent filing of a replication by the plaintiff, as formerly, unless the defendant sets up a counter claim or set-off in his answer, in which case ten days is allowed the plaintiff to answer this. The case is then ready immediately for the taking of testimony and trial.

As stated above, the testimony is now usually taken orally in open court. That is to say, a day for trial having been set and the parties having prepared their respective cases as fully as possible, in advance, the proceedings will be begun and usually carried through uninterruptedly to a conclusion before the judge.

The plaintiff's lawyer will first address the court, explain briefly his client's patent and outline the case of infringement he expects to prove. The defendant's counsel will then briefly outline the defense he expects to make. Complainant's witnesses will then be sworn, examined and cross-examined by the opposing counsel until the plaintiff's attorney deems he has won or lost a case. The defendant's witnesses will then be examined, and the defense made clear by the testimony, including, in most cases in which an unadjudicated patent is sued

upon, an attempt to prove the patent in suit invalid in view of certain prior patents or publications which the defense sets forth and explains. The plaintiff then immediately takes his rebuttal testimony, after which the opposing lawyers sum up and the case is left with the court.

This proceeding has taken only a few days, in place of the months and often many years consumed under the old practice in taking testimony in typewriting before a notary public or other so-called examiner, usually in the office of one of the lawyers. The judge has had the opportunity of seeing the witnesses while they give their testimony, of forming his opinions as to their frankness and apparent credibility, and of asking questions when a technical or other point is not clear to him.

He rules immediately upon any question of the admissibility of evidence which arises, and evidence which is clearly immaterial is not allowed to take up time in presentation and space in the record. When he feels that a point has been fully presented, and is clearly understood by him, he will indicate to the examining lawyer that nothing further is required along that line.

At the end of the trial the judge in most cases will have such a clear understanding of the matters at issue that he will be able to hand down his decision from the bench; or if he wishes to take the matter under advisement and hear further from the lawyers by written briefs as to any particular points of law which may have arisen, a shorter time additional than formerly will be required.

The saving of time in this respect will, of course, depend upon the judges themselves and the number of cases they have to hear. If a judge postpones the preparation of his opinion until after numerous other cases have been heard, the saving in this respect will not, of course, be so great. There is now, however, clearly the possibility of getting an opinion from the bench or shortly after the end of the trial, whereas formerly there could not be an opinion until after the judge had considered the records of testimony, etc., which was always done some time after the final hearing.

This form of procedure will doubtless reduce the cost of taking testimony very materially. The records of patent suits under the old practice were often "padded" with immaterial matter and almost always were longer than necessary. This was for the reason, first, that the examiner before whom the testimony was taken had not authority to pass upon the admissibility of evidence. Everything offered was taken down, together with the objections of the opposing counsel, if any.

The lack of power of the examiner was often taken advantage of, and the rules of evidence sadly mistreated by lawyers who wished to get certain testimony upon the record, regardless of its propriety in the case. This greatly increased the cost of the record of the testimony, exhibits, etc., which was usually printed before trial, and also the labor of the judge in reading through the same after trial.

Another reason for the longer records under the former practice was the difficulty of the lawyer in knowing when he had proved enough to make the matter comprehensible to the court, which would later derive its information by reading the record of the testimony. Another cause was the tendency of certain experts and other witnesses to spar and answer by lengthy indirection to embarrassing questions put. These exercises of mental gymnastics have sometimes consumed an excessive amount of time and will certainly not be attempted, or permitted if attempted, when the testimony is taken before a judge in court.

The causes above referred to have often resulted in patent cases in records of many printed volumes of testimony and in lengthy printed briefs submitted by counsel referring to or sometimes quoting at length from the most pertinent parts of the testimony. In such a case it is almost a certainty that the hard-worked judge will not fully read the entire record of the case, but will rely largely on the extracts brought to his attention by the opposing counsel.

Thus, under the old practice, we had the following elements in the trial of a patent case: volumes of testimony taken with

great toil and at considerable expense, with none of which the judge was familiar before the hearing; then the arguments of counsel before the judge, largely directed to giving him the fundamental information required to pass upon the questions involved; then a long period of waiting, in which the judge grappled with the printed records and briefs in whatever time he could find from his study of other records and the hearing of arguments in other cases. At the end of this period the decision was handed down. In one case in my experience the decision was not rendered for about fourteen months after the argument, so that the points made at the argument could hardly have stood out very clearly in the judicial mind at the time the opinion was written.

The new rules cannot fail to make a procedure immensely more rational and expeditious than the above. In one suit with which I am familiar, tried, by consent of the parties in open court in New York the latter part of January, the bill of complaint had been filed the previous October, just three months previously. The suit was on a mechanical device of some complexity, and the theory of operation of this device and those of the prior art set up by the defendant involved the consideration of various mechanical principles. The devices were explained fully to the court by working models, and the whole trial consumed only two days and a half. Decision was reserved in this case, but it is expected very shortly at this writing.

Depositions may be taken out of court under the new practice in exceptional cases only. They will only be permitted upon the application of the party desiring so to take the testimony in cases in which this mode is permitted by statute or when the moving party shows, by affidavit, a good and exceptional cause for departing from the general rule.

Certain sections of the Revised Statutes of the United States provided for the taking of testimony before an examiner, if desired, in the case of witnesses who live more than 100 miles away from the place of trial, and the Supreme Court rules cannot, of course, abrogate this statute. The rules provide, however, that if proper notice of the taking of such testimony has not been given the opposing party, the latter may be entitled to have the witness examined orally in open court if the court deems it proper under the circumstances.

If depositions are taken out of court, under a statute or order of the court, the names of all the witnesses must be given to the court in advance and the testimony must all be taken in a limited time, unless otherwise ordered by the court for good cause shown, sixty days for the plaintiff, thirty more for the defendant and twenty for rebuttal depositions.

The equity rules formerly in force also provided for limited time for the taking of testimony, but this was more honored in the breach than in the observance. As stated, the federal judges now are all determined to strictly enforce the provisions of the new rules with a view to expediting proceedings in all cases in which some good reason for delay is not made to appear.

Special provisions are made for the taking of the testimony of expert witnesses in patent and trade-mark cases. Such witnesses may, and in many cases will, be examined in open court according to the general rule. The court may, however, upon petition order that their direct testimony be set forth in affidavits and filed in a limited time; forty days for the plaintiff, twenty days more for the defendant, and fifteen days more for rebuttal affidavits. Should the opposing party wish to cross-examine any of these experts the court may order that such cross-examination and any re-examination take place before the court upon the trial, and unless the expert is produced for such examination his affidavit cannot be used.

Cases are now placed on the trial calendar as soon as the time has elapsed for taking and filing depositions, where these are permitted by the new rules. Thereafter no further testimony shall be taken by deposition except for some strong reason shown by affidavit. That is, even when some of the testimony is permitted to be taken out of court by deposition, as where witnesses live in a distant place, the case will go on the trial calendar after 110 days have elapsed since the case

was at issue. The federal courts all over the country are now calling these trial calendars, and patent suits which have been dragging for a long time are being disposed of or dismissed for want of prosecution.

The Rules of the District Court for the Southern District of New York

The District Court for the Southern District of New York, in which Manhattan Island is situated, have also promulgated new rules, which went into effect on Feb. 1, 1913. It is then provided that in patent cases each side shall be allowed only one expert witness unless leave to employ more be granted on cause shown.

This court also rules that expert witnesses, where their testimony is taken by affidavit as provided for in the Supreme Court rule referred to above, shall confine their testimony strictly to an explanation of the operation of relevant arts, processes, machines, manufacturers, or compositions of matter, and of the meaning of terms of art or science and of diagrams or formulas. They shall not give their opinion as to the meaning of any patent claim or specification, and if matter is hidden by this rule, or irrelevant or immaterial matter is contained in an affidavit, it shall not be answered by the opposite party nor be the basis of any cross-examination and the court may at any time strike such matter from the deposition.

This provision will doubtless result in much shorter expert depositions than most of these formerly taken. The rule is so novel that its actual working out in practice will have to be awaited before its relative merits and disadvantages can be accurately determined.

Another interesting paragraph in the New York rules is one which provides for the engagement of an expert to assist and sit with the judge at the hearing of the evidence in the trial of a patent case which in the opinion of the court involves intricate technical or scientific questions of fact. In such a case when all parties consent the court will appoint some disinterested person skilled in the art to act as an assessor, as it is termed by the rules, the reasonable fees for whose services are to be fixed by the court and charged as part of the taxable disbursements. The assessor will sit with the judge, assist the latter in his deliberations when requested and render the judge a written opinion when the court desires it, which shall be a portion of the record, on appeal.

It is believed that this rule should prove beneficial, provided a disinterested expert can be found acceptable to both sides in a given case. This may often be difficult, and many parties will prefer to await the trying out of the scheme before consenting to try it in their own cases.

The rule is designed, however, to meet the objection often urged by critics of our patent system that a judge, learned in the law, but not especially trained in technical sciences, should not be left unaided to decide important patent suits, the determination of which may depend upon a nice appreciation of intricate chemical or physical phenomena or a highly technical art.

It is a testimonial to the value of the legal training that judges lacking in special scientific education are as able as they are to lay hold of the important point of such a controversy and rightly determine the merits of the case.

Such an assessor, if broad minded and unbiased, will doubtless be of great assistance to the court in cases in which the opposing experts hold widely differing views. This official should not, of course, usurp the functions either of the court or of the witnesses, and will doubtless merely advise the court, upon request, in the interpretation to be given the testimony and the correlation of technical facts. The working of this innovation will be watched with great interest.

A considerable saving in expense to patent litigants under the new practice should appear in the matter of making up the record. The testimony taken in open court or before the trial need not be printed, as it formerly was in most cases, but will usually be taken down in shorthand by an appointed stenographer, and, if required, transcribed. When a case is appealed to the higher court the evidence to be included in the record

for the upper court will not be set forth in full and printed, as formerly, but will be stated in simple and condensed form, usually in the narrative manner; and those parts only which are essential to the decision of the questions presented by the appeal will be presented. The statement and condensation of the evidence for appeal will be made by the parties under the direction of the court, and all formal and immaterial parts of documents are to be excluded.

The system above briefly outlined is believed to be fundamentally sound and a conclusive answer to many of the criticisms of the former practice. Its advantages will appear in proportion as it is enforced by the courts and lived up to by the parties themselves. Unnecessary delays in patent litigation will occur chiefly when they are agreed to by both sides. Even in such cases suits will not be allowed to slumber in the peace of former years. The District Court sitting in New York City, for example, ordered some time ago that all suits in equity filed since 1902 be brought forth and placed on a special calendar, which was heard the first week in March. This resulted in hundreds of suits which had not been actually prosecuted for several years being dismissed. The same sort of a cleaning-up process is being carried out in the other District Courts of the country.

As to the possible disadvantages of the new system, it may be remarked that cases will have to be prepared in advance with the greatest thoroughness, and the defense anticipated by the plaintiff sufficiently to have the rebuttal witnesses primed and in attendance, since there will usually be no period of delay between the taking of defendant's proofs and the presentation of the rebuttal.

The hardship, however, if any, will be more than counterbalanced by the advantages, among which may usually be included the lack of opportunity to prepare defenses too fine-drawn to the facts and opinions educed by the opposing party. The cause of truth will probably be advanced when well-trained witnesses are given no considerable time to revise their views and testimony.

The fact that testimony in open court is not immediately typewritten, as in the usual former practice, but is buried for the day at least in the notebook of the court stenographer will also be embarrassing to some forgetful or uncertain witnesses, but will inure to the advantage of frankness and consistency.

The amount of gain in time required to secure a decision under the new practice will depend somewhat upon the calls upon the time of the judges. The actual hearing of patent cases, will, of course, consume more time now than formerly when in the usual equity case the judge only heard the arguments of counsel upon the evidence previously taken. On the other hand, the judge will be saved a good deal of time previously needed in reading the heavy printed records of testimony. In order to get the best results from the new system it may be that a number of additional federal judges will need to be appointed.

It seems clear that the Supreme Court has accomplished a constructive and well-considered reform by its new rules, for which the inventors and manufacturers of the country should be thankful. It is the more welcome since Congress has so merged and confused the inventor's demand for patent reforms with the supposed popular cry for greater regulation of all monopolies as to result in bills being introduced which would leave the inventor with less rights and remedies than before. There is probably little cause to fear that these bills will pass without material changes, because of the widespread protest against them on the part of patentees and technical interests, but even if they should pass a positive gain has undoubtedly resulted from the action of the Supreme Court.

New York City.

The gold output of the Rand to date amounts to a grand total of about \$1,685,000,000. Of this sum about \$44,375,000 has been paid in dividends. The production in 1912 broke all records, amounting to about \$180,000,000, and equalled about 40 per cent of the gold production of the world.

Iron Pipe Used for Electric Conduit

By Oliver W. Storey

The manufacture of rigid metallic conduit has become an industry of importance to the chemical engineer during the past decade. The competition among the manufacturers of this class of material has become very keen and has helped to develop a variety of methods of treating pipe to make it conform to the requirements of the National Board of Fire Underwriters. As the competition increased, the quality of the finished product increased and now the conduit is of a very high grade.

An electric conduit system may be defined as "a raceway in a building to provide for the ready introduction and removal of electric wires and to provide efficient mechanical protection to them."

This definition requires the following necessary features for such a conduit. It should occupy the least possible space; it should be rugged and mechanically strong, yet be easy to work and install; it should resist corrosive action and it should efficiently protect and preserve the insulation of wires contained in it; and it must be moderate in cost.

Materials possessing all of the qualities mentioned are available but their cost is prohibitive. Ordinary iron or steel pipe is cheap and possesses all of the necessary qualities except resistance to corrosive action, but by treating it to certain processes it becomes available for conduit purposes. It is the purpose of this article to describe these various methods of treatment and their comparative durability.

In the past many substances were used for the manufacture of conduit such as fibre, paper, rubber and moulded pulp. Metal pipe was also lined with paper, wood or compositions but these methods have been practically abandoned for the present practice.

At the present time butt-welded mild steel pipe 10 ft. long is used for conduit. This makes a cheap and excellent base since it is very rugged and can be easily worked.

The present manufacture of unlined conduit may be divided into two general classes, enameled and galvanized. The enameling affords protection to the pipe by a corrosion resisting film of enamel while the galvanizing affords protection by galvanic action.

It is only within the past few years that the sale of galvanized or so-called "white pipe" in contrast to the black enameled or "black pipe" has reached large proportions with a consequent decrease of production of the latter. This condition is caused primarily by the reduction of price of galvanized conduit with increasing competition and also by the general conservation tendency in the iron and steel trade to secure a maximum of protection even with greatly increased cost.

In this article the cleaning of the pipe previous to galvanizing or enameling will not be described. The subject of pickling of conduit pipe is an extensive one and has been discussed in another article.¹ It is assumed that all pipe is clean and in a condition to turn out a good grade of conduit.

Manufacture of Black Enameled or "Black Pipe"

The manufacture of "black pipe" appears as a simple proposition, but the large number of difficulties that appear in carrying out the process of enameling requires a great deal of technical skill to insure success.

The selection or compounding of a suitable enamel is of primary importance. Such an enamel when baked must conform to the specifications of the National Board of Fire Underwriters which are as follows: Section 58, Rule 1: "The pipe from which the conduit is made must be thoroughly cleaned to remove all scale and must then be protected against effects of oxidation, by baked enamel, zinc, or other approved coating which will not soften at ordinary temperatures, and of sufficient weight and toughness to successively withstand rough usage likely to be received during shipment and installation; and of sufficient elasticity to prevent flaking when 1/2-in. conduit is bent in a curve the inner edge of which has a radius of

¹Met. & Chem. Eng., Vol. 11, No. 1, p. 45.

3½ in. All conduit must have an interior coating of a character and appearance which will readily distinguish it from ordinary commercial pipe commonly used for other than electrical purposes."

Other large users of conduit issue specifications that are usually more rigid. Typical of these are the "Specifications for Conduit and Conduit Fittings" issued by the U. S. Navy Department of May 20, 1912. The sections relating to enameled conduit follow:

a. The enamel shall be of an approved type and be not less than 0.005 in. in thickness.

b. Not to be affected by moisture or at a temperature of 100 deg. C.

c. To be of sufficient elasticity to prevent flaking and to adhere with smooth surface when conduit is bent at normal temperatures in a curve, the inner surface of which has a radius of seven times the normal I. P. S. for ½ in. and to times the normal I. P. S. for all other sizes.

d. Not to be affected by the following after an immersion of 24 hours:

1. A 30 per cent solution of sulphuric acid.

2. A 30 per cent solution of nitric acid.

3. A 30 per cent solution of hydrochloric acid.

The specifications of the Board of Underwriters will allow much cheaper enamels to be used than those of the Navy Department. The latter's requirements are very exacting and only the best of enamels may be used.

The baking enamels used contain linseed oil as a base. The other constituents are tars, asphaltums, pitches, and gums, giving it its black color. The usual thinner is benzine. The enamels are made in many grades depending upon the amount of linseed used. By using a small percentage of linseed oil and a large proportion of asphaltums dissolved in benzine a very cheap enamel baking at a low temperature may be made. By using a high percentage of linseed oil with dissolved gums and asphaltum added an excellent one baking at high temperatures may be made. The addition of small percentages of other materials is practiced to improve certain qualities.

A method of testing for a good enamel is as follows:

Remove a strip of enamel from the pipe with a sharp knife. This strip should approach rubber in its elasticity and yet should be hard enough while on the pipe not to be easily scratched by the finger nail. Even after ageing for several years in a warm dry atmosphere this test should still apply.

The general methods used in conduit pipe enameling are alike and differ merely in details. The pipe is hung on racks and dipped into a tank of enamel, and after withdrawing and dripping is run into an oven and baked.

The clean, dry pipe coming from the pickle room are ready for enameling. The couplings on one end found necessary to secure complete separation of the pipes in the pickling vat are again found necessary to hang them upon the enameling racks.

The racks are usually made of cast iron with a double row of notches into which the pipe will slip easily but upon which the shoulders of the coupling will rest. With small sizes of pipe the notches may be deep enough to support two lengths. The racks are supported by a monorail system. The capacity of each is from 50 to 200 ½-in. pipe and requires a different rack for each size of pipe. Since a large percentage of conduit is ½ in. the number needed for the other sizes is small especially for those above 1 in.

After the pipes have been hung upon the racks they are dipped into the enamel which is usually contained in a riveted iron tank about 11 ft. deep, 2 ft. wide and long enough to accommodate the dipping rack. The tanks often contain 1000 gallons of enamel. With an enamel worth 65 cents per gallon the value of the contents is a considerable sum and precautions must be taken to prevent its becoming ignited, to prevent any deleterious foreign matter entering it or to prevent adding a poorer quality of enamel which necessitates testing of all new lots of enamel.

The danger of fire is increased by heating the enamel by steam coils in the bottom of the tank. The heating is neces-

sary to maintain a constant temperature and to thin the enamel. This method is safer, cheaper and gives better results than thinning with benzine. If a volatile solvent is added the enamel continually changes in consistency with corresponding variation in the finished product. Warm enamel also coats the pipe more evenly by eliminating any air bubbles clinging to the pipe surface.

The rack of pipe is lowered into the enamel and allowed to remain until it has reached the same temperature. It is then hoisted out of the tank and the pipe allowed to drip for about a minute after which they are run over dripping pans and allowed to remain until ready for baking. The excess enamel on the dripping pans flows back into the tank.

With the various grades of enamel this preliminary air exposure accomplishes different purposes. With a high grade, high temperature baking enamel this exposure to the air does not cause any hardening of the enamel but merely allows some of the excess enamel to drain. With a medium grade, benzine-thinned enamel the benzine evaporates, leaving a jelly-like coating that hardens in the furnace with medium temperatures. With a cheap enamel containing a larger percentage of benzine, the latter evaporates leaving an enamel that is but slightly tacky and which becomes hard with a slight baking.

After the preliminary air treatment the pipes are run into the baking oven. These vary in their construction and in capacity which is usually rated in terms of ½-in. pipe. An oven having a capacity of 25,000 feet of ½-in. pipe would hold about 4000 ft. of 4-in. pipe.

The design of the baking oven is dependent upon the enamel used. In general they are made of firebrick with ample ventilation for all fumes and gases evolved.

If a low temperature baking enamel is used steam coils may be placed in the bottom of the oven and adequate heat furnished by these to finish the drying of the enamel.

Where higher temperatures up to 700 deg. Fahr. are desired two general methods are in use, the direct and indirect.

The direct method is used where natural gas is available or other gas can be obtained at low cost. The gas is burned directly in the baking oven in a kind of hearth or brick that are heated to a white heat. The object is secure as complete combustion of the gas with as little excess of air as possible. The hot gases of combustion circulate among the pipe and bake the enamel. With small excess of air there is little danger of igniting the fumes evolved. With incomplete combustion soot would be deposited upon the tacky enamel and spoil it.

With the indirect method of heating the hot gases of combustion are led through radiators which heat the air to be used in baking the enamel.

The efficiency of the direct method is very high when compared to the indirect. With the latter there is no danger of spoiling the enamel with the products of combustion and there is no danger of an explosion should the oven door be accidentally opened during the "heat."

When a medium enamel which hardens slowly in the air is used the oven is run at a temperature of 300 deg. to 400 deg. Fahr. When the oven is first warmed a small amount of enamel may drip from the bottom of the pipe due to the thinning of the jelly-like film left after the evaporation of the solvents.

With one that must be baked at a temperature of 600 deg. Fahr. the operation is not as simple. When the oven warms the enamel thins and the excess will drip to the bottom of the furnace. This dripping will continue for some time, usually until the solvents have evaporated and the linseed oil has begun to harden. The excess is drained into a well from which it is periodically returned to the tanks, apparently having suffered no deterioration.

The securing of a uniform film of enamel over the surface of the pipe is a difficult problem. After baking a rack of ½-in. pipe, that have received one coat of enamel, the following imperfections will be noted:

1. The upper end of the pipe, both interior and exterior, has received a thinner and harder coating than the lower end

2. The threads at the lower end are filled with enamel.
3. The surface of the exterior of the pipe that is very close to another pipe has a coating thin enough to be transparent, while the balance of the pipe has a heavier coating.
4. The interior surface of a piece of $\frac{1}{2}$ -in. pipe has received a thin coat while the same surface on a piece of 4-in. pipe is as heavily coated as the exterior.

These are the chief difficulties encountered in enameling conduit.

The coating at the upper end of the pipe is usually half as heavy as that at the lower end. This is caused by the upper part of the enameling oven becoming hotter first and the draining of the surplus enamel to the bottom of the pipe. The enamel flows slowly to the lower end, and, before it can all drip, it has begun to harden since the rate of heating is usually high. With slower heating this difficulty could be lessened. At the upper end of the pipe it is also harder than at the lower owing to the thinner film and higher temperature.

These imperfections may be remedied in the second application by upending the conduit on the racks. This is not generally done owing to the extra operation of changing couplings which would require that the racks be unloaded and reloaded. Some manufacturers depend upon the second coat being more uniform than the first and covering any pronounced difference in the upper and lower end.

The unevenness of coating due to the proximity of other pipe can obviously only be remedied by cutting down the capacity of the oven. Since the second coat of enamel covers this defect it is usually disregarded. If but one coat of enamel were used this defect would have to be remedied.

The difference of thickness between the interior and exterior coat is primarily caused by the lack of circulation in the interior of the pipe and consequent slow evaporation of the solvents and slow oxidation of the oils, allowing more enamel to drain. The enamel film is also very thin at the top when compared to the bottom. As the size of the pipe increases these difficulties decrease.

The two coats of enamel are necessary to give a flexible coat of suitable thickness as required by the specifications and to overcome some of the defects mentioned. A coat of the requisite thickness could not be applied in one baking for the same reason that a thick coat of paint is never used.

The second coat is applied in the same manner as the first and results in a bright smooth surface. This coat is usually baked at a slightly higher temperature than the first giving a harder surface, yet flexible owing to the softer foundation. A third coat is sometimes required but such an additional film is of doubtful value as an added protection.

The conduit is now ready for the market after an inspection of the bore to discover obstructions. This inspection is carried out by flashing a light through it.

The enamel interior provides a smooth raceway for wires. Its generally excellent appearance, its cleanliness, smooth interior and its cheapness makes the black pipe very popular among the installers.

The racks used for dipping the pipe soon become covered with enamel and unfit for use. This can be easily removed by burning in a scrap fire.

Elbows are finished by running wires through the bores and dipping them as with pipe.

The enameling plant should be so planned that the process may be carried on without interruption during the different operations. The layout should allow enough space to provide for the hanging of an oven-full of green pipe and enough dripping space to take care of an oven-full of dipped pipe. The mechanism should be so arranged as to allow a batch of baked pipe to be removed from the oven while a second batch is run into it. During the baking of this, the first can be given its second dip and allowed to drain.

Since the quality of the enamel is dependent in a large measure upon the temperature control of the ovens, recording gauges should be used to keep an accurate record of the temperature at various parts.

Methods of Manufacture of Galvanized or "White Pipe"

While the general practice in the manufacture of "black pipe" is essentially the same at all plants, the practice in the "white" or galvanized pipe industry varies widely. This difference is due primarily to the three galvanizing methods used, these being hot galvanized, electro-galvanized and sherardized. Approved conduit of all these three types are on the market, the electro-galvanizing predominating since the one sherardized conduit is patented, and there has been but one hot galvanized conduit approved by the underwriters.

The specifications of the Board of Underwriters have been quoted and are easily complied with. In direct contrast are the rigid specifications of the Navy Department which exclude hot galvanizing.

"2. h. All steel conduits shall have the exterior surface galvanized and the interior surface either enameled or galvanized and enameled. All galvanizing shall be done either by electro-plating process or dry galvanizing process, and shall show no signs of rust when conduit is immersed in warm salt water (90 deg. C.) for 48 hours. The outer surface of conduit shall be free from paraffin or other material which would tend to prevent the adhesion of paint. Before any tests are made, conduit shall be washed thoroughly in carbon tetrachloride.

"3. d. 4. When conduit is both enameled and galvanized on inside tests under 3. (d) 1, 2, and 3 (acid tests) will not be applied."

The chief trouble that has been encountered in the manufacture of hot galvanized conduit has been the wiping of the interior of the conduit to secure a thin, flexible, and adherent coating. The present hot galvanized conduit on the market has a smooth wiped interior and exterior surface while the threads are clean, yet well coated and need no recutting. The coating is somewhat heavier than is found on electro-galvanized or sherardized pipe yet it is flexible and adheres well to the iron base.

This coating contains aluminium in varying amounts, often being as high as 10 per cent. The addition of this metal improves the hot galvanizing bath without materially increasing the cost owing to its low specific gravity. The aluminium gives the finished conduit a pleasing white appearance.

The presence of about 4 per cent of aluminium reduces the melting point of zinc from 788 deg. Fahr. to 716 deg. Fahr., and allows the bath to be run at lower temperature and decreasing the amount of zinc-iron alloy formed. Since 4 per cent is the eutectic percentage any other low percentage of aluminium will make the solidification occur over a shorter or longer range of temperature depending upon the composition and will, therefore, make it easier to wipe.

The interior of this hot galvanized conduit is not enameled but a peculiar blue black wash is used. It is a thin coating and is merely used as a distinctive finish and helps the fishing qualities slightly.

The one sherardized conduit is made by the sherardizing process and has a coating of zinc-iron alloy on the interior, exterior, and threads. While it is not as heavy as that obtained by the hot galvanizing process it is much heavier than that obtained with the electro-galvanizing. It adheres firmly to the iron base, does not flake and is hard, making it capable of withstanding rough usage.

Analyses of the zinc-iron alloy made at different times will show varying amounts of iron present. The iron varies with the metallic zinc in the zinc dust, thickness of coat, sherardizing temperature and length of time in the furnace. The amount of iron in the alloy has been subject to a great deal of discussion since the different investigators have found widely varying percentages. Patrick and Walker¹ found 11.70 per cent, Leighou and Calderwood² 18.15 per cent, while Cushman³ found about 30 per cent. In none of these cases was

¹ *Trans. I. & E. Ch.*, Vol. 8, No. 4, p. 219.

² *Reports of Tests on Sherardized and Electro-galvanized Conduits*, Carnegie Inst. of Tech.

³ *Bureau of Zinc Coated Conduit Pipes*, Inst. of Industrial Research, Washington.

it known under what conditions the alloy was formed. From a large number of analyses made by the author it was found that the above variables determined the iron content of the alloy formed. By controlling these factors any zinc-iron alloy containing from 8 to 30 per cent of iron may be obtained.

Sherardized conduit receives an exterior and interior coat of clear enamel. Previous to this practice a coat of clear enamel was put upon the interior only and the exterior was paraffined. The clear enamel was used to aid fishing while the paraffin was used to lay the objectionable loose zinc dust that remained on the pipe.

Electro-galvanized conduit is made by varying methods. Only the exterior is plated since the plating of the interior has not been put on a commercial basis. Usually the threads have little or no coating on them, the interior of the couplings are rusty and elbows are poorly coated. With one exception the zinc coating is thin.

One brand of conduit is made by flashing the cleaned iron pipe in a copper bath and plating over this copper with zinc.

Another manufacturer obtains a smooth velvety adherent coating by wire brushing the pipe before coating with zinc. A second brushes the conduit after plating and obtains a bright finish which is pleasing to the eye and does not discolor easily.

All "white pipe" is enameled in the interior to help the fishing and with the cold galvanizing to provide against oxidation of the iron. In the Navy Specifications the acid tests do not apply to the enamel in the interior for that conduit which has a protective zinc coating beneath. This allows a very thin enamel to be used by those manufacturers who apply zinc to the interior wall while a good grade giving a heavy film must be used by others. This somewhat equalizes the costs.

The coatings used in the interior of conduit cover a wide range. The colored wash in the hot galvanized and clear enamel in the sherardized have been mentioned. Electro-galvanizers use a black enamel except one who uses a red iron oxide enamel paint which is of an excellent quality.

To enamel or paint the interior of conduit requires a special apparatus. A tank similar to the one used for black pipe is used. The pipes are let down into this on racks. The enamel is stored in a small tank about 10 to 15 ft. above. A circular spraying nozzle is let into the top of the pipe. A valve allows the enamel to flow and the head of enamel sprays it on the interior wall of the conduit. Enough is introduced to thoroughly cover the entire interior surface. The surplus drains to the bottom of the tank from where it is pumped back to the storage tank to be used again. This method of enameling is very effective and one man can work at the rate of 600 lengths of pipe per hour. The remaining procedure is similar to that of the black pipe.

Enameled or painted "white pipe" is now sold extensively. The enameling of this pipe does not present any difficulties beyond those found with the black pipe. The applying of a paint presents difficulties of a different nature. The pigments used settle quickly and some method must be devised to keep it evenly distributed throughout the oil. The stirring may be accomplished by means of a propeller. Even with stirring the conduit will not be evenly coated by the pigments. The more practical way of applying the paint would be by means of a spray. In this way the pigment would be more evenly distributed and not be washed away as it is when the conduit is withdrawn from the tank.

When sherardized pipe was paraffined the following method was used. A solution of paraffine of the requisite strength was made in warm benzine. This solution was contained in a horizontal tank capable of holding 5000 ft. of $\frac{1}{2}$ -in. conduit. Steam coils kept the solution hot to prevent the paraffine from freezing out. About 5000 ft. of $\frac{1}{2}$ -in. pipe, after enameling the interior, were dipped into this tank in one bundle. Any superfluous dust was washed off and settled to the bottom while a thin layer of paraffine remained on the pipe after the benzine had evaporated.

After the conduit has been galvanized and enameled it is inspected, labeled, banded and is ready for shipment.

Black Pipe versus White Pipe

At present the relative merits of the various brands of white pipe are subject to a large amount of controversy. The "black pipe" has been almost entirely neglected, it being generally taken for granted that it is an inferior conduit. Large numbers of tests of galvanized pipe have been made both by partial and impartial investigators and the results made by the former are contradictory while those made by the latter agree fairly well.

If the "man on the job" were asked what pipe he would prefer there would be but one answer which would be "black pipe." He prefers it because of its clean, smooth surface, superior appearance, excellent fishing qualities and because it "takes" paint. He objects to the rough surface of the electro-galvanized because of the "acid" which he says eats into his skin. He objects to the hot galvanized pipe because of its rough surface and some even complain of the "acid." The sherardized paraffine finished pipe was objectionable because of the paraffine finish which became soft in warm weather. Without the paraffine the conduit was dusty.

The stiff competition in conduit manufacture has raised the standard to a high plane. Where, about six or seven years ago, most any kind of enameled pipe would have been acceptable, now this is no longer true and a good quality has to be made. By way of confirmation, one only needs to look at conduit installations of a few years ago and compare them with those of to-day.

As stated above the durability of the different kinds of "white pipe" has been the subject of much argument. The relative merits of black and galvanized pipe are not often mentioned and the writer wishes to emphasize the high merit of the former. Each class of pipe is best under certain conditions and before an installation is made these should be studied to obtain the greatest service per unit of cost.

Conduit is subject to a wide range of conditions. Outdoors it is subject to the action of the weather; it may be buried and here a variety of conditions may affect it, ranging from a dry to a wet, acid soil; it is subject to temperature extremes; it may be buried in cement, cinder, concrete or various plasters; it may be exposed to corrosive vapors, coal smoke, or gases; it may be attacked by oils, acids or alkalis, and it is always subject to mechanical injury in transportation during installation and sometimes in service.

Prof. C. F. Burgess¹ has shown that a high grade enameled conduit, the surface of which has not been injured, is superior to galvanized conduit under a variety of corrosive conditions. Those cases coming under the writer's observation have confirmed this, but only where a good grade of enamel was used. Any enamel that conforms to the Navy Specifications as given above must be resistant to corrosive influences.

But under ordinary conditions of shipment and installation the exterior enamel is scratched and scraped, exposing the iron beneath and subjecting it to corrosion. This condition is necessarily important when comparing with galvanized conduit.

Under the following conditions enameled pipe is cheaper to install and will be fully as satisfactory as the galvanized. In room interiors where the conduit is afterward painted, under similar outdoor conditions, in walls of buildings including bath plastered and concrete where moisture is not present; in short, any place where dry conditions prevail and any exposed iron is not subject to permanent corrosive conditions. This would cover a large percentage of installations.

By painting the enameled pipe after installation it could be made superior to non-enameled galvanized conduit under many conditions. Where the latter is installed and is exposed to acids and fumes the galvanizing soon is corroded. The resistance of unscarred enameled pipe to such conditions has been pointed out by Prof. C. F. Burgess.² By giving the scarred places a coat of slow air drying enamel after installation, the conduit could be made comparatively permanent.

Under exposure to alkalis there would be an advantage to the galvanized conduit but under such conditions even the latter

¹Sherardizing Magazine

shows a high rate of corrosion and brass, lead, or composition conduit are necessary. Under conditions where the application of this enamel after installation would be impractical or under peculiar conditions of exposure where the enamel would blister and peel the galvanized conduit would be better suited.

The importance of securing as permanent an installation as possible in modern buildings even at a much higher cost has been recognized by some engineers and architects. These men have specified conduit that is both galvanized and enameled, and, so insistent has been this demand, that manufacturers are turning out this class of conduit.

The importance of such a protecting coat over the galvanizing was first noticed in the testing of the paraffined sherardized pipe. The paraffine was primarily used for laying the dust but at the same time it increased the resistance to corrosion to a remarkable degree. It seems as though this increase in the life of the sherardizing would have been the important feature but was not recognized as such by the manufacturers. Tests^{2,3} on paraffined and non-paraffined conduit showed the marked superiority of the former, being fully as durable as the enameled conduit.

The writer believes that any enamel that is used over the galvanizing should be equal in quality to that used on "black pipe." A cheap enamel will soon disintegrate and leave the galvanizing exposed. And where such a conduit is installed the scarring of the enamel during installing will make it little better than the raw galvanized material. The painting of all scars is very important and should be rigidly enforced if the maximum of service is to be received.

The U. S. Government has recognized the value of such a double protection, especially on war vessels, and along the Panama canal, where galvanized conduit, usually sherardized, is painted after installing and is repainted regularly.

The value of the different galvanized coatings when used for conduit protection is a much discussed subject. With the constantly improving methods and with the new brands continually appearing, no data are ever complete.

Some of the earliest work was done by Prof. C. F. Burgess³ at the Chemical Engineering Laboratories of the University of Wisconsin and showed the relative merits of hot, cold and dry galvanized and enameled conduit. Messrs. Leighou and Calderwood⁴ of the Carnegie Technical Schools made some exhaustive tests in 1912 of the value of electro-galvanizing and sherardizing as a protection for conduit purposes.

The value of galvanizing is usually determined primarily by the amount of zinc per unit of area. The other important factors are homogeneity, purity, continuity and tenacity with which it clings to the iron.

The cold galvanized conduit is made by several manufacturers and the product varies. The first of such conduit showed up poorly under actual service conditions as well as in accelerated tests. This was the result of the light deposit of zinc and its uneven distribution. The threads and couplings were usually subject to immediate corrosion owing to the absence of zinc. The amount of zinc has been materially increased but even now the threads and couplings rust quickly. The writer has under his direct observation a large amount of cold galvanized conduit of recent installation and every length is badly rusted at the joints.

The copper flashed conduit employing a thin zinc deposit has shown no superiority over the ordinary electro-galvanizing under actual service conditions.

The sherardized conduit has been subject to many tests and has shown that when the alloy is formed under proper conditions it results in a coating that is superior to the electro-galvanized or hot galvanized material. The present practice of sherardizing shows a lack of uniformity of practice and results obtained. The theoretical alloy that should be obtained is one corresponding to FeZn_{10} , containing about 8 per cent of iron. Where this alloy has been obtained a coating highly resistant to corrosive action has resulted. The writer's experience has shown that a low temperature is necessary for the production of such an alloy, probably lower than any temperature

used in commercial sherardizing plants of today with but one or two exceptions.

With the increase of iron content of the coating the more quickly will it corrode. With coatings containing from 25 to 30 per cent of iron the writer has found that the sherardizing affords little protection since it pits easily. Such a coating is obtained by using high temperatures and a lean zinc dust.

A peculiar property of this alloy is the formation of a blue black covering after being exposed to the weather for some time. Cushman⁵ calls attention to the highly protective powers of sherardizing, through its ability to take on this highly resistant black coating, which he explains as being the formation of Fe_3O_4 from the oxidation of the iron in the alloy. He states that this oxide will greatly increase the life of the coating owing to its superior weathering qualities. This black appearance is especially found on exposed paraffined conduit.

The successful introduction of the hot galvanized conduit is of great importance since it is almost impossible to obtain a thin coating with this process. This conduit should, if the weight of coating and other factors are considered, have a long life. Hot galvanized pipe stands high in corrosion tests.⁶ What effect the introduction of aluminium will have can only be conjectured.

The importance of the interior galvanizing is often over-estimated. In most cases few fumes and vapors come in contact with the interior and a coat of good enamel should be sufficient to protect against all corrosive influences.

The testing of galvanized conduit is usually done by the standard Preece copper sulphate method and the coats are usually spoken of as being so many "dips" in thickness.

The writer has tested the Preece method and has found that it is very reliable, cheap, and entirely satisfactory as a quick factory method for testing galvanized conduit. The heaviness of the coating is a straight line function of the number of dips and makes the factory control very simple.

But results obtained by this method on two different types of galvanizing are not comparable. It will be found that sherardizing of the same weight as electro-galvanizing will give twice the number of dips in copper sulphate.

The writer has often heard the statement that a coating standing one dip will be equal to three years of atmospheric corrosion. On the strength of the copper sulphate tests resulting from the above statement, a large amount of conduit has been sold. Some manufacturers, quickly realizing that the addition of certain materials to the coating would raise the Preece test without materially increasing the coat, began this practice. As a result of this the test when used by the salesman is practically worthless.

To arrive at a fair basis, depending upon the actual amount of zinc present, a new method has been proposed.⁷ In this method the zinc or zinc iron alloy is removed by basic lead acetate. This method is very satisfactory where an accurate determination is desired and in the comparison of two unlike conduits. For routine factory testing it is not practicable since it involves several weighings with the balance. It may be used in occasionally checking up the copper sulphate tests. The actual amount of zinc present will not be indicative of the serviceability of the conduit for this will depend on the process.

From this description of the making of rigid conduit after leaving the tube mill the process is seen to be a product of the chemical engineer and electrochemist. Unfortunately they have not been instrumental in the early development of the processes that have only received their aid during the past decade.

How long the galvanized conduit will be supreme is a question of when some other cheap resistant coating may be applied to iron or some other resistant metal or alloy may be produced cheaply enough to compete with steel. The possibility of using a lead coating instead of zinc is an attractive proposition. Samples of such conduit have come to the notice of the writer from time to time and to all appearances should make an excellent conduit.

Chemical Engineering Laboratories, University of Wisconsin,

The Determination of Vanadium in Ferro-Vanadium

By Wm. W. Clark

As all ferro-vanadium is sold on a basis of the vanadium content, the determination of vanadium is the most important one in the analysis of the alloy. It is not a difficult determination, but is one that is easily influenced by conditions, and consequently very few analysts get correct results by a few tenths of a per cent.

A number of methods have been introduced for the determination, which when run on pure salt, give fairly concordant results, but when an alloy of indefinite composition is encountered, the chances of error multiply, or if the vanadium is separated from the other metals present, considerable time is wasted, with corresponding losses of vanadium in the separation. The majority of the methods depend upon the oxidation of the vanadium from the vanadyl to the vanadic salt or vice versa. Some of these methods are indirect, such as the reduction of the vanadic salt to the vanadyl, with hydrochloric, hydroiodic, or hydrobromic acid and the absorption of the evolved chlorine, iodine or bromine in potassium iodide, and subsequent titration of the liberated, or absorbed iodine with sodium thiosulphate solution. These methods are erratic on account of the reduction not stopping at a specified compound, some of the vanadium going to a lower valency, with the liberation of the corresponding amount of the halogen.

The method which seems to be in general use is the separation of the vanadium from the metals by means of sodium hydrate, reduction of the acidified filtrate with sulphur dioxide, removal of the excess sulphur dioxide by boiling and a current of carbon dioxide, and subsequent oxidation of the reduced vanadium with potassium permanganate. This method, while theoretically correct, gives results which as a rule are high, if the vanadium is all separated from the precipitate of iron. It requires two or more separations to get all the vanadium out of the bulky iron precipitate, and the filtrate must be boiled and refiltered on a very close filter as some iron always goes through on the first filtration. The method is slow, taking not less than four hours for a single determination, and most all of the operator's time. It takes several filtrations, which we try to avoid as much as possible, as they take time and introduce errors. The method in detail is as follows:

A factor weight of the finely powdered alloy is dissolved in sulphuric and nitric acids, evaporated to white fumes of sulphuric anhydride, diluted, and the iron precipitated by pouring the nearly neutralized solution into an excess of boiling sodium hydrate, which contains a small amount of sodium peroxide. Sodium should always be used when separating vanadium, as sodium vanadate is more soluble than the potassium salt. The solution is boiled and filtered. The precipitate is dissolved in hot sulphuric acid and reprecipitated. The combined filtrates boiled, again filtered on a close filter, acidified with sulphuric acid, adding about 10 per cent excess, reduced with purified sulphur dioxide, the excess of which is removed by boiling and the passage of a current of carbon dioxide, which has been passed through a solution of sodium carbonate. The gas is passed until the steam will not discolor a solution made slightly pink with a drop of permanganate. The vanadium is then oxidized with N/10 potassium permanganate, titrating to a pink in the hot solution.

Following are typical results on three different vanadium compounds: a pure solution of vanadium, a synthetic solution made of approximately the same composition as a solution of an alloy, and a standard ferro-vanadium. The pure solution was made from pure vanadium pentoxide, which was made by recrystallizing ammonium vanadate several times. The salt was dried and the major portion of the ammonia driven off in a drying oven. The oxide was then melted in a platinum dish in a current of oxygen. The resulting oxide was ground to a fine powder and showed the following composition by analysis:

Vanadium	55.06%
Vanadium Pentoxide	99.86

Vanadium Tetroxide	0.70
Ammonia	0.00
Sodium Oxide	0.948
Silica	0.063
Alumina	0.041
Alkaline earths	0.06
Hydrogen sulphide metals	0.00
Iron group	0.009

A weighed amount of this oxide was fused with pure sodium carbonate, the water solution acidified with sulphuric acid, and diluted to a definite volume, 1 c.c. of which contained .003136 grams vanadium. The synthetic solution was made from this solution, and C. P. salts. The standard ferro-vanadium has been carefully analyzed and the results averaged. Fifty c.c. of the pure solution was taken in each case, this being equal to a percentage of 30.74 vanadium on a factor weight of alloy.

Pure solution	Synthetic solution	Standard
Vanadium present.	Vanadium present.	Ferro-Vanadium
30.74%	30.74%	Vanadium present
Vanadium found.	Vanadium found.	Vanadium found
30.68%	30.82%	35.56%
30.74	30.92	35.86
30.78	30.88	35.74
30.70	30.82	35.88
30.78	30.90	35.72
Average results	Average results	Average results
30.74	30.88	35.70

The pure solution gives very good results, but the synthetic solution and the standard ferro-vanadium show an increase of 0.14 per cent and 0.24 per cent respectively. This increase of vanadium has always been found, and if the work is very carefully done, keeping all conditions constant, the results will keep down between 0.10 per cent and 0.30 per cent high, but at the present price of ferro-vanadium this is too great an error. It has been noticed that often when no more sulphur dioxide can be detected in any way, the results will run several per cent high. This has not been accounted for, except on the theory that some product is formed in the solution which is oxidized by permanganate, nor has the cause of the ordinary high results been determined. Of course this method cannot be used when metals reducible by sulphur dioxide are present, as they will again be oxidized by the permanganate with the vanadium.

All the other methods, which were at all practical, were tried in the same manner, and discarded for very much the same reasons as the above method.

The following method has been found to be accurate under all conditions, and on all varieties of work. The solutions needed are:

N/10 potassium bichromate, made by dissolving 4.008 grams of the pure salt per liter of water, and carefully standardizing to exactly N/10 at 20 deg. C. against pure iron wire, and also against N/10 potassium permanganate which has been standardized at 20 deg. C. against sodium oxalate, procured from the Bureau of Standards. Ferrous ammonium sulphate was used as an intermediate solution. The bichromate is constant and needs very little care after it is standardized, except that it should be restandardized once a week to guard against accidents.

N to ferrous ammonium sulphate solution, made by dissolving 40 grams of the salt per liter of 10 per cent sulphuric acid. As the strength of this solution changes about .004 c.c. per c.c. a day, it is standardized against the bichromate under working conditions every day.

A blank of 0.32 grams pure iron is run the same as a test to the point of titration, when 35 c.c. of the ferrous solution is added. This is titrated back with the bichromate to a definite end point. For instance 35 c.c. of the ferrous solution took 34.20 c.c. of the bichromate to oxidize it. This shows the ferrous solution to be 0.80 c.c. weak in 35 c.c., or 0.023 c.c. per c.c.

This factor is subtracted for each c.c. used on a determina-

tion, or if the ferrous solution is strong, the factor is added.

Potassium-ferri-cyanide solution made by taking a piece of the salt about the size of a pea, and washing the outside several times with pure distilled water to remove any ferro-cyanide which may have been formed on the outside, and dissolving the remainder in 50 c.c. distilled water. The ferri-cyanide must contain no ferro-cyanide, as vanadyl salts gives a green precipitate with ferro-cyanide and this can easily be mistaken for the green of the ferrous iron. A paper appeared some time ago (Cain, J., *Ind. & Eng. Chem.*, Vol. III, 476), claiming that vanadyl salts in acid solution, reduced ferri-cyanide to ferro-cyanide, thus giving erratic results. This has been tried from every angle on pure salts and under working conditions, and no immediate action could be noticed. Fifty c.c. of a 10 per cent solution of vanadyl sulphate was added to 50 c.c. of a 1 per cent solution of ferri-cyanide, and allowed to stand for twenty-four hours. No precipitate had formed at the end of that time, while when one drop of a 1 per cent solution of ferri-cyanide was added to the mixture, a light green precipitate developed in a few minutes. The presence and amount of ferro-cyanide in ferri-cyanide can very easily be determined by means of this reaction. Care must be taken not to make the ferri-cyanide solution too strong as the yellow color will mask the last traces of green in the spot.

Potassium permanganate solution, made approximately N/10 but not standardized.

Sulphuric acid—one part acid to two parts water. This strength acid is used, as it can be poured into hot fuming sulphuric acid without spattering, saving considerable time cooling and again heating the solution.

The apparatus used is an automatic burette, for the ferrous ammonium sulphate solution, made from our specifications by Bausch and Lomb. The burette is made to deliver 50 c.c. and is graduated in 1/20 c.c. from 30 c.c. to 50 c.c. A 25 c.c. burette graduated to 1/20 c.c. is used for the bichromate and an ordinary 50 c.c. burette for the permanganate. A white porcelain spot plate can be used. The depressions should be shallow as it is easier to control the spot in a shallow one. They are cleaned by dropping into a sodium hydrate solution after using, and then washing. The best spot is shown on a piece of waxed dead white, high gloss cardboard. The card is cut to size and immersed in a solution of white paraffine, repeating until the coat of wax is heavy enough to keep the solution from coming in contact with the paper.

The drop remains spherical and reflects the smallest traces of green. The waxed cards are easier to handle than the plates, and after being used are thrown away. The ferri-cyanide is kept in a flat top dropping bottle, which regulates the size of the drop.

The method is as follows:

Dissolve a factor weight (0.510 grams) of the finely powdered alloy in 25 c.c. of the dilute sulphuric acid and 10 c.c. concentrated nitric acid, and evaporate to white fumes of sulphuric anhydride. If the alloy is refractory to these acids, it is transferred to a platinum beaker with the addition of 20 c.c. hydrofluoric acid, and again evaporated to white fumes. Seventy-five c.c. of the dilute sulphuric acid is added to the hot fuming mass and the solution heated until all the salts have gone into solution, when it is boiled for two minutes. The solution is transferred to a 5-in. white porcelain casserole. 75 c.c. more of the sulphuric acid added, and hot water until the bulk of the solution is from 400 to 500 c.c. The solution is then heated to above 65 deg. C. Potassium permanganate is added from the burette to a deep red. This red is just discharged with the ferrous solution, then permanganate is added drop by drop until the last drop just gives a pinkish tinge to the solution, which is discharged by one drop of the ferrous solution. The ferrous sulphate burette is now filled to O, and an excess is run into the solution, this excess is shown by the ferri-cyanide spot. An excess of at least 2 c.c. should always be added. This excess is titrated back with the bichromate solution being careful to come to the same end point as in the blank. The c.c. of the ferrous sulphate solution

used, plus or minus the correction, less the c.c. of bichromate solution used to oxidize the excess of the ferrous solution, gives the percentage of the vanadium in the alloy directly.

The only points to be carefully watched are the neutralization with the ferrous and permanganate solutions, the end point with the ferri-cyanide spot, the variation of the strength of the ferrous ammonium sulphate solution, and the temperature of the solution while being titrated.

The neutralization point can be caught within 1/20 c.c. after a little practice. The operator can alternate drop for drop of the two solutions as long as he is uncertain as to the exact point, without impairing the accuracy of the determination in the least. An alternate method can be used for this neutralization as follows: The alloy is dissolved in 25 c.c. of the sulphuric and 10 c.c. nitric acid, evaporated to white fumes, the salts dissolved in hot water, and diluted to 500 c.c. Potassium permanganate is added in excess, and the solution boiled until all permanganate is decomposed, or a crystal of manganous sulphate can be added to hasten the decomposition. The precipitated manganic oxide is filtered on a tight asbestos plug, using suction. The filtrate is boiled, and if any precipitate comes down, it is again filtered. The solution is now evaporated to a convenient bulk (about 300 c.c.), 150 c.c. of the dilute sulphuric acid added, the solution heated to above 65 deg. C., and an excess of ferrous solution added directly. This method gives good results, provided all the manganic oxide is removed by filtration, but takes at least an hour longer than the regular method.

The end point with the ferri-cyanide is very easily seen, and if the operator bears in mind the color of the end point on the blank, and on the standardization, and always titrates to that point he will get good results. No two operators see the end point the same, so they must use their own standardization, and blank. If the end point is over titrated more ferrous solution can be added and again titrated back with bichromate.

The temperature of the solution must be kept above 65 deg. C. or the results will be low. Tests run at different temperatures show that the amount of ferrous solution oxidized by the vanadium decreases with the temperature, running as low as .60 c.c. in the cold, but it makes no apparent difference how far the temperature is raised above 65 deg. C.

The above method is fast, taking about 30 minutes for a single determination, does not vary when other metals are present, chromium being the only interfering element, and its interference can be eliminated by getting the neutralization point between the ferrous and permanganate solutions in the cold, and then heating to above 65 deg. C. for the titration. The presence of chromium can be shown with an acetic acid solution of diphenylcarbazide, this reagent being very sensitive to chromium.

Each and every determination of vanadium made on ferro-vanadium is run in duplicate, and not once in a hundred times is there a difference of more than 0.10 per cent.

The following results obtained on regular routine work, show how the method runs.

		Standard
Pure solution	Synthetic solution	Ferro-Vanadium
Vanadium present.	Vanadium present.	Vanadium present.
30.74%	30.74%	35.56%
Vanadium found.	Vanadium found.	Vanadium found.
30.76%	30.72%	35.58%
30.70	30.78	35.50
30.74	30.80	35.60
30.75	30.72	35.54
30.78	30.71	35.58
30.79	30.73	35.60
30.72	30.76	35.58
30.72	30.80	35.54
30.76	30.80	35.52
30.75	30.72	35.55

The following results were taken from one of the chemist's record books, and represents one day's routine work on the

determination of vanadium in ferro-vanadium, showing how close it is possible to duplicate.

Determination.	Duplicate.
36.70% V	36.70% V
30.44	30.44
39.47	39.48
38.35	38.36
28.22	28.24
34.13	34.13
38.24	38.26
30.53	30.51
29.78	29.83
28.41	28.43
37.22	37.24
26.69	26.80
36.79	36.81
29.33	29.36
37.06	37.01

American Vanadium Company,
Pittsburgh, Pa.

Spelter Production in 1912

The Geological Survey has just issued its advance statement of spelter production and consumption in 1912. This statement gives the final figures of output by the zinc smelters for the year, distributed both by states producing the ore and by states in which the ore was smelted, thus giving a measure of the zinc mining industry as well as the zinc smelting industry. The zinc ore made into pigments is not included in this statement, hence the full extent of the zinc mining industry is not covered. The imports and exports of spelter, zinc dross and zinc ore are also given, as well as a list of smelters and their capacity, revised to the close of 1912, together with additions being built during the first months of 1913. A long chart shows graphically the fluctuations for the last seven years in the price of spelter at St. Louis and London, and in the price of 60 per cent zinc concentrates at Joplin.

The production in the United States of spelter made from ore, both domestic and foreign, was 338,806 short tons, an increase of 52,280 tons, or 18.2 per cent, over that of the previous year, and by far the largest output in the history of the industry. The final figures show that the Survey's estimate of production given out January 2, 1913, was too low by only 176 tons, or 0.05 per cent. The spelter made in the United States from foreign ore in 1912 amounted to 14,809 tons, almost exactly the same as in 1911, and considerably less than in 1910 and several preceding years. The production of spelter from secondary sources such as skimmings and drosses also made large gains, being estimated at 50,000 tons, of which 21,000 tons was redistilled, partly at regular zinc smelters using ore and partly at plants devoted exclusively to the redistillation of secondary materials.

The consumption in the United States of spelter made from ore was 340,372 tons, an increase of 60,313 tons, or 21.5 per cent, over that of the previous year. The increase in consumption was made possible by the large imports of spelter. For the last four months of the year the average St. Louis price of spelter was more than 1½ cents above the London price. It was during this period that the larger part of the 11,115 tons of foreign spelter was imported.

The list of smelters shows a total capacity of 107,948 retorts at the close of 1912, with additions of 12,216 retorts under construction. With the exception of 576 retorts, the additions are all in Illinois. It should be borne in mind that all this capacity will not be effective for smelting ore, for several of the plants listed are devoted partly or exclusively to the recovery of spelter from secondary materials.

An aluminium solder has been patented by Richard Seitter, of Union Hill, N. J., consisting of 38 per cent pure tin, 32 per cent phosphor-tin and 30 per cent pure zinc. It is claimed that this solder will be effective in joining pieces of aluminium, or pieces of brass or bronze with aluminium.

Electrolytic Production of Iron Sheets and Tubes, Etc.

By Wilhelm Palmær and J. A. Brinell

Upon request of the directors of Jernkontoret (the Swedish Iron Masters' Association) the authors have made a study of the S. Cowper-Coles electrolytic method for producing iron sheets and tubes, etc., at the inventor's laboratories in London. The results are given in the following paper. For the purpose of investigating the quality of the products, the composition of the electrolyte and other matters, tests have been made at the testing laboratories of the Royal Technological Institute at Stockholm, the results of which tests also will be given in the following.

The inventor has described his method in a lecture: "The Production of Finished Iron Sheets and Tubes in One Operation," delivered at a meeting of the Iron and Steel Institute at Middlesbrough in 1908, published in the *Journal of the Iron and Steel Institute* and in *Electrochemical and Metallurgical Industry*, November, 1908, vol. VI, p. 447, and he has also received a number of patents.*

Before entering into the description of Cowper-Coles' process we wish to call attention to the fact that electrolytic refining of iron has been subject of much research work, partly for practical purposes. In the above-mentioned lecture Cowper-Cowles reviewed some of this work, a more complete synopsis of which will be found in a recently published book by F. Foerster¹ and in a bibliography written by A. Müller.² As yet it seems that electrolytic refining has found practical use only for electroplating purposes, such as steel plating of copper engravings for printing of bank notes at the Imperial mint at St. Petersburg, Russia, where such a plant has been used since 1860. It seems that the Langbein-Pfanhauser Werke Aktiengesellschaft, Leipzig-Sellerhausen, Germany, has lately been working along the same lines as Cowper-Coles, on a method suggested by F. Fisher.³

Principles of the Method

The Cowper-Coles method is an electrolytic refining process. From pig iron he produces iron sheets or tubes of nearly pure iron. The principles of the method are therefore identical with those of other commercial electrolytic refining processes for the purpose of producing pure copper in ingots, sheets and tubes, as well as silver, gold, zinc, nickel and lead in pure quality. The pig iron is used as anode in a solution of an iron salt. On the application of direct current pure iron is deposited on the cathode, while at the same time a corresponding amount of iron is dissolved from the pig iron anode so that the composition of the solution remains unchanged.

The difficulties to be overcome in the development of such a method are mainly following:

(1) A salt solution must be found suitable as electrolyte, with a sufficiently high conductivity and one which is not fouled too rapidly.

(2) Beside the nature of the electrolyte, the voltage and current density must be correctly chosen in order to get a pure dense deposit in form of an even sheet or ingot without spongy formations and without the actual energy consumption exceeding the theoretical requirement to any appreciable extent.

(3) Electrolytic refining is easier with comparatively pure metals such as copper, silver or gold, than with more impure metals such as iron and zinc, since in the latter case hydrogen will be evolved at the cathode as easily as metal deposited.

The electrolyte used by Cowper-Coles is a concentrated solution of ferrous chloride, containing also iron salts of certain organic acids and an amount of iron oxide is added to give a consistency of a gruel. Additional organic compounds are also used to produce more satisfactory results. The organic acid,

*English patents, No. 250,060, 21,481, 06, 21,382, 06, 188,000, 18, 188,074, 06, 29,300, 06, 10,367, 07, 12,747, 07, 22,311, 37, 22,313, 37, 21,010, 08, 36,260, 08, 10,655, 09.

¹"Beiträge zur Kenntnis des elektrochemischen Verfahrens des Eisens," pp. 59-81, published by W. Knapp, Halle, Germany, 1910.

²See the journal, "Metallurgie" (German), Vol. 6, p. 152, 1910.
³See, for instance, Svensk Kemisk Tidskrift, 1911, p. 135 (Sweden).

of which the iron salt is contained in the solution, is according to a statement in the inventor's former lecture, $\text{CH}_3(\text{OH})\text{C}_6\text{H}_4\text{SO}_3\text{OH}$, consisting of several isomeric cresol-sulphonic acids. The additional iron oxide is intermingled for the purpose of reducing the acidity and polishing the iron sheet which is deposited on a rapidly rotating cathode. The anode used by Cowper-Coles consists of pig iron.

It might be expected that nearly pure iron would be deposited—that is soft sheet iron without carbon or impurities. There is, however, in fact some carbon in the iron obtained by the

of the latter, the thinnest sheets, can most easily be produced.

As to the application of the process for the far more important problem of producing iron of highest purity and quality from ore, it seems that this has been only slightly studied by the inventor, and we shall briefly return to this subject later.

Construction of the Electrolytic Cell

A sketch of the electrolytic cell used at the experimental plant at 20 Danvers street, Chelsea, London, is shown in Fig. 1. The illustration is made partly from sketches and dimensions taken by the authors and partly from a drawing furnished by the inventor. It is drawn at a scale of 1:8, the dimensions being in millimeters.

B is a tank of riveted 5-mm sheet iron containing the electrolyte. Its height is 310 mm and it consists of two concentric cylinders with a bottom. In the horizontal cross section the outer and inner circles show the walls of the tank. On the inside of the tank walls is applied a heavy coating of insulating paint which is of great importance as they otherwise would serve as an anode and rapidly be corroded.

At the inside of the outer wall there are three iron anodes, *A*. They are each 300 mm long and 12 mm thick and are made of curved pig-iron plates, each covering about 1/12 of the circumference. The anodes are separated from the tank through an insulating material, *I*. As will be noted the anodes cover together only about one-quarter of the surface of the tank.

The cathode proper, *K* consists of a 0.5-mm. copper-plated iron cylinder, which is placed outside another cylinder, *C*, of heavier sheet iron (perhaps 5 mm.). The upper and lower edges of the cathode, *K*, are turned out slightly and the space filled with asphalt, probably for the purpose of preventing iron to deposit there. The height of the cathode, *K*, is about 185 mm and its diameter is about 275 mm. The space between cathode and anode is about 40 mm.

The cylinder, *C*, serves as a conductor to *K* and also to support the cathode in such a way as to allow a rapid rotation.

For the latter purpose, the cylinder, *C*, with its sheet-iron cover, *H*, is connected with a vertical steel axle, *E*, supported by bearings *S* and *D*. Through the belt, *R*, the cylinder, *C*, and the cathode, *K*, are rapidly rotated. The shaft, *E*, has just above the pulley a ring attachment, *U*, which during the rotation of the shaft is partly submerged in a bowl filled with mercury, which bowl is not connected to the shaft. The mercury is connected to the negative busbar and transmits current to the rotating cathode *K*.

Finally there is under the tank a circular gas burner for the heating of the electrolyte.

Experiments Performed at the Authors' Visit

The electrolyte used on this occasion was, according to a statement by the inventor, prepared as follows:

Concentrated hydrochloric acid (sp. gr. 1.17), mixed with the cresol-sulphonic acids mentioned before, is heated with a large amount of scrap iron, which is dissolved, while developing hydrogen, until the solution is saturated with iron. The solution, the specific gravity of which has increased as a result of the solution of the iron, is diluted until the specific gravity is 1.4, after which, even on cooling to ordinary temperature no iron salt (ferrous chloride) crystallizes out. The solution contains then about 40 per cent ferrous chloride (FeCl_2) to which are added before using about 300 grams of ferric oxide (Fe_2O_3) per liter, so that it gets about the consistency of gruel.

The addition of this ferric oxide is primarily made for the purpose of reducing the acidity of the solution as far as possible, with proper consideration given to the hydrolysis of the iron salt solution. Moreover, the ferric oxide affects the condition of the finished product. As mentioned before, since the cathode is rapidly rotated and through friction with the iron oxide, the sheet or tube deposited on the cathode receives a high finish. In the Elmore process for the electrolytic production of copper tubes the finish is obtained by means of polishing slate of agate applied under pressure and moved up and down in the longitudinal direction of the tube, while in the Cowper-Coles process the same result is obtained by rapid rotation with accompanying friction with the liquid.

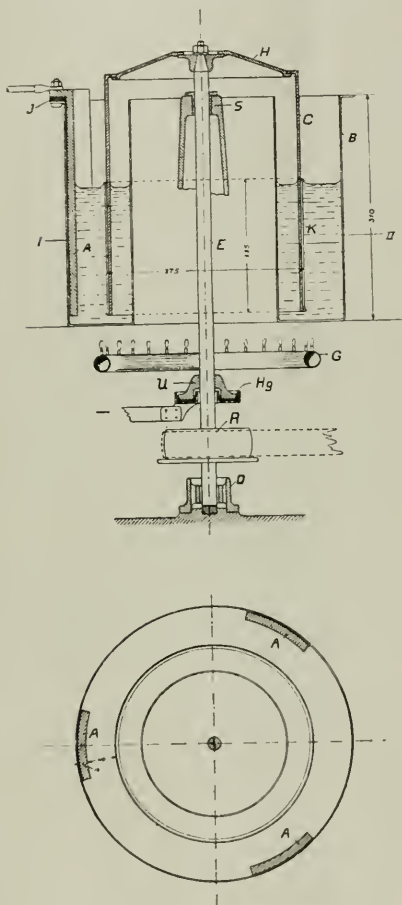


FIG. 1—CONSTRUCTION OF ELECTROLYTIC CELL

Cowper-Coles process and there are also traces of phosphorus and sulphur.

That carbon can be electrolytically deposited to a small extent, has been stated,* but without endeavoring to control the percentage of carbon, one should be satisfied that the product will always be a soft sheet metal of very high quality. It is also likely that the carbon, which is found by analysis, is due to slime mechanically transported from the anode to the cathode.

According to Foerster (p. 77 of his book) consideration should also be given to the colloidal organic substances which may be deposited on the cathode, as a result of the oxidation of carbon of the pig-iron anode.

On the other hand the electrolytic process has the advantage over the rolling process that the most expensive product

* *Zeitschrift f. Electrochemie* 1896, 2, 541

Experts have testified that an iron sheet produced in a solution containing iron oxide is solid while a sheet produced under other circumstances is perforated by numerous small holes.

The information given by the inventor in regard to the composition of the electrolyte is confirmed by the analysis we have had made at the Material Testing Laboratories of the Stockholm Royal Technical Institute of samples of the solution taken immediately before and after the electrolysis. These tests are referred to as I and II respectively. At first the solution was tested with the ferric oxide contained in it.

	I.	II.
	Before	After
	Electrolysis	
Specific gravity	1.60	1.67
Amount of slime in grams per liter:		
Dried at 105 deg. C.....	245	293
Heated to glowing.....	240	288

The percentage of slime is found to be somewhat different with the amount of stirring at the different times when the samples were taken. Due to the same fact the specific gravity varied somewhat.

Subsequently the filtered solution was analyzed. In both samples it had a light green color, showed a weak acid reaction, and had an evident smell of carbonic acid (phenol).

The result of the analysis is as follows:

	I.—BEFORE		II.—AFTER	
	Grams per Cu.Cm.	Per Cent Weight	Grams Per Cu.Cm.	Per Cent Weight
Specific gravity of solution at 150° C.....	1.46		1.46	
Bivalent iron	0.235	16.1	0.235	16.1
Trivalent iron	Trace		Trace	
Sulphur as SO ₃	0.0189	1.29	0.0223	1.53
Chlorine	0.290	19.9	0.285	19.5
Ferrous chloride, Fe Cl ₂ , calculated from quantity of chlorine	0.518	35.5	0.510	35.0
Cresol-sulphonic acid, corresponding to the quantity of SO ₃	0.0445	3.0	0.0524	3.6

The small differences in the analysis should be considered as experimental errors and we conclude that no difference in the composition before and after electrolysis has been established. This was not unexpected, since the time of electrolysis was only two hours.

The amounts of FeCl₂ and cresol-sulphonic acid correspond approximately to the figures given by the inventor.

For the purpose of comparing the energy consumption and voltage of the Cowper-Coles process with that used in other electrolytic refining processes, it was of interest to investigate the conductivity of the electrolyte used. The result is stated below, the conductivity, *C*, being the reciprocal of the resistance, in ohms, of a column of 1 cm length and 1 sq. cm cross section. For comparison we have given some results of measurements of the ordinary electrolyte for copper refining (1 normal CuSO₄ + 1 normal H₂SO₄).

Cowper-Coles Electrolyte		Copper Electrolyte	
Temperature	C.	Temperature	C.
100 deg. C.	0.213	101 deg. C.	0.270
13 deg. C.	0.0506	42 deg. C.	0.231
		13.8 deg. C.	0.170
		0 deg. C.	0.144

According to these and other measurements the conductivity of the Cowper-Coles solution at 100 deg. C decreases 0.88 per cent per degree.

For the experimental temperature of 92 deg. C. of Cowper-Coles solution $C = 0.198$ and if corrected for the ferric oxide which occupies in volume 58 cu. cm per liter and which is a non-conductor of electricity $C = 0.187$.

In copper refining ordinary or somewhat raised temperatures (45 deg. C.) are used and it seems that at these temperatures the conductivity of the solution is about 0.2 that is nearly the same value as above. The voltage for copper refining is stated

to be 0.25 volt to 0.4 volt, while in the Cowper-Coles process it is about 1.4 volt. This great difference in voltage is not due to the lower conductivity of the Cowper-Coles solution but as calculation shows is due to the greater current density and the greater distance between the electrodes, while the anode surface is smaller than the cathode surface. It may be mentioned that in electrolytic silver refining a still higher voltage is used, namely, 2 volts to 2.5 volts.

The volume of the electrolyte used at our visit may be estimated at about 15 liters.

In calculating the current or current density, the surface of the cathode was taken as 16 sq. dm. as given in the description of the electrolyzer. The voltage varied in electrolysis I between 1.25 and 1.45 volt according to the temperature, the voltage decreasing as the temperature increased. The temperature was read a number of times and varied in electrolysis I between 88 deg. C. and 94 deg. C.

The speed of rotation at the cathode varied in electrolysis I between 108 and 160 revolutions per minute.

In calculating the efficiency it has been assumed that one ampere will deposit 0.0174 gram of iron per minute from the solution of ferrous salts, there being no appreciable amount of ferric salts.

Electrolysis I

In the first electrolysis a cylinder of thin copper-plated sheet iron was used as cathode. To loosen the electrolytically deposited sheet the cathode cylinder had to be heated until glowing.

The electrolysis was carried out for 2 hours, 2.5 min.
Average current 107.9 amp.
Current density at cathode..... 6.74 amp. per sq. dm.
Average voltage 1.34 volt
Average temperature 92 deg. C.

The average speed of the rotating cathode was 130 r.p.m., corresponding to a peripheral speed of 120 meters per minute.

Efficiency.—After heating the cathode in a reducing atmosphere, whereby deposited hydrogen was also removed, the iron sheet could be removed from the cathode and weighed. It was soft and had a thickness of about 0.15 mm. Its weight was 102 grams, not including some pieces which had fallen off nor the iron deposited on the cylinder C. This amount (102 grams) corresponds to an efficiency of 83.5 per cent, assuming that 230 grams could be deposited by the number of ampere-hours used.

In order to account for the total ampere-hours.

the iron sheet produced corresponds to..... 83.5 per cent
Hydrogen evolved corresponds to..... 11.8 per cent
Iron lost and deposited on cylinder C, escaped

hydrogen, errors in test, corresponds to . . . 4.7 per cent

100.0 per cent

Electrolysis II

In the second electrolysis a cylinder of thin zinc-plated sheet iron was used as cathode. From this cathode the deposited iron sheet could be removed without previously being heated. The electrolysis was carried out for 2 hours 30 minutes.

Average current 110.6 amp
Current density at cathode, 6.91 amp per sq. dm.
Average voltage 1.43 volts
Average temperature 94.5 deg. C.

The average speed of the cathode was 145 r.p.m., corresponding to a peripheral speed of 125 meters per minute.

Efficiency.—The sheet, in this case deposited on a zinc-plated cathode, could be removed without heating, but was very brittle and broke into a number of pieces. This was due to the high percentage of hydrogen. The pieces had a total weight of 274 grams, some pieces, however, being missing. In this case it is doubtful whether the sheet was perfectly dry. Furthermore, it was already somewhat oxidized at the time of weighing. After dipping a piece in boiling water it was a little less brittle. After heating in air it was no longer brittle but rather hard.

The sheet contained before heating a considerable quantity of hydrogen, namely, not less than 0.45 per cent, according to the analysis of the Material Testing Laboratories, of Stockholm, given below. In 27.4-gram iron sheet this amounts to 1.23 grams hydrogen and its deposition required 11.8 per cent of the coulombs consumed.

This percentage the inventor considers unusually high, but we have decided to figure with the values obtained in Electrolysis II, as well as Electrolysis I, which was performed in the same manner. In heating the sheet the hydrogen (1.23 grams) is removed and the weight of the tempered sheet is thus $27.4 - 1.2 = 26.2$ grams, corresponding to an efficiency of 94.5 per cent, since the theoretical weight of the deposit is 288.7 grams.

The ampere-hour balance sheet is therefore:

The iron sheet deposited corresponds to..... 94.5 per cent
Deposited hydrogen corresponds to..... 11.8 per cent

106.3 per cent

Oxidation and moisture on sheet and errors... -6.3 per cent

100.0 per cent

The current density in the Cowper-Coles method is approximately 6.8 amp. per sq. dm. at the cathode (it was in the two electrolyses 6.74 and 6.91 amp per sq. dm. respectively). The highest current density mentioned by Foerster is only 3 to 4 amp per sq. dm. and it seems, therefore, that Cowper-Coles has accomplished a considerable improvement in this respect. A high current density is desirable, even at the expense of a higher voltage, as it has a direct bearing on the time consumed, and thus increases the output of the plant.

Loss in Weight of Anodes.—In test II the total loss in weight at the three pig-iron anodes was ascertained in order to compare how this agreed with the ampere-hour consumption. After the anode slime had been removed the total loss was found to be 291.4 grams, while, under the assumption that the iron is dissolved as well as precipitated as bivalent, it is estimated that the iron taken off the anodes is equal to the amount deposited on the cathode, or 288.7 grams.

However, the anode slime removed weighed 20 grams and the weight of the iron deposited on the cathode was 272.8 grams, and as no increase in weight of the total percentage of iron in the solution had been noticed, it seems reasonable to expect that the difference between loss in weight of the anodes and the amount of iron deposited on the cathode would approximately equal the weight of the anode slime. This difference is $291.4 - 272.8 = 18.6$, which, indeed, nearly corresponds with the actual weight of the anode slime (20 grams).

That the anodes lose more than the weight of the iron deposited on the cathode is customary in electrolytic refining of impure metals and is in the present case due to the following causes:

It is to be expected that the carbon and silicon compounds and possibly sulphur and phosphorus compounds in the pig iron are affected not at all or only slightly by the current and remain as anode slime, which is gradually separated from the anode.

Further, small particles of iron are mechanically separated if the anodes are unevenly consumed.

If the weight of the anode slime is 20 grams, this would be 7 per cent of the iron consumed (219.4 grams). However, it is possible that this percentage of loss is too low.

On one hand it is possible that some slime has been separated from the anode during the operation, in which case the slime removed from the anode and weighed after the test was finished would be too low.

On the other hand, it is also possible, as stated above, that the value of deposited iron (272.8 grams) is somewhat high and consequently the difference between this value and the loss in weight of anodes is also too low.

In copper refining the anode slime is stated to be 4 per cent as a maximum, but the crude copper is considerably purer than the pig iron.

Since the composition of the anode slime, compared with that of the pig iron, is of considerable interest, the slime was analyzed by the Material Testing Laboratories of Stockholm, as well as the pig iron used in these experiments. The result was as follows:

	Pig Iron Used For Anodes	Anode Slime
Iron.....	93.11% (being the difference)	13.5%
Graphite.....	2.51, 2.81% total carbon	23.3, 23.8% total carbon
Combined carbon.....	0.30, 2.81% total carbon	0.5, 23.8% total carbon
Silicon (Si).....	2.30	33.9, corresponding to 15.9% Si
SiO ₂		
Manganese.....	0.43	1.5
Sulphur.....	0.10	1.78
Phosphorus.....	1.23	2.14
Copper.....	0.02	0.16
Chlorine, water, oxygen, etc.....		23.22 (difference)
	100%	100%

From this it is evident that in the anode slime, which still contains some mechanically separated iron, carbon, silicon and sulphur is greatly concentrated, while the phosphorus has been largely oxidized (phosphoric acid) and gone into the solution.

It is interesting to note that the copper is also concentrated in the anode slime and is not dissolved, quite in accord with observations in other electrolytic refining processes. Manganese is also somewhat concentrated, probably due to oxidation to peroxide.

Composition and Properties of the Deposited Sheet Iron

At the Material Testing Laboratories of the Royal Technical Institute, in Stockholm, the iron sheets produced in the tests have been analyzed as to chemical composition and tested as to rusting and mechanical and electrical properties. The analyzed material consisted of:

1. Iron sheet, 0.52 mm. thick, which had been heated and thereby rendered soft. It had been submitted by the inventor and is designated as El. 1 in the following.

2. Sheet produced in electrolysis 2, about 0.15 mm. thick, not previously tempered and consequently hard and very brittle, designated as El. 2 in the following.

A. CHEMICAL COMPOSITION

The results of the analysis are given below, together with the previously given analysis of the pig iron used for the anodes, which shows plainly the result of the refining.

	El. 1. 0.52 Mm. Tempered	El. 2. 0.15 Mm. Not Tempered	Pig Iron
Carbon.....	0.08%	0.06%	2.81%
Silicon.....	0.009		2.30
Manganese.....	0.05		0.43
Sulphur.....	0.02		0.10
Phosphorus.....	0.042		1.23
Chlorine.....	0.18 (=0.32% FeCl ₃)	0.49	
Hydrogen.....	0.021	0.45	
Hydrogen, after heating to 850° C.....	0.002		
Copper.....			0.02

A glance at these analyses shows at once that in regard to carbon, silicon, sulphur and phosphorus very good results are obtained, these elements being present in large quantities in the pig iron, while only small amounts are to be found in the electrolytically deposited sheet. The fact that even if steel anodes are used with a small percentage of carbon, and otherwise as pure as possible, their presence cannot totally be avoided is shown by older observations. The percentages in electrolytic iron are (according to Foerster) as follows:

C.	Si.	S.	P.
0.007-0.080%	0.003-0.01%	0.001%	0.004-0.02%

In regard to manganese it was shown previously by the analysis of the anode slime that it is somewhat concentrated. This indicates, as mentioned, that part of the manganese is oxidized at the anode and does not dissolve. Another part of

the manganese is probably dissolved as manganese chloride. That, nevertheless, such a small quantity of manganese is present in the cathodic iron sheet is probably due to the fact that the manganese being less "noble" than iron is not so readily deposited. It is, therefore, probable that it is concentrated in the solution in the same manner as the iron contained in crude copper is concentrated in copper refining. An older statement* that the manganese is not dissolved with the iron is not confirmed by the Cowper-Coles method.

As already mentioned, the percentage of hydrogen is remarkably high in the non-heated sheet. For information it may be mentioned that a percentage of 0.45 of hydrogen corresponds to not less than 394 units of hydrogen per unit of iron in volume, while the 0.021 and 0.002 per cent of hydrogen in the sheets, which had been heated, correspond to 18.4 and 1.75 units of hydrogen per unit of iron in volume respectively. That electrolytic iron contains a high percentage of hydrogen has long been known, earlier records, however, showing about 0.1 per cent (see Foerster, pp. 64-73). The high percentage in the iron sheet, produced by the Cowper-Coles method, is probably due to the high current density employed in this process.

Since by heating the sheet El. 1 at 850 deg. C. the percentage of hydrogen decreased from 0.021 to 0.002 per cent, the heating carried out by the inventor must have been done at a lower temperature or for a shorter time. As mentioned several times, the hydrogen makes the iron hard and very brittle and it must therefore be removed by heating.

The high percentage of chlorine is very remarkable and if it cannot be avoided it is very disadvantageous. It may be that the sheet was not well washed before heating or the sheet may have been a little porous and the pores retain some electrolyte, viz., iron chloride.

Therefore the percentage of iron chloride, FeCl_2 , was calculated from the percentage of chlorine in the above table. The fact that the very thin sheet, produced at our visit, which was not washed particularly well, gave a considerably higher percentage of iron chloride (0.88 per cent) than the other sheet (0.32 per cent) indicates that a thorough washing is very desirable. This might also decrease the percentage of sulphur and phosphorus.

As presence of iron chloride seems to promote the rusting of the sheet and also impair the mechanical properties of the iron, it is evidently of great importance to try to prevent the presence of iron chlorides. That there is a possibility of success in this respect has already been pointed out, and it is to be noted that in copper refining chloride has not been found in the cathode copper.

B. RUSTING

Comparative tests were made with sheet El. 1 and also a soft basic Martin sheet from Avesta (Sweden), 0.55 mm. thick, and with 0.1 per cent C., in the following designated as Av. Two polished pieces of each sheet were suspended in damp air; after three days the electrolytic sheet was considerably more defective than the Avesta sheet.

The explanation might be found in the presence of iron chloride in the electrolytic iron. This salt is deliquescent in damp air and can thus form a solution of the salt on the surface of the sheet, promoting electric local currents, which are believed to cause the rusting.

C. MECHANICAL PROPERTIES

We shall only mention some qualitative tests made with sheet El. 2 in order to show how the mechanical properties are improved by heating to a higher temperature and consequent removal of hydrogen.

The Testing Laboratory's bending test, after heating to various temperatures, was as follows:

A number of pieces of iron sheet were placed in an electric oven which was heated to increasing temperatures. At each 50 deg. rise in temperature one piece was taken out and this was continued until a piece could be bent double and straightened out again without cracking.

After heating to 450 deg. C. the sample broke at first attempt to bend. After heating to 500 deg. C. the sample could be bent to 90 deg., but cracks appeared when bent to 180 deg. After heating to 550 deg. C. the sample bent double without cracking. In straightening out cracks appeared. After heating to 600 deg. C. the sample bent double and straightened out without cracking.

According to these tests the sheet should be heated to 600 deg., at least if time is limited, for improving the sheet.

The effect of heating to higher temperatures (850 deg. C.) is shown in the following quantitative tests:

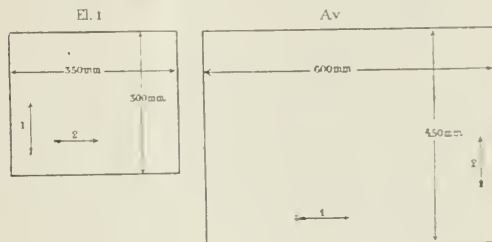


FIG. 2.—TEST PIECES

At these tests was used:

Electrolytic sheet, El. 1, (Fig. 2) the thickness of which was 0.52 mm and other dimensions 350 x 300 mm.

Martin sheet from Avesta, Av. (Fig. 2), the thickness of which was 0.55 mm and other dimensions 600 x 450.

Strips were taken from these sheets in two directions perpendicular to one another as shown by arrows 1 and 2 in Fig. 2.

Sheet El. 1 was not made in our presence and we cannot tell with certainty which direction on the deposited sheet cylinder corresponds to 1 and 2. However, it had streaks running parallel with arrow 1.

The samples were tested in their original condition and after heating to 850 deg. C. and cooling they were submitted to stress and bending tests.

The results are given in Table I:

TABLE I—STRESS TEST

Sample	Direction in Sheet	Physical Condition	DIMENSIONS			Tensile Strength, Kg. per Sq. Mm.	PLASTICITY IN PER CENT	
			Width, Mm.	Thickness, Mm.	Cross-Section, Sq. Mm.		Unit Length	100 Mm.
El. 1	1	Not heated	24.8	0.52	12.84	55.9	10.8	7.1
	2	Not heated	24.2	0.52	12.95	55.8	0.5	11.2
	1	Heated	24.5	0.52	12.58	58.2	15.3	11.8
	2	Heated	24.5	0.52	12.74	52.8	2.3	2.0
Av.	1	Not heated	24.7	0.55	13.59	41.0	11.3	8.6
	2	Not heated	24.1	0.55	13.50	33.1	16.0	15.2
	1	Heated	24.6	0.55	13.53	40.3	14.3	12.2
	2	Heated	24.8	0.55	13.64	36.3	13.3	11.3

Bending Test.—In this test samples 20 mm wide were bent over a bar with 2 mm radius, until the break occurred. The results are given in Table II:

TABLE II—BENDING TESTS

Sample	Direction in Sheet	Physical Condition	Number of Bends to 90° and Reversals, Average of 3 Tests
El. 1	1	Not heated	3
	2	Not heated	3
	1	Heated	11
	2	Heated	10
Av.	1	Not heated	12
	2	Not heated	28
	1	Heated	18
	2	Heated	20

The results of further tests are given in Table III

*Burgess and Taylor, "Electrochemical Industry," 1906, 4, 208

TABLE III

Sample	Direction in Sheet	DIMENSIONS			ELASTICITY PER		Distance of Breaking Point from Nearest End Mark, Mm.
		Width, Mm.	Thickness, Mm.	Cross-Section, Sq. Mm.	Elastic Limit K _{g.} per Sq. Mm.	40 Mm. %	100 Mm. %
El. I	Parallel with streaks, . . .	24.7	0.52	12.84	66.8	12.5	10.3
	Parallel with streaks, . . .	19.9	0.52	10.35	65.7	6.5	4.1
	Right angle to streaks, . . .	24.2	0.52	12.58	65.3	10.0	7.1
	Right angle to streaks, . . .	19.8	0.52	10.30	53.7	1.3	1.2

D. ELECTRIC PROPERTIES

The electrolytic sheet (El. I) tempered at 850 deg. C. had at 16 deg. C. a resistance of 0.116 ohm per 1 m. length and 1 sq. mm cross section or 11.6 microhm per cm². With Benedick's formula* the specific resistance of the sheet in non-heated condition (percentage of hydrogen = 0.002 per cent) is calculated to be 11.4 microhms per cm² at "ordinary temperature," which corresponds very nearly with the direct measurements.

In the calculations the chlorine is not considered as it probably exists as iron chloride. It was our intention to investigate also the magnetic properties but shortage of material prevented us from doing so.

It is to be expected that, due to its pure state, the sheet will show very small hysteresis loss. For purposes where this is desirable high electric resistance is usually also wanted, which the electrolytic iron does not possess.

But the electrolytic sheet iron can be made so exceedingly thin (probably about 0.05 mm) that the hysteresis and eddy currents can be brought down to a reasonable amount.

It seems therefore possible that the electrolytic iron can favorably compete with the nickel and aluminium steel which are produced at a thickness of about 0.35 mm by means of rolling for purposes where low hysteresis loss and high electric resistivity is desirable.

Another desirable property of the electrolytic iron is its high permeability. It is stated that an 1/2-hp motor, after having been equipped with poles of electrolytic iron, made according to F. Fischer's method, delivered 1 1/4 hp. (See Svensk Kemisk Tidskrift referred to above.)

Some Factors Which Influence the Economical Result of This Process

As stated above the percentage of carbon can not be regulated, but it can always be assumed that the product obtained by this process contains only a small percentage of carbon, viz., it is a soft sheet or tube. If sheet iron of great length or width is wanted the cylinder must be of very large dimensions. On the other hand tubes, of which we have seen samples, can be made of greatly varying diameter and length and we wish to call attention to the fact that copper tubes have been made successfully by electrolytic processes in a similar manner.

In regard to thickness of material it seems that the inventor has not as yet produced tubes thicker than 2 mm. He did not, however, anticipate any other difficulties in the production of heavier material than possibly that of the weight.

On the other hand in rolling the thin sheet is the final and most expensive product while in the electrolytic process it is the one most quickly and easily produced. The prospects of the process would, therefore, probably lie in the production of thin sheets and tubes of soft steel.

One defect in the process is the method of removing the sheet and heating. In the description of Electrolysis I it was mentioned that the iron was deposited on a copper-plated cylinder, K, which was taken off the cylinder, C, and heated, after which the deposited sheet iron could be taken off and was

soft. This method will undoubtedly be troublesome in making large sheets, as the cylinder, K, would have to be of very large size. In making tubes the method is simpler.

More serious will be the condition of the cylinder, K, when in constant use, which will probably require renewed copper plating. To remove the sheet without previous heating as was done in Electrolysis II is impossible in commercial practice, as the sheet will break into pieces. The inventor made several suggestions for remedying this defect.

Consumption of Pig Iron

As stated above, the consumption of pig iron is larger than the weight of the deposited iron. The weight of the anode slime is according to our observations about 7 per cent of the weight of the consumed pig iron. This estimate, however, may be too low. We shall, therefore, assume a somewhat larger loss—say 1.1 ton of pig iron is required for producing 1 ton sheet iron—without stating, however, that this figure is on the safe side. This 1.1 ton of pig iron should then represent the amount of iron dissolved and deposited plus the anode slime per ton of product.

In this connection it should be pointed out that, disregarding the anode slime, the anodes cannot be completely utilized: when only a portion is left the voltage increases and it must be replaced, while it also happens that an anode is eaten through and a large portion is entirely separated.

The portions remaining can, of course, be collected and recast, which will slightly add to the cost of production. The same applies to the trimmings of the sheet iron. How long the anodes can be used before replacing can only be decided from experience extending over a long time. As a matter of information it may be mentioned that 90 per cent of the anodes used in copper refining can be utilized before recasting.

Energy Consumption

A. FOR ELECTROLYSIS

In the two electrolyses performed in our presence the mean emf. was 1.34 and 1.43 volt respectively or an average of 1.385 volt per cell. This voltage was measured between anode and cathode and to this should be added losses in conductors. This voltage could probably be decreased to some extent by covering the whole inside of tank, B, with anodes. It may, therefore, be safe to figure on a tension of 1.5 volt per electrolyzer.

The ampere-hour efficiency we assumed to be 85 per cent, figuring on 12 per cent loss in precipitation of hydrogen and 3 per cent mechanical losses in removing and trimming the sheets.

For the horsepower year we assume the year to be 350 × 24 = 8,400 hours as usual in electrochemical work.

Further as 1 amp-hour at 100 per cent efficiency will deposit 1.045 grain iron from a ferrous salt solution and as 1 hp equals 0.736 kw we find that one electric hp-year at the electrolyzer should produce 3.7 tons (metric tons of 2,204 lbs. each) of iron sheets or tubes. Or for the production of 1 ton of iron sheet or tube 0.27 electric hp-year is required.

B. FOR MOTOR SERVICE

Electrical energy is also required for the operation of the motors for rotating the cathode, the operation of the pumps, the mechanical treatment of the product and for lighting. Exact information as to the energy required for these purposes cannot be obtained at present, but it can hardly be of great importance compared with the energy requirements for the electrolysis, probably amounting to 1/5 of the latter.

Cost of Labor.—The inventor states, without furnishing further data, that about 50 hours of manual labor is required per ton of products. In this connection it is to be noted that electrolytic processes are practically automatic and require very little attention.

Heating and Maintenance of the Electrolyte.—We did not receive from the inventor any information from which satisfactory estimates can be made as to such cost. The analyses given herein show no change in the composition of the solu-

* Benedick, *Recherches physiques et physico-chimiques sur l'acier au carbone*, Upsala 1904, p. 113.

tion due to the electrolysis, with the exception that the amount of slime had increased, which may be due to the difficulty in obtaining uniform samples. From the short investigation we made of the process it is too precarious to draw any conclusions in regard to the life of the electrolyte.

The cost of finishing the author has estimated at about \$1.50 per ton.

Depreciation.—According to figures submitted by the inventor the first cost—exclusive of power station and iron supply—amounts to approximately \$100 per ton of iron sheet produced per year, buildings amounting to about \$1,600. Allowing 5 per cent for depreciation of buildings and 10 per cent for other items the annual amortization amounts to $\$0.80 + \$8.40 = \$9.20$. The inventor figures only with an amortization at \$1.60 which seems to be too low. This is mentioned only to show that the cost of amortization is a rather large factor, principally due to the high cost of the electrolyzer.

Interest on Investment in Stored Pig Iron.—Electrolytic refining processes operate rather slowly. In Electrolysis II a 6-mm sheet was deposited in 1 hour with the current density used by Cowper-Coles and to produce a 10 mm sheet would require about 7 days of 24 hours. In connection with such processes consideration must, therefore, be given to the interest on stored raw material.

In electrolytic copper refining the interest on the amount invested in raw material is of very great importance and this has been the primary reason for endeavoring to increase the speed of operation as much as possible, that is, operate with highest possible current density. In the Cowper-Coles process the current density is higher—since in copper refining it hardly exceeds 4 amp. per sq. diameter—and beside this the raw material is of comparatively small value, and, therefore, the interest on the amount invested in raw material in iron refining is of no consequence. The inventor has estimated that with an annual production of 5000 tons, a permanent supply of 1000 tons would be sufficient, which is probably ample, judging from the experiences of copper refining. If the value of 1000 tons of pig iron is \$16,000—the interest at 6 per cent would be \$970—or about 19 cents per ton of production.

The Application of the Method for Reduction of Iron Ore

In his lecture before the Iron and Steel Institute Mr. Cowper-Coles also mentioned the application of his method for producing iron sheet and tubes of high quality from iron ore. The inventor has not, however, as yet developed his method for this purpose. On account of the great importance of this matter we wish to add a few words on the subject.

The method which would first suggest itself for realizing the purpose would be to dissolve the ore by acids and extract the iron electrolytically.

In regard to the solubility of iron ores we wish to point out that siderite and bog ore dissolve relatively easily in acids while magnetite and hematite require heating with concentrated sulphuric acid or concentrated hydrochloric acid.

For dissolving 1 ton hematite about 2.76 tons of pure H_2SO_4 are required. On account of the hydrolysis of the iron salts more acid is required than the equivalent weight, for which allowance has been made and 5 per cent has been added; 2.76 tons pure sulphuric acid corresponds to 4.4 ton chamber acid with 62.0 per cent H_2SO_4 . The price of chamber acid is about \$5.50 per ton, possibly less. The cost of only the acid will thus be about \$24 per ton. For magnetite the cost is 8/10 of this or about \$21.50.

With hydrochloric acid the cost is considerably higher. It is, therefore, evident that this application of the process will be commercially possible only if the acid can be regenerated, which is a possibility. In this case carbon or some other inert substance should be used, as anode.

Furthermore, for the regeneration of the acid a diaphragm would have to be used to separate anode and cathode. Under this circumstance and as long as the cathode consists of a rotating cylinder, the apparatus will be rather complicated.

If hydrochloric acid is used the regeneration could be ac-

complished by passing the chlorine, which is developed at the anode from the iron chloride solution, together with steam over incandescing coke.

In using hydrochloric acid there would be added, however, a considerable expense for evaporating the water absorbed in the circulating process. This might also occur in connection with sulphuric acid.

On account of the necessity of using insoluble anodes instead of soluble pig-iron anodes the voltage would be higher. From observations in other electrochemical processes it may be estimated that not less than 5 volts would be required. Assuming an efficiency of 85 per cent as before hematite ore would yield per hp-year 0.73 ton or 1 ton iron would be deposited by 1.37 hp-year. Magnetite would yield 0.82 ton per hp-year and 1 ton of iron would be deposited at the expense of 1.22 hp-year.

In this case, soft steel, practically free from carbon, would be produced, unless some new startling innovations are made, and the amortization of the expensive plant would be quite heavy.

At the present state of developments we shall not venture to express any opinion as to the future possibilities of the process with respect to its application to the direct reduction from iron ores.

Stockholm, Sweden.

The Outcome of the Carborundum Patent Suit

The patent law suit of many years between the Electric Smelting & Aluminum Company and the Carborundum Company has now reached what would appear to be its final stage, by a decision rendered on March 15 in the United States Circuit Court of Appeals for the Third Circuit.

The case was tried before Circuit Judge Gray and District Judges Bradford and Young. The decision is written by Judge Bradford. A dissenting opinion was handed down by Judge Young.

From the decision written by Judge Bradford we quote the following:

"These appeals have been taken from a final decree of the Circuit Court of the United States for the Western District of Pennsylvania made Feb. 21, 1911, confirming the report of a master on an accounting of profits for patent infringement ordered by that court, in this case Dec. 7, 1900. The original suit for infringement was brought by the Electric Smelting & Aluminum Company, hereinafter called the complainant, against the Carborundum Company, hereinafter called the defendant, for infringement of three United States patents, No. 319,795 for 'Processes for Smelting Ores by the Electric Current,' No. 319,945 for 'Electric Smelting Furnaces' and No. 335,058 for 'Electric Furnaces and Method of Operating the Same.'

"The two first-named patents were issued to Eugene H. Cowles and Alfred H. Cowles June 9, 1885, and the last-named patent to Alfred H. Cowles Jan. 20, 1886, and all of these patents were assigned to the complainant. At the final hearing in the Circuit Court the charge of infringement of Patent No. 335,058 was abandoned, and that court held that neither of the other two patents had been infringed.

"On appeal to this court the decree of the Circuit Court was in May, 1900, affirmed as to Patent No. 319,945, but reversed as to Patent No. 319,795; it being held by this court that claims 1, 2 and 4 of the last-named patent had been infringed by the defendant, and the case was referred by the court below pursuant to the decree of this court to a master for an accounting of profits derived by the defendant from the infringement.

"The defendant thereafter and while the case was in the hands of the master adopted, Jan. 1, 1901, what it claimed to be a different and non-infringing process, and in the language of its counsel, 'did it almost on the instant,' and claimed that it was not bound to account for any profits derived from its manufacture and sale of carborundum after Dec. 31, 1900, as at that time it discontinued using the process held to have been infringed. This claim the complainant denies, contending that

the alleged new process was a clear infringement of Patent No. 319,795.

"The master decided in December, 1906, that the alleged new process was not an infringement of that patent, limited the accounting to Dec. 31, 1900, and made an award of \$22,411.62 in favor of the complainant, all of which was confirmed by the court below.

"Before taking up the question of the sufficiency of the sum awarded to the complainant as profits during the accounting period up to and including Dec. 31, 1900, we shall briefly consider whether the defendant was not thereafter guilty of infringement of the complainant's process for which it should have been compelled to account.

"We think the court below was in error in holding that the process employed by the defendant from and including Jan. 1, 1901, to June 9, 1902, the date of the expiration of Patent No. 319,795, was not an infringement of that patent, and that the complainant was not entitled to profits gained by the defendant from the use of that process during that period. This court in and by its decree in 1900 reversed the decree of the court below and decided that claims 1, 2 and 4 of that patent were valid and had been infringed by the defendant. These claims are as follows:

"1. The method of generating heat for metallurgical operations herein described, which consists in passing an electric current through a body of broken or pulverized resistance material that forms a continuous part of the electric circuit, the ore to be treated by the process being brought into contact with the broken or pulverized resistance material, whereby the heat is generated by the resistance of the broken or pulverized body throughout its mass, and the operation can be performed solely by means of electrical energy.

"2. The method of smelting or reducing ores or metalliferous compounds herein described, which consists in subjecting the ore in the presence of carbon to the action of heat generated by passing an electric current through a body of broken or pulverized resistance material that forms a continuous part of the electric circuit, the ore being in contact with the broken or pulverized resistance material, whereby the ore is reduced by the combined action of the carbon and of the heat generated solely by the resistance of the broken or pulverized body throughout its mass.

"3. The method of smelting or reducing ores or metalliferous compounds herein described, which consists in subjecting the ore in the presence of a reducing agent to the action of heat generated by passing an electric current through a body of broken or pulverized resistance material that forms a continuous part of the electric circuit, the ore being in contact with the broken or pulverized resistance material, whereby the ore is reduced by the combined action of the reducing agent and of the heat generated solely by the resistance of the broken or pulverized body throughout its mass.

"The claims of the patent should be liberally construed for reasons heretofore given by this court in this suit, as follows: 'On careful examination we have failed to find in any patent, publication or other matter alleged as an anticipation or as showing the prior art, a practical process for metallurgical or analogous operations involving the use of a discrete body of conductive but resistant material rendered incandescent by the passage of an electric current and mixed or otherwise in contact with the material to be treated. This is the broad underlying idea of the process patent in suit, and is well covered by its claims. The Messrs. Cowles were the first to invent and use this process and the patent must be sustained. It is a meritorious one and its claims are entitled to considerable liberality of construction.'

"This court has declared in this case in relation to the core used by the defendant 'The use of the central core of resistance material is within the process of the appellant under claims 1, 2 and 4. These claims require merely contact in contradistinction to a mixture between the resistance material and the material to be treated, and it is immaterial whether, under those claims the body of resistance material assumes the

form of a central core or any other shape, so long as it is in contact with the material to be treated, is discrete, granular or pulverized in its composition, is rendered incandescent through the passage of electric current through its mass or over its area, and thereby affords the heat to effect the desired end.'

"And further: 'If claims 1, 2 and 4 are broad enough to include actual contact of any kind between the two materials, the resistance material forming part of the electric circuit may be in contact with the material to be treated, whether in the form of a core or several cores, or of strata, or in any other form, to secure the most effective operation of the process under varying conditions involving the nature of the material to be treated, amount of the charge, etc.'

"And in the description of the patent it is stated: 'The scope of this invention is not limited by the degree of granulation, as that may vary with the conditions of the case, and with a large furnace and a powerful current the size of the carbon particles may pass beyond what is ordinarily understood by the term granular and be in fact pieces of carbon of considerable size.'

"The defendant in its alleged new process employed a block core consisting of a number of blocks of electric light carbon, each 32 in. long and 4 by 4 in. in thickness. They are arranged in 'zig-zag' fashion and the ends of each two blocks forming a V rest upon a carbon block from 12 to 14 in. long and 4 by 4 in. in thickness, called a connector block. A large quantity of fine granular or pulverized carbon, or graphite, is disposed between the carbon blocks and around their ends on the connector blocks, and also a large quantity of granular or crushed coke is disposed at each end of the furnace to convey the electric current to and from the electrodes to the black core, that core being in electrical connection with the electrodes at each end of the furnace.

"Notwithstanding the furnace in the arrangement of the resistance material we are satisfied on the evidence that the alleged new process practised by the defendant since Dec. 31, 1900, is the process, the use of which by the defendant this court held to infringe claims 1, 2 and 4 of Patent No. 319,795; the mere mechanical difference in the form and arrangement of the core not serving to affect or change the nature of the process or to shield the defendant from responsibility. The defendant must, therefore, be required to account to the complainant for all profits gained by the former from its manufacture of carborundum under the alleged new process, and its sale, from and including Jan. 1, 1901, to June 9, 1902."

The opinion then takes up in detail the manner in which the sum awarded to the complainant had been arrived at by the master. The total net profits during the accounting period had been \$104,525.74.

"The master deducted from the \$104,525.74, representing total profits, 49.55 per cent thereof, or \$50,792.51, for 'profit properly belonging to the invention of the defendant of the product, carborundum itself,' and 'the profit belonging to the invention of defendants of the bonding material, that under the evidence made the great bulk of the sales of carborundum possible by the defendant company.' After deducting from the total profits the above sum of \$50,792.51, the master apportioned to the complainant, but subject to further deduction, the remaining 50.45 per cent, amounting to \$52,733.23.

"This apportionment was arrived at by ascertaining that the cost of the entire output of the defendant's business during the period of infringement was \$469,665, and that the cost of making carborundum, including its cleaning, crushing, grading, and preparation for sale, was \$236,944.53, or 50.45 per cent of the total cost of the defendant's output, and by assuming that 'each dollar of expenditure by the defendant earned an equal amount of profit.'

"He then proceeded to deduct from the above sum of \$52,733.23 15 per cent thereof as representing the difference between the 'whole cost of carborundum production' and the 'furnace cost,' being \$7,009.99, leaving a balance, subject to further reduction, of \$44,823.24 as 'the part of the profits to which the use of complainant's process in the production of

carborundum contributed, after consideration of the foregoing elements as entering into that profit."

"The master next proceeded to deduct one-half of the above sum of \$44,823.24 as the amount of the supposed contribution to the defendant's profits through its ownership of an Acheson patent and its re-issue for carborundum granted subsequent to the issuance of Patent No. 319,795 in suit, and awarded to the complainant the remaining half, namely, \$22,411.62 as the total amount it was entitled to receive under the accounting ordered by the court below pursuant to the decree of this court, together with the costs of accounting. . . .

"The complainant contends that in addition to such sum as may be decreed to it for profits made by the defendant from its infringement of Patent No. 319,795 during the period from Dec. 31, 1900, to June 9, 1902, the complainant is entitled to recover from the defendant the above sum of \$104,525.74 without deduction. This contention is based upon two grounds: first, that whatever profit there is or has been in the defendant's business has been derived solely through its infringement of the complainant's process, and, secondly, that, if it be assumed that the first ground is untenable, there has been such negligence on the part of the defendant in failing to keep accounts showing the amount of profit derived by it from the infringement that the complainant is entitled to recover the total profits of the defendant's business.

"The first of these grounds we think is well taken. It clearly appears from the evidence that the business of the defendant is based upon and practically confined to the manufacture and sale of carborundum in one form or another. As stated by the court below in its opinion (189 Fed. 710) as to a proper form of decree to be made under the decision of this court on final hearing (102 Fed. 618): 'The respondent has a large plant built and adapted for the manufacture of carborundum. The machinery and apparatus were likewise built for that special work. It is respondent's sole business, and in that article and its application they have built up a large trade and extensive use.'

"The making of wheels, cloth, paper, etc., covered in whole or in part with carborundum is wholly subsidiary or incidental to its advantageous sale. Admittedly it would be impossible for the defendant without its wrongful use of complainant's process to carry on its carborundum operations. Not only could it earn no profits, but its business would cease to exist; for no other process for the manufacture of carborundum than that of the complainant has been disclosed.

"Much has been said on the subject of the Acheson process and product patents No. 492,707, of Feb. 28, 1893, and re-issue No. 11,473, of Feb. 26, 1895, for 'Improvements in the Production of Artificial Crystalline Carbonaceous Materials.' In the description in both of these patents Acheson states: 'My invention consists in the various processes and products of said processes substantially as hereinafter more particularly set forth. . . . I do not limit myself to any of the particulars set forth, as I intend to cover and claim broadly the new crystalline carbonaceous materials whether produced by the particular processes or methods set forth, or by any substantially equivalent methods, and for all purposes to which it may be applied.'

"And his claims were as broad as his statement of invention. It is contended in substance by the defendant that the patent in suit, while covering the process of manufacturing carborundum, does not cover the article itself; that Acheson was the original inventor and discoverer of the article; that the defendant, as his assignee, has a monopoly in that article how ever or wherever or by whomsoever produced in this country; and, therefore, that the carborundum sold by the defendant in the conduct of its business, although manufactured in defiance of the patent in suit, must be treated as a contribution by the defendant towards profits realized and be considered on the accounting in question.

"A valid process patent, though for a process only, confers on that patentee a right to exclude all others from using such process for the attainment of its object. Others may secure

the desired result, if they can, by the employment of a different process, if open to them, but they cannot without leave or license of the patent owner use the patented process with impunity. And on an accounting for profits, and not for damages, in a case of infringement, where profits to the infringer are impossible save through his infringement, he must be treated as a trustee *ex maleficio* and can withhold none of his gains from the patentee. The fact that in such a case the patent is not for the product, but only for the process, is wholly immaterial; for the infringer will not be permitted by a court of equity to take advantage of his own wrong, but will be held accountable as a trustee of the profits.

"It is well settled that, in the case of mechanical inventions, where the infringed patent covers a mere improvement upon mechanism before known and open to the defendant to use, the complainant can recover only the excess of such profits as have been realized through the use of the improvement over what the defendant might have made by the use of such mechanism without such improvement. But it is equally well settled that where the entire commercial value of the mechanism arises from the patented improvement the owner of the patent will be entitled to recover from the infringer the total profits derived from the manufacture, use or sale of such mechanism. . . .

"Nor is the application of the rule affected in a case where profits, and not damages, are to be accounted for by the fact that the owner of the patent has not used the mechanism containing such patented improvement. *Crosby Valve Company vs. Safety Valve Company*, supra. The principle of the foregoing cases, involving the consideration of mechanical inventions, is not without much weight in the determination of the question of profits in the case before us. We think it applies *a fortiori* here, as it was only through the infringement of the complainant's process that profits were made, or indeed, were possible. For, as above stated, no other process than that of the complainant is disclosed under which carborundum can be manufactured.

"Further, the holder of a valid process patent has a right not only to manufacture under the patented process, but to use and sell the product, unless that product be the subject of a valid patent monopoly in favor of another person. The defendant contends that under and by virtue of the Acheson patent it has a monopoly in the product carborundum. If it had such a monopoly it would seem to follow that, save in so far as equitable considerations might produce a different result, it would have the exclusive right to use and sell carborundum whether wrongfully manufactured by itself or rightfully manufactured by the complainant; and in the latter case under the process patent upheld by this court as valid and meritorious while the complainant would have the exclusive right to manufacture carborundum under its process, it would be powerless to use or sell a pound of such manufactured product. It is not perceptible that under such circumstances the process patent would have any value.

"A clear case should be made to justify the court in treating the defendant's product patent as operative or enforceable against the complainant and thus practically nullify the process patent so far as it covers the manufacture of carborundum. Acheson did not discover, invent or have any idea of carborundum until long after the patent in suit was granted.

"On final hearing not only his patents, but much evidence touching his connection with the product, were before this court when it was said: 'We have no doubt on the evidence that carborundum can be and has been produced under the process covered by any of the claims of the process patent in suit, although not so efficiently or profitable under the method of claim 3. So long ago as 1884 the Messrs. Cowles produced it under their process with and without the use of a prepared or pre-formed central core of resistance material. It is true they did not manufacture it for commercial purposes and, failing to appreciate its importance, though aware of its abrasive qualities, applied their process to other branches of metallurgy.'

"And, further. 'The Messrs. Cowles were the first to invent and use this process and the patent must be sustained.'

"It appears from the evidence that, although carborundum was not manufactured on a commercial scale before the Acheson product patent was obtained, specimens of carborundum made by the Messrs. Cowles and the Cowles Electric Smelting & Aluminum Company, the assignor of the complainant, were placed on exhibition in different parts of the country and much notoriety was given through lectures and printed publications to the incandescent feature of the patent in suit. The Messrs. Cowles, it is true, failed to appreciate the importance of the product, as above stated, and did not know it by the name of carborundum—for it had yet not been so christened—and were not familiar with its composition, but they were aware of its highly abrasive qualities.

"There is no evidence that the Messrs. Cowles or their successors in the title to the process patent in suit ever intended to abandon that process so far as the manufacture of carborundum is concerned. The fact that they exhibited from time to time and in different places specimens of what is now known as carborundum strongly indicates an intention on their part not to exclude that product from the operation of the process patent. This court has decided that the process patent in suit is valid, meritorious, and entitled to considerable liberality of construction.

"This case presents some unusual features. Under the process Patent No. 319,795 the Messrs. Cowles and the Cowles Electric Smelting & Aluminum Company had a right, fully sustained by this court to manufacture carborundum, and they also had a right—certainly until the Acheson patent was obtained—though not necessarily an exclusive right to use and sell any carborundum made by them. In fact, they did make carborundum under the patented process, though not on a commercial scale. As before stated, there is no evidence that they or the complainant ever abandoned the right to apply the patented process to the manufacture of that article. The Acheson patent and its re-issue, which were obtained a number of years after the patent in suit, claimed not only carborundum but also a process for the production of that article. This court has decided that the defendant in using the process claimed by Acheson used the process of the patent in suit and was guilty of infringement. It does not appear that there was any other process under which carborundum could be produced.

"If the Acheson patent so far as it claims the product in contradistinction to the process should be held to confer upon the defendant the exclusive right to use and sell carborundum, then, as before indicated, the complainant's process patent was held absolutely at the mercy of the defendant; for the bare right to manufacture without a right to use or sell what is manufactured is valueless.

"While it has repeatedly been stated that a patent may be obtained for a process in contradistinction to its product, or for a product in contradistinction to the process under which it is created, we can recall no case in which it has been held that after a valid process patent has been granted and applied to the production of an article, though not on a commercial scale, and there has been no abandonment by the patent owner of the right to apply the process to the production of the article on such scale, and such article can be produced only under that process, another person can obtain a patent conferring upon him an exclusive right, as against the holder of the process patent, to the use and sale of the article so produced and thereby wholly destroy the value to him of the process patent so far as it may be applied to the production of such article.

"If such a product patent can validly be obtained it would be necessary in order to avoid gross injustice to the owner of the process patent that all profits rendered possible only through the infringement of the process patent should be awarded to the latter, and to that end the infringer should be treated as a trustee *ex maleficio* of the profits.

Patents are not granted to serve as instrumentalities for the perpetration of fraud or the doing of injustice. They are

property, but, like other property, must be used legitimately. We perceive no reason why the aphorism *sic utere tuo ut alienum non laedas* is not applicable to the use of a patent as well as of other property. To permit one who holds a subsequent product patent to infringe a prior process patent, without which process the product could not be created, and then shield himself in whole or in part from liability for his wrongdoing by setting up his product patent, would not accord with the principles usually obtaining in courts of equity. Under such circumstances the least that equity demands is that the wrongdoer be compelled to disgorge his ill-gotten gains.

"But it is not necessary that we should in this case express an opinion upon the validity or invalidity of the Acheson product patent or its reissue. For whether the defendant did or did not hold a valid patent for carborundum as a product is, as has appeared, immaterial to the decision of the question of profits; for profits having been rendered possible only through the infringement of Patent No. 319,795 the whole amount under the doctrines of the authorities above cited must be awarded to the complainant.

"The decree of the court below must be reversed, with costs to the Electric Smelting & Aluminum Company in this court and the court below, and this case be remanded to the court below with directions that the Electric Smelting & Aluminum Company recover from the Carborundum Company upon a further accounting in the court below the total profits realized by the latter from its manufacture of carborundum and its sale from and including Jan. 1, 1901, to June 9, 1902, together with legal interest thereon from the proper date; and further, that the Electric Smelting & Aluminum Company recover from the Carborundum Company the above-mentioned sum of \$88,743.30, together with legal interest thereon from Feb. 21, 1911; and that such further proceedings be had in the court below as shall dispose of this case in accordance with the views herein expressed. A decree may be prepared and submitted."

Judge Young, in his dissenting opinion "concurs in the foregoing opinion so far as it finds the Carborundum Company guilty of infringement of the process patent in suit during the period from and including Jan. 1, 1901, to June 2, 1902; but dissents from such opinion so far as it holds that the profits realized by the Carborundum Company from its infringement of said patent, together with interest and costs, should not be apportioned between the parties in accordance with the decree of the court below, but awards the total profits, interest thereon, and costs to the Electric Smelting & Aluminum Company."

Barium sulphate precipitated in the determination of barium is likely to be contaminated with occluded alkali salts. Owing to the fine state of subdivision of the precipitate it is difficult to wash these salts into the filtrate. Gooch and Hill, (*Amer. Journ. Sci.*, March, 1913), have found that the precipitate may be made coarsely crystalline and easily washed by dissolving the sulphate in sulphuric acid and then evaporating to dryness by directing the flame of a blast lamp directly onto the solution. In order to avoid loss by spattering, a platinum gauze cone is inverted in the mouth of the crucible. The operation is more rapid than evaporation by a Hempel burner, the time required being about 15 minutes, as compared with from one-half hour to several hours by other methods.

The second annual report of the Director of the U. S. Bureau of Mines, for the year ended June 30, 1912, has just been issued. Among the tangible benefits resulting from the Bureau's work are, improvements in mine safety and in mining conditions, laws and regulations for mine safety, and fuel investigations. The general adoption of permissible explosives, having a short and relatively cool flame, has resulted in reducing accidents. A great deal of money has been saved to the government by specifying grades and quality in fuel purchases. Among the needs of the Bureau are suitable buildings, grounds and equipment for carrying on experimental work at Pittsburgh, extension of mine-rescue and first-aid work, extension of investigations on mine accidents, extension of investigations on mineral waste and metal mining industries.

"Exothermic Steel."

Discussion Before the Pittsburgh Section of the American Electrochemical Society

The third meeting of the Pittsburgh section of the American Electrochemical Society was held at the German Club on the evening of February 19, at the conclusion of an informal dinner. The chairman of the section, Mr. C. G. Schleudergberg, presided.

In the preceding meeting Mr. Walter O. Amsler had spoken on the "exothermic steel" process (this journal, vol. X, p. 559, September, 1912) and the present meeting was devoted to two reports on experiments made with this process by Mr. W. W. Clark, chief chemist of the American Vanadium Company, and Mr. Philo Kemery, metallurgical engineer of the Crescent Works of the Crucible Steel Company of America.

Mr. Clark first presented his report as follows:

Report by Mr. W. W. Clark

At our last meeting Mr. Amsler gave us a little talk on his so-called new process of making steel and its product "exothermic steel." The paper had previously been published in *METALLURGICAL AND CHEMICAL ENGINEERING* (vol. X, p. 559), where it received considerable criticism from the different iron and steel metallurgists throughout the country (vol. X, pp. 712, 713, 774, 775, 776).

To give a short synopsis of Mr. Amsler's talk, he takes a mixture of iron ore, feldspar, lime and bauxite, which he heats to 1800 deg. Fahr., or about 1000 deg. C. in a graphite crucible. At this temperature a violent reaction is supposed to take place, with sufficient evolution of heat to raise the temperature of the entire mass above the melting point of the steel, while all the iron oxide is reduced to the metal.

He first claimed that the feldspar and bauxite did the trick, and in his article had a composition formula for feldspar, which, if it were possible, would reduce some iron. He also had the reaction which took place between the iron oxide and the feldspar, showing how the iron was reduced.

He admitted that he did not know anything about this formula, as he was not a scientific man at all, but had just assumed the formula to suit the case, but said that if it were that way it would reduce some iron, but we know from analysis and from all the known laws of chemistry and metallurgy, that it is impossible for feldspar to take up oxygen. He later admitted that maybe feldspar did not do the work, and that carbon entered into the reaction, but still insisted that he was making steel that way, and would be glad to show one or all of us, how it was done.

He also stated that the steel had been in use for some time at the Bureau of Mines in the shape of cannon for the testing of the explosive power of the different powders. Upon investigation of this steel, the Bureau of Mines did not know anything about exothermic steel, but said that they had bought steel from Mr. Fogler, who had a shop in the west end of Pittsburgh, and that it was very good steel, but they were unable to say how it was made. The cannon came to them ready to use.

Taking Mr. Amsler at his word, I offered him the use of our laboratories to test out his process in the presence of two or three of the eminent steel metallurgists of the Pittsburgh district, and although since that time, I have telephoned and written Mr. Amsler, I have been unable to get him to come.

A number of experiments were made, following Mr. Amsler's directions given at our last meeting with negative results, and also some following new directions sent by Mr. Amsler in a letter to me which reads as follows:

"I am at the present time very busy with contract work which must be gotten out at a certain date, but if I can possibly arrange it I will come out to your laboratory some time next week. In making your experiments in a small crucible, try using 6 or 8 oz. of ore, $\frac{1}{2}$ oz. of bauxite, 1 oz. of feldspar and $\frac{1}{2}$ oz. of lime. In heating this charge be sure to use a reducing flame, or cover in such a manner that the oxygen

from your fire cannot enter the crucible. This is essential as otherwise the oxygen absorbing properties of the flux will be satisfied with the free oxygen, rather than that which is in combination with the iron in the ore. I believe under these conditions you will be able to show practically all of the iron in the ore in a metallic state. (Signed) Walter O. Amsler."

A No. 4 Dixon graphite crucible was used in each case where carbon was present, while an alundum crucible was used for the experiments in the absence of carbon. A platinum-platinum rhodium couple and a Siemens-Halske galvanometer were used to record the temperatures. When carbon was present carbon fire ends outside quartz were used, while in the experiments without carbon, the couple was protected by magnesite. The crucibles were placed inside a larger crucible, the space between packed with pulverized coke, and tight covers placed on both. A small hole was bored through both of the covers for the insertion of the thermo-couple.

Experiment 1.—Mix: 6 oz. pure iron oxide, $\frac{1}{2}$ oz. bauxite, 1 oz. feldspar and $\frac{1}{2}$ oz. lime.

The material was carefully weighed and mixed, and then ground to 80 mesh. The crucible was placed in a larger crucible, packed with coke, charged with the above mix, the covers adjusted, and the whole transferred to the furnace which had previously been heated to 1000 deg. C. The mix was held at this temperature for 3 hours. The melt was pasty and after cooling showed small globules of iron scattered throughout the mass with the majority around the edge, where it had come in contact with the carbon of the crucible. The whole melt was weighed and showed by analysis that about 7.5 per cent of the iron had been reduced. Temperature variation 106 deg. C.

Experiment 2.—Mix: Same as in experiment 1.

Brought to 1000 deg. for 1 hour and then raised to 1250 deg. for 2½ hours. The melt was fluid and a small amount of iron had been reduced. This by analysis showed 13 per cent of the iron reduced. Temperature variation at 1000 deg.—64 deg. C. and at 1250 deg.—94 deg. C.

Experiment 3.—Mix: Same as in experiment 1. Brought to 1450 deg. for 3 hours. Melt was very active and poured clean. The iron had been melted and settled to the bottom. This by analysis showed 28 per cent of the iron reduced. Temperature variation 103 deg. C.

Experiment 4.—Mix: Same as in experiment 1. Melted in an alundum crucible. Held at 1250 deg. for 3 hours. The melt was fluid and after cooling showed fine homogeneous structure. By analysis this showed 0.20 per cent of the total iron reduced. Temperature variation 47 deg. C.

Experiment 5. Mix: Same as in experiment 1. Melted in an alundum crucible. Brought to 1450 deg. C. for 3 hours. The melt was very fluid but quiet. This gave 0.38 per cent of the iron reduced. Temperature variation 41 deg. C.

Experiment 6.—Mix: 6 oz. iron oxide, $\frac{1}{2}$ oz. fluor spar, 1 oz. erythrite and 1 oz. soda ash. Melted in a graphite crucible. Held at 1000 deg. for 1 hour and then brought to 1250 deg., where it was held for 2 hours. Showed 11.4 per cent of the iron reduced. Temperature variation at 1000 deg.—82 deg. C. at 1250 deg.—61 deg. C.

Experiment 7.—Mix: Same as in experiment 6. Brought to 1450 deg., and held for 3 hours. This showed 31.4 per cent of the iron reduced. Temperature variation 45 deg. C.

Experiment 8.—Mix: Same as in experiment 6. Melted in an alundum crucible. Brought to 1250 deg. C. for 3 hours. Showed 0.63 per cent of the iron reduced. Temperature variation 23 deg. C.

Experiment 9. Mix: Same as in experiment 6 with the addition of 1 oz. 80-mesh charcoal in an alundum crucible. Held at 1250 deg. for three hours. Showed 86.3 per cent of the iron reduced, but was not melted. Temperature variations 94 deg. C.

Experiment 10.—Mix: Same as in experiment 1 with 1 oz. charcoal added. Held at 1250 deg. for three hours. The melt was more fluid than in experiment 9 and showed 88.8 per cent of the iron reduced, but the iron was not melted. Temperature variation 85 deg. C.

Experiment 11.—A quantity of the mix of experiment 1 without the iron oxide was melted in a platinum crucible in an atmosphere of hydrogen, cooled and weighed. The weight of the material was 21.6076 grams. It was again melted and held for three hours in an atmosphere of oxygen, cooled and weighed. The weight was 21.5937 grams, showing a loss, instead of an increase in weight, as might have been expected from Mr. Amsler's talk and his letter.

From these experiments the following conclusions are drawn: That the mixture of feldspar, bauxite, and lime, as recommended by Mr. Amsler, does not facilitate the reduction of iron from its oxide; that it does not absorb oxygen in any way, nor does it help carbon or other reducing agents to absorb oxygen in any way, except by means of its fluxing action. That any other easily melting flux has the same effect. That any reduction which takes place is entirely due to the action of carbon, and not due to any properties of the mix as reported. That the reaction is not exothermic as reported; in fact it does not show any appreciable change of temperature, during the melt, and that without the carbon no iron is reduced, while if carbon does the reducing, the reaction is endothermic. The total effect of the mix is only one of a flux, and not one which is easily handled. It seems to me that Mr. Amsler has taken this process on faith and not on actual practice as he claims.

Mr. Amsler's process is not a process at all, and I for one am very sorry that the patent laws of our country are so loose, that they will permit the patenting of such a thing, that on the face of it is impossible, and which on trial proves to be a pipe dream, and the exploiting of such things for the innocent investors, who think that if a thing is patented, it is all right.

In the discussion of Mr. Clark's paper Mr. Amsler objected to the phrase that "the exothermic steel process was his process." Mr. Amsler said that Mr. Fogler and not he was responsible for the process. He could not explain why the reactions did not take place in Mr. Clark's experiments. During the discussion Mr. Amsler also quoted Dr. Munroe, of George Washington University, as vouching for the process.

Mr. Philo Kemery, metallurgical engineer of the Crucible Steel Company of America, then presented his report on his own experiments, as follows:

Report by Mr. Philo Kemery

I have made ten experiments with the "exothermic steel" process, following Mr. Amsler's instructions as given in his talk at our former meeting. My experiments with results briefly stated, follow:

Experiment 1.—36 lb. hammer scale, 1 lb. feldspar, $\frac{1}{2}$ lb. bauxite, 1 lb. lime, were all thoroughly mixed and placed in a regular steel melting crucible. The temperature was brought up to a good yellow heat in a small heating furnace. Results: no reaction.

Experiment 2.—The crucible from Experiment 1 with its contents was then placed in a crucible furnace along with a heat. At the end of one hour it was noticed that the crucible was cutting badly, due of course, to the scale and the fluxes. At the end of two hours the crucible was withdrawn, the bottom portion still held together while the top crumbled in the process of drawing. In this experiment 11 lb. of iron was reduced which was to be expected since years ago hammer scale and iron ore were reduced in the same way without the use of the above fluxes.

Experiment 3.—In this experiment, iron ore was substituted for hammer scale as it is much more readily reduced.

80 oz. iron ore, 4 oz. feldspar, 2 oz. bauxite, 4 oz. lime were well ground, thoroughly mixed and placed in a small plum-bago crucible. At the suggestion of Mr. Amsler it was heated in a round cast-iron stove using pulverized coke as fuel. He directed that the coke be piled around and on the top of the crucible so as to insure a reducing atmosphere which was followed out.

Some two or three tests were tried, the temperature being brought up to a good yellow heat for from two to three hours

In all there was found some reduced iron, but in no case was there any evidence of an exothermic reaction. The volume of the contents of the crucible was reduced to about one-half the original in nearly all cases.

Experiment 4.—Later, it was further suggested by Mr. Amsler that a little carbon, say, about 10 per cent, facilitated the reaction, so a new mixture was made up as follows:

No. 1.—40 oz. iron ore, 20z. feldspar, 1 oz. bauxite, 2 oz. lime, 4 oz. charcoal.

No. 2.—40 oz. iron ore, 2 oz. lime, 4 oz. charcoal.

All were ground and thoroughly mixed. Several parallel runs were made by heating in the stove. Crucible No. 2 showed about the same as No. 1. Both contained reduced iron in the form of a spongy mass, but no evidence of any exothermic reaction.

Experiment 5.—Parallel runs with the above mixtures were then made in a heating furnace in which the temperature was raised almost to a white heat with negative results. Beginning with experiment 4 parallel crucibles were run in which No. 1 contained the active fluxes, feldspar and bauxite, while in No. 2 they were left out for comparison of results.

Experiment 6.—Since the heating furnace did not produce as good a reducing flame as is thought necessary, several parallel experiments were tried in a blacksmith's forge using the same formulas as in experiment 4.

Here again the coke was filed all around and over the top of the pot and in 40 minutes the temperature brought up to the melting point of steel.

Crucible No. 1 showed a spongy mass of more or less reduced iron. A glassy slag was formed. The contents were not sufficiently melted to teem. In crucible No. 2 there was less slag, the other conditions being about the same as in No. 1.

The volume was much reduced, the residue or reduced iron from No. 1 could not be differentiated from that of No. 2. No exothermic reaction.

A crucible of steel scrap was then placed in the forge and heated for 40 minutes as near as possible under the same conditions. When removed the pieces of scrap were all fused together. A $\frac{1}{4}$ -in. iron rod was melted at this temperature in about 1 minute, which shows that the temperature was much above that where the exothermic reaction is reported to begin, namely, 1000 deg. C.

The "matte" from experiment 6 was heated in an open-hearth furnace, the crucible being placed on the bank just inside the door of the furnace and examined by tilting the crucible every five minutes. At the end of 35 minutes the contents was boiling. When teemed on the floor 11 grams of metallic iron was separated from the slag. A crucible of steel scrap placed alongside the above melted in 25 minutes.

Experiment 9.—Another crucible of "matte" was heated in the open-hearth furnace at a slightly lower temperature for 35 minutes. A good slag was formed, but when the crucible was teemed a semi-reduced mass of iron was left in the bottom.

The crucible of steel scrap was partially melted.

Experiment 10.—In this experiment three crucibles were used. No. 1.—40 oz. iron ore, 2 oz. feldspar, 1 oz. bauxite, 2 oz. lime, 4 oz. charcoal.

No. 2.—40 oz. iron ore, 2 oz. lime, 4 oz. charcoal.

No. 3.—All steel scrap.

All three were placed in an open-hearth furnace at same time. No. 1 in 55 minutes showed a fine glassy slag, but when teemed there was left the same spongy mass of semi-reduced iron.

No. 2 in 35 minutes developed a little slag with a semi-reduced residue in the bottom that could not be distinguished from that of No. 1. No. 3 was melted in 25 minutes.

These crucibles were tilted over every five minutes for the purpose of examination.

In all of these experiments the crucibles were closely watched, for it is claimed that when the content of the crucible is raised to about 1000 deg. C. the chemical reactions within develop sufficient heat to raise the temperature to that of melted iron. It

is further claimed that when the exothermic reaction begins the content of the crucible is likely to boil over. In no case did I see anything that resembled or approximated these claims.

In a majority of these experiments the contents of the crucible was much reduced in volume. A crust of semi-reduced iron was formed on the top and all around the side the



ONE OF THE PRODUCTS OF EXPERIMENT 10 OF MR. KEMERY

thickness of which depended upon the time and temperature. The results further show that the oxides of iron were reduced about as well without the use of feldspar and bauxite as with. The products of experiment 10 were exhibited to the society in which No. 1 could not be distinguished from No. 2.

General Discussion

Following Mr. Kemery's report Mr. Amsler stated that he himself did not believe at first Mr. Fogler's contentions that he could reduce iron ore to steel by means of his flux. Mr. Amsler re-quoted from his first paper: "This steel was first brought to the writer's attention about a year ago by Mr. George Fogler, the inventor. At that time the claims and descriptions of the process appeared rather absurd and the statements of results were hard to believe. Close investigation of this steel already in use and of the reactions and the theoretical side of the operation led to the conviction that the process has considerable merit." Mr. Amsler again quoted Dr. Munroe, of George Washington University, as endorsing the process.

Mr. Kemery then asked Mr. Amsler whether in the melting this "matte" in an open-hearth steel furnace any difficulty was experienced in the handling of the slag.

Dr. Schluederburg, chairman of the Pittsburgh Section, asked Mr. Amsler if Mr. Fogler would not come down to Mr. Clark's or Mr. Kemery's laboratory and be present while a laboratory experiment was being made.

Mr. Amsler replied that Mr. Fogler was adverse to laboratory experiments and placed no reliance on them. Mr. Amsler said that the material was produced commercially and quoted a case where the Brahm Steel Company had made some Otis elevator shafts in one case where eleven pots of about 100 lb. had been required for one cast.

Mr. Amsler was asked if this flux had not been used some ten years ago in the reduction of copper from its ore.

Mr. Amsler replied that this flux would also apply to copper.

Mr. Kemery reported that in the case where anything like

steel was produced in his experiments, the steel obtained ran 0.134 in sulphur which alone would condemn the steel produced. This sulphur undoubtedly was obtained from the feldspar.

Mr. Amsler questioned as to why the Brahm Steel Company would not furnish a sample of steel for test purposes replied that the Brahm Steel Company did not have a license to make this material and hence would not make this steel or sell this steel to anyone outside of Mr. Fogler.

We are indebted to Mr. Sydney Cornell, secretary of the Pittsburgh section, for the above report of the discussion.

The Willard Gibbs Medal for Dr. Baekeland

The Chicago Section of the American Chemical Society has elected Dr. Leo H. Baekeland, of Yonkers, N. Y., to be the recipient of the Willard Gibbs Medal founded by William A. Converse. The first award was made in 1911 to Prof. Svante Arrhenius, director of the Nobel Institute, at Stockholm, Sweden. The second medalist was Prof. Theodore W. Richards, of Harvard University.

The formal presentation of the Willard Gibbs Medal will be made to Dr. Baekeland at the May meeting of the Chicago Section of the American Chemical Society.

Dr. Baekeland is at present in Europe, but is expected to return early in April.

The jury of award which selected Dr. Baekeland comprised Prof. Alexander Smith, Dr. W. R. Whitney, Dr. E. C. Franklin, Prof. W. A. Noyes, Dr. J. D. Penlock, Prof. G. B. Frankforter, Prof. John H. Long, Prof. Julius Stieglitz, Mr. William Brady, Mr. E. B. Bragg, Mr. S. T. Mather and Dr. G. Thurnauer.

Iron and Steel Market

March developed no material change in the iron and steel markets. The situation has indeed been rather colorless since the first of the year, a definite pace being established and showing no material variation. In brief the history of the finished steel trade has been as follows: Production has been maintained at capacity, but operations were particularly good in March, on account of weather conditions, making March the record tonnage month in the history of the industry. New buying, represented by the closing of contracts, has been at from 60 to 90 per cent of capacity, perhaps averaging 75 per cent. Specifications received against existing contracts have uniformly exceeded shipments, though by only a negligible margin. Realized prices on shipments have slowly increased, through the closing out of old contracts, and in March were only about \$2 a ton below the openly quoted prices on new business.

The course in the pig-iron market has been altogether different from that in the steel market. While the advance in steel prices practically ceased early in October, leaving the market stationary thereafter, pig-iron prices continued to advance until about the middle of December, and only a fortnight or so later evidences of weakness began to appear, these developing into a definite decline which continued through the first three months of this year, making the total decline from the high point in December to date between 75 cents and \$1 a ton, though by no means equally distributed. In foundry iron the declines have averaged fully \$1 a ton, while Bessemer iron has not declined at all, and basic iron has declined only 50 cents.

Thus there has been a divergence between steel prices and pig-iron prices, and again a divergence in the movement of different grades of pig iron, but the latter divergence fits with the former: in other words the strength of the steel market has prevented steel making iron from declining much in sympathy with foundry iron.

The physical situation explains the market movement. Through the building of many furnaces by steel works in recent years the steel works have become more self-supporting, as to pig iron, than ever before. The merchant furnace capacity nevertheless has been increased, chiefly by the building of

furnaces by Lake Superior ore interests, seeking a more certain outlet for their ore than sales in the season to season market. The demand for steel, moreover, has for many years increased more rapidly than the demand for iron castings. The gradually increasing demand last year for all classes of material brought the steel industry to a point in October at which it was operating practically full, but there remained many idle furnaces. Some of these furnaces have been blown in, so that from October to March there was practically no increase in steel works production of pig iron, but an increase of 15 to 20 per cent in merchant furnace production. It is probable that there is now surplus production of merchant pig iron, but this is not certain. The decline in the market could be explained, in part at least, by the conduct of buyers, who are in the habit of buying farther and farther ahead if the market is advancing, but who the moment any weakness develops determine to buy no more iron until their contracts and stocks are entirely exhausted. Thus buyers have been playing a waiting game, sufficient to cause a decline in so open a market as pig iron even were the statistical position wholly in favor of producers. Similar movements do not occur in steel partly because pig-iron markets are local, whereas nearly all steel products are sold on a Pittsburgh basis and partly because the steel interests normally work more in harmony with each other than do producers of pig iron. Pig iron contracts are usually enforced, while steel contracts usually are not enforced if the market goes against the buyer. Hence the steel maker is very anxious to see quoted prices maintained, while the furnaces are relatively indifferent.

Pig Iron

Demand for pig iron throughout the country was very light in March, the chief activity being in low-grade iron for pipe foundries in the East and in basic iron in the Central West. Prices declined in almost all markets, Eastern foundry iron losing 50 cents, and valley foundry iron as much or more. The Southern market dropped 50 cents, to \$13, Birmingham, and there are reports that even that figure has been shaded. Lake Erie furnaces west of Buffalo showed the greatest weakness, quoting prices to distant points, not normally in their territory, such as to undermine prices made by furnaces in such territory. Thus Cleveland furnaces offered iron in Pittsburgh, although at a 50-cent freight disadvantage compared with the valleys, having \$1.40 freight to Pittsburgh against 90 cents for the valleys. Consumers of foundry iron, instead of being tempted by reduced prices, are more firm in their attitude of holding off. In several districts increased inquiry is reported, but such inquiry is obviously for the purpose merely of securing information as to the extent of the drop, consumers being quoted lower prices and yet not buying. In the Pittsburgh-valley district sales of about 30,000 tons of basic iron were effected in March, prices declining slightly from sale to sale, until \$16, valley, was reached for third quarter delivery, against a price of \$16.50 maintained from the beginning of December to the middle of February. The market is now quotable as follows: Bessemer, \$17.25; basic, \$16; No. 2 foundry and malleable, \$16.25; forge, \$15.75, f.o.b., valley furnaces, 90 cents higher delivered Pittsburgh; No. 2 X foundry, delivered Philadelphia, \$17.75; basic, delivered eastern Pennsylvania, \$17.50; No. 2 foundry, f.o.b. Buffalo furnaces, \$16.50; at Cleveland furnaces, \$16; at Chicago furnaces, \$17.25; at Birmingham, \$13. Freight from Birmingham are \$3.25 to Cincinnati, \$4.35 to Chicago, and \$4.90 to Pittsburgh.

Steel

No improvement has occurred in shipments of billets and sheet bars on contracts. Mills having adequate contracts are sheet well supplied, but mills indifferently covered by contract are unable to secure extra tonnage in the market. The United States Steel Corporation has continued to be short of steel, and during March purchased 6000 tons from Butler for consumption at Ellwood City, also taking 15,000 tons in the East and South. The market for such odd lots as are occasionally sold continues to be quotable at \$29 for billets and \$30 for sheet

bars, f.o.b. maker's mill, Pittsburgh or Youngstown. Prices charged on long-term requirement contracts are considerably less, but are advancing. The Carnegie Steel Company on March 20 named prices for second quarter on quarterly adjustment contracts \$1 a ton higher than were named for first quarter.

Finished Steel

There have been further decreases in premiums on finished steel for early delivery, and there is now relatively little of such business except in bars, for which product premiums have rather been tending to increase. For early delivery there are now rarely premiums on sheets, and premiums on plates in reasonable tonnages do not run above \$1 to \$3 a ton at the outside. The East mills are obtaining large premiums on steel bars, sometimes \$4 a ton or more, but in the Central West there are hardly any mill sellers for prompt shipment, and jobbers are obtaining an unusually large amount of business.

In steel manufactures more irregularities have appeared. Rivets have been shaded so much that the quotable market is down \$2 a ton and the same can be said of bolts and nuts. There are slight irregularities in merchant pipe, and prices for fabricated steel on small jobs are considerably lower than in January.

Regular prices at Pittsburgh, unless otherwise stated, are as follows:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Steel hoops, 1.60 cents, base.

Common iron bars, 1.65 cents, Pittsburgh; 1.60 to 1.65 cents, Philadelphia; 1.57½ to 1.60 cents, Chicago.

Sheets, blue annealed, 10 gage, 1.75 cents; black, 28 gage, 2.35 cents; galvanized, 28 gage, 3.50 cents; painted corrugated, 28 gage, 2.55 cents; galvanized corrugated, 28 gage, 3.55 cents.

Tin plates, \$3.60 for 100-lb. cokes.

Merchant steel pipe, ¾ to 3-in., 80 per cent off list.

Steel boiler tubes, ¾ to 4½ in., 70 per cent off list.

Standard railroad spikes, 1.85 cents.

Button head structural rivets, 2.10 cents; cone-head boiler rivets, 2.20 cents.

The reduction of stannic oxide to tin can be accomplished by molten metallic zinc, according to a patented process of Zdenko Metzl, of Rouen, France. When metallic zinc is melted in a reducing atmosphere and treated with stannic oxide, tin is reduced rapidly and quantitatively, and zinc oxide is volatilized.

The Dorr Cyanide Machinery Co., Denver, Col., has just issued a new catalog descriptive of the three Dorr machines, classifier, continuous thickener and agitator. The booklet contains 36 pages, and is illustrated with installations of the different machines in various parts of the world. Practical data also are given regarding the operation of the machines, and the results obtained by them. The classifier and continuous thickener are better known than the agitator, the last being a new development.

The launder type of Richards pulsator classifier is the subject of a new Bulletin, No. 1050, just issued by the Denver Engineering Works Company, Denver, Colo. This is a comparatively new development in classifiers, and consists of a settling pocket which is bolted to the launder, a sorting column under the pocket, a hatch below the sorting column and the Richards pulsator valve. It is a single compartment classifier, and one is used for each table in the mill.

The behavior of nitroglycerine when heated has been investigated by Snelling and Storm, (*Technical Paper No. 12*, U. S. Bureau of Mines), who find that the substance begins to decompose at 50 to 60 deg. C. At 70 deg. C. the commercial product gives off nitrous acid after 15 to 20 minutes. Boiling occurs at 145 deg. C., due partly to decomposition and partly to volatilization. Explosion takes place at about 218 deg. C.

Synthetic Ammonia

An account of the Report of Professor Haber which led the Badische Company to take up the Process on a Large Scale

The report on "the technical production of ammonia, from its elements," which was submitted by the inventors, Prof. Fritz Haber and Dr. R. Le Rossignol, to the Badische company three and a half years ago, has now been published in the *Zeitschrift für Elektrochemie* of Jan. 15, 1913 (Vol. 19, p. 53).

Since the report was submitted the Badische company has developed this process on a large scale and in his paper before the International Congress of Applied Chemistry last September (this journal, September 12, Vol. , p. 637, 1912) Dr. H. A. Bernthsen stated that the "problem has now been solved fully on a manufacturing scale and that the walls of the first factory for synthetic ammonia are already rising above the ground at Oppau, near Ludwigshafen-on-Rhine."

Since the report of Haber and Le Rossignol contains a great many technical details which were not given in the paper of Bernthsen, we give here as full an abstract as possible.

Professor Haber and Dr. Le Rossignol first point out that from a commercial viewpoint the production of ammonia from its elements is an exceedingly attractive proposition. The world's demand of ammonia is several hundred thousand tons a year and these are supplied almost completely from nitrogen in coal. In form of 25 per cent ammonium sulphate the ammonia is worth 22.25 cents per kilogram, while the cost of 14/17 kilogram of nitrogen is 1.125 cents and the cost of 3/17 kilogram of hydrogen is 4.375 cents (1 kilogram of ammonia containing 14/17 kilograms of nitrogen and 3/17 kilograms of hydrogen).

In order to determine the best conditions of the synthesis, that is, especially the most favorable temperature and pressure, two points are of chief importance, first, the thermodynamic equilibrium, and, second, the reaction velocity. The thermodynamic equilibrium determines how far at the given temperature and pressure the reaction between the nitrogen and hydrogen gases, forming ammonia, will proceed and where it will stop, and it determines how great the percentage of ammonia is in the final gas mixture of nitrogen, hydrogen and am-

monia, in which the constituents are in equilibrium with each other. The reaction velocity determines how much time will be required between the hydrogen and nitrogen in order to reach the final condition of equilibrium. It is important to note that while the thermodynamic equilibrium depends only on temperature and pressure, the reaction velocity which also depends on these two factors may further be influenced to a very large extent by the use of catalytic agents.

To develop a practical process the conditions of temperature and pressure should, of course, be so chosen as to

yield an equilibrium in which the percentage of ammonia in the final gas mixture is as high as possible. On the other hand, they should be so chosen that the reaction velocity between nitrogen and hydrogen should be as high as possible. But these two conditions are contradictory to some extent. This will be obvious if an attempt is made to use ordinary atmospheric pressure and to select the most favorable temperature at which the reaction should be carried out. If a low temperature is chosen the thermodynamic equilibrium is favorable, since a relatively large amount of ammonia can exist in contact and in equilibrium with hydro-

gen and nitrogen gases, but in this case the reaction velocity is exceedingly low since practically the two gases do not react with each other at all and no catalytic agent has so far been found which would be efficient at temperatures of 300 deg. C. or less. On the other hand, if a high temperature is used the reaction takes place quickly but stops very soon since the equilibrium is reached when only a very small amount of ammonia is formed.

It is evident that to solve the problem ordinary atmospheric pressure cannot be used, but the best conditions of both pressure and temperature have to be selected. Practical considerations have led Haber and Le Rossignol to the conclusion that the temperature should be chosen between 500 and 700 deg. C. If a higher temperature than 700 deg. C. would be chosen the conditions of the thermodynamic equilibrium become unfavor-

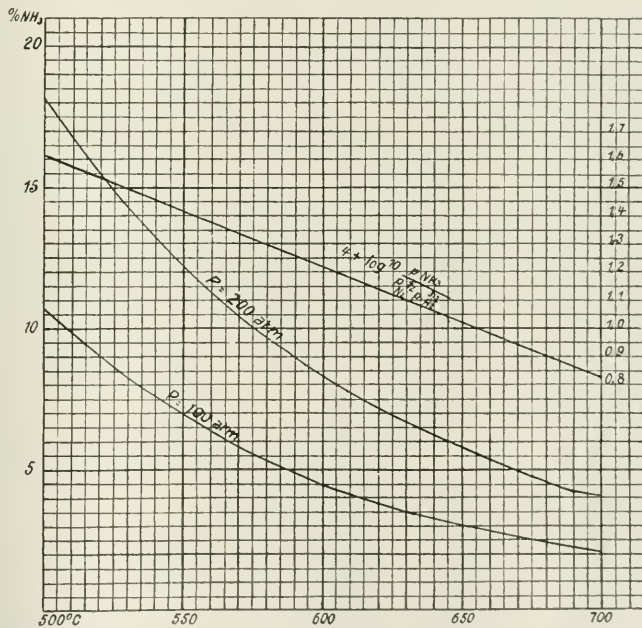


FIG. 1.—THERMODYNAMIC EQUILIBRIUM CONDITIONS OF SYNTHESIS OF AMMONIA

able. If temperatures below 500 deg. C. would be chosen the reaction to be complete would require too much time and too large an amount of catalytic agent. Within the limits of 500 and 700 deg. C. the conditions of the thermodynamic equilibrium are given in Fig. 1.

This diagram gives in the straight line the equilibrium constant and in the two curves marked $P=200$ atmospheres and $P=100$ atmospheres, the percentages of ammonia in the final gas mixture which represents thermodynamic equilibrium. For instance, with a pressure of 200 atmospheres the final gas mixture in which its constituents are in equilibrium contains 18 per cent ammonia at 500 deg. C., 8 per cent ammonia at 600 deg. C. and 4 per cent ammonia at 700 deg. C.

While these percentages are satisfactory, they are not high enough to conduct the process in such a simple way that the gas would be compressed, passed over the catalizer, whereupon the ammonia formed would be absorbed, and the balance of the gas would be wasted. The waste would be too great. The process must, therefore, be conducted in such a way that after the ammonia is recovered from the final gas mixture, the remaining gases, together with a fresh supply of H and N gases, are subjected again to the action of the catalytic agent. This problem has been solved by Haber and Le Rossignol by devising a partial cyclic process for the movement of the gas, the ammonia being formed at a high temperature and the ammonia being removed from the gas mixture in form of liquid ammonia at a low temperature, the remaining gas mixed with fresh gas being then subjected again to the action of the catalytic agent, and so on.

Care must be taken not to use too low a temperature for the removal of the ammonia from the gas mixture, since there

Practical experience with liquid air indicates that "regeneration of heat and cold" becomes particularly efficient and easy when using high-pressure gases. The same has been found to be true by Haber and Le Rossignol in their experiments.

FURNACE CONSTRUCTION

Both quartz and metallic containers were used in their experiments. Above 700 deg. C. and with pressures up to 50 atmospheres Haber and Le Rossignol used quartz tubes, but did not dare to use them at 100 and 200 atmospheres. Later on when they found that it would be advantageous to use temperatures below 700 deg. C., Haber and Le Rossignol developed various metallic furnaces, several types of which were heated from the inside by electric resistance wire. Their construction is described in detail and illustrated in the report.

Later on it was found possible to use so low a temperature that it was sufficient to use iron containers heated from the outside. The continuous production of ammonia became possible by means of a simple sort of test tube of steel, shown in Fig. 2. A steel cylinder with 6 mm. inside diameter and a 4 to 5 cm. length contains the catalytic agent. The inside space slants outside at the top, forming a cone, so that a steel stopper can be pressed into it, also of conical form. The angle of the cone of the stopper (16 deg.) is less than the angle of the cone in the cylinder head (20 deg.). By turning the screw as shown in the illustration the apparatus can be closed absolutely air tight.

The nitrogen and hydrogen gases are introduced through a steel capillary tube which reaches down to nearly the bottom of the reaction chamber. The gaseous products of the reaction leave the chamber through another capillary tube at the top.

About 2 or 3 grams of osmium are used as catalytic agent in this apparatus. To heat it, the whole apparatus is placed in a fused salt-peter bath and the nitrogen-hydrogen gas mixture is passed under high pressure slowly through the apparatus. With a bath temperature of 555 deg. C. and a pressure of 83 atmospheres the percentage of ammonia in the gases passing out of the apparatus was 5 per cent, which is very nearly equal

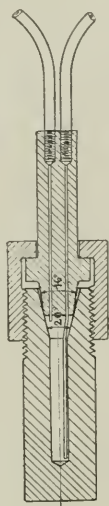


FIG. 2. — SIMPLE FURNACE FOR SYNTHESIS OF AMMONIA

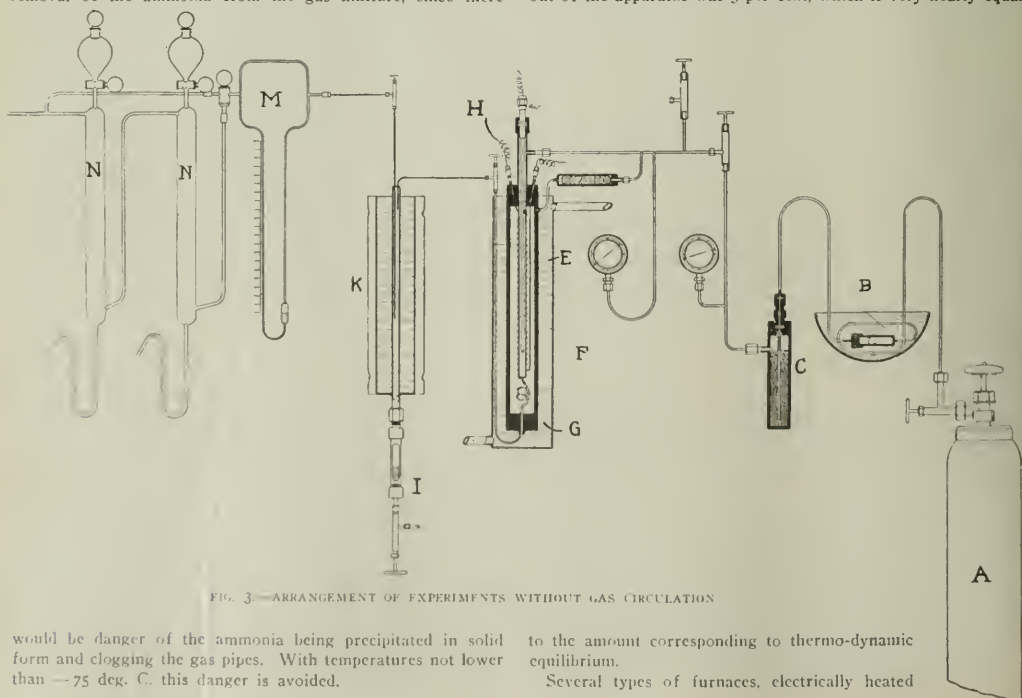


FIG. 3. — ARRANGEMENT OF EXPERIMENTS WITHOUT GAS CIRCULATION

would be danger of the ammonia being precipitated in solid form and clogging the gas pipes. With temperatures not lower than -75 deg. C. this danger is avoided.

to the amount corresponding to thermo-dynamic equilibrium.

Several types of furnaces, electrically heated

from the inside, are described, the construction being given in considerable detail and in various diagrams in the original report. Especially if electric heating is employed it is important not to lose the heat, but to regenerate it. The use of heat regeneration has also the advantage that the gases which leave the catalyzer at a high temperature have a low temperature when they reach the apparatus exit.

The method of heat regeneration employed by Haber and Le Rossignol is very simple, the gas mixture being forced to pass through prescribed passages, the heat regenerator con-

In the second series of experiments the technical practicability of the whole process as a continuous process had to be tested in that the gas mixture coming from the furnace had to be cooled, the ammonia removed, and the remaining gases returned to the furnace, etc., in continuous operation.

The arrangement of the first series of experiments (with gas circulation) is shown in Fig. 3. The mixture of hydrogen and nitrogen gases is passed from the steel container *A* through a copper capillary tube into an iron cylinder *B* filled with palladium asbestos and heated in a salt-peter bath. On account of the combination of the last traces of oxygen in the gas with hydrogen some water is formed which is removed by passing the gas through calcium chloride *C*. The gases then pass to the furnace *E* in which *F* is the thermo-element, *H* the heating wire, and *G* the cooling water.

The mixture of hydrogen and nitrogen gases and of the ammonia formed in the furnace *E* passes out through a thin steel capillary tube into the iron tube *K* which is subjected to cold so as to liquefy the ammonia. To the iron tube *K* is connected at its bottom a glass tube *I* of thick walls in which the liquefied ammonia accumulates and can be measured and drawn off at will.

This refrigerating apparatus *K* for the liquefying of the ammonia was not used in all experiments. In the majority of the experiments the liquefier *K* was disconnected and the gas mixture passed from the furnace *E* directly into the meter *M* which measured the velocity of the gas. The determination of the ammonia in the gas mixture could be carried out in a great many cases by an optical method (apparatus *N N*) by using the Rayleigh gas interferometer of Carl Zeiss, according to the method proposed by Haber for the analysis of furnace and mine gases.

The arrangement of the second series of experiments (with gas circulation) is shown in Fig. 4. The hydrogen and nitrogen gas enters at *A* into the furnace *F* in which the ammonia is formed, and the ammonia, nitrogen and hydrogen gas mixture leaves the furnace through the capillary coil *B*, which is embedded in cooling water. The gases pass two valves, pass through the dryer *C* (where any traces of water are removed) and then enter the "cold regenerator" *D*. This consists of a system of three copper capillary tubes wound as shown in the detail sketch *X*. The gases having been strongly cooled in the regenerator, enter the liquefier *D* in which the ammonia is liquefied. The level of the liquid ammonia can be seen at *E*.

The liquid ammonia may be drawn off through *F*. The hydrogen and nitrogen gas mixture from which most of the ammonia has been removed by liquefaction, is then returned to the cold regenerator *D* and is then forced through the pump *P* back into the furnace *F*, while a valve permits to introduce new hydrogen and nitrogen gases simultaneously into the furnace.

All the constructional details of the pump and the other high-pressure apparatus were worked out by Professor Haber and Dr. Le Rossignol themselves and the apparatus were built by them with the aid of the machinist of their laboratory. Since doubts had been expressed concerning the practicability of working

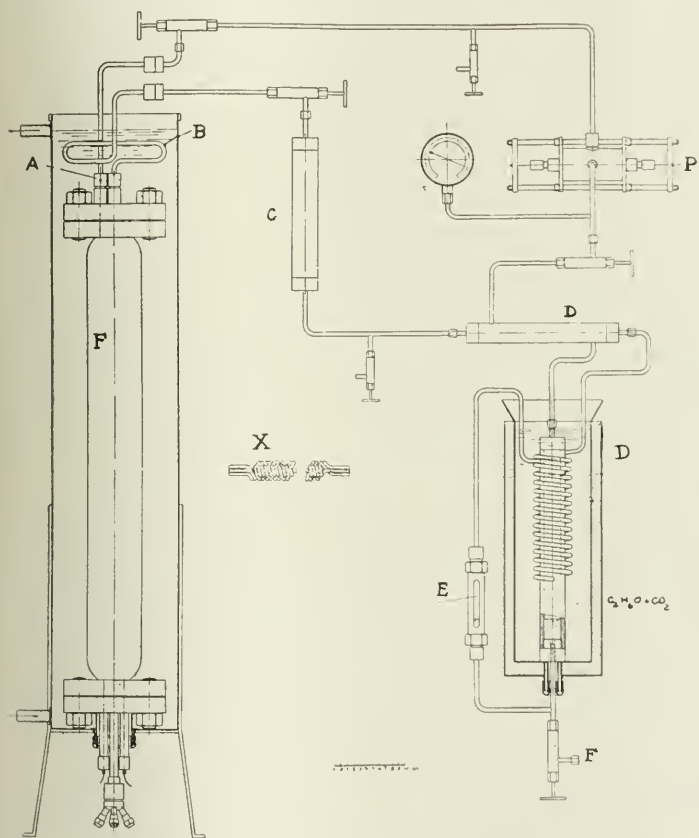


FIG. 4 ARRANGEMENT WITH GAS CIRCULATION

sisting essentially of a great many capillary steel tubes held together by perforated plates at the two ends.

The catalytic agent was placed into the catalytic chamber in a manner which is acknowledged to be rather primitive. Upon the vertical iron tube which contained the thermo-elements, wire-net disks were fixed so as to divide the catalyzer room into a number of chambers one above the other. Some osmium was sprayed into each of these chambers, asbestos being employed to prevent the very fine osmium powder from dropping down through the wire net or from being blown away. The osmium filled only a very small part of the chambers.

Arrangement of Experiments

Two kinds of experiments were made. In the first series of experiments the object was simply to determine the efficiency of different catalytic agents and for this purpose it was sufficient to pass the mixture of hydrogen and nitrogen gases (in the proportion of 3 to 1 by volume) through the furnaces and determine the ammonia that was formed.

with moving gases under high pressure, the authors emphasize that in their work of several years there was never a break of any apparatus or any appreciable trouble.

Catalytic Agents

In investigating the properties of the different elements as to their ability to act as catalytic agents in the formation of ammonia, Professor Haber and Dr. Le Rossignol used the periodic system as the leading idea. It was known that *manganese* and *chromium* are catalyzers. Since in the group containing manganese the elements with highest atomic weights are missing, no advance was possible along this line. On the other hand, in the group containing chromium the two elements with highest atomic weights are tungsten and uranium. In fact the authors obtained a fairly good success with *tungsten* and surprising success with *uranium*.

Uranium may be used in the form in which it is obtained in the electric furnace; it only needs to be roughly crushed. For if it is then slowly heated in the high-pressure gas mixture of one volume gas mixture and three volumes hydrogen, it disintegrates by itself into a fine powder. Commercial metallic uranium, made by Kahlbaum, was used, also commercial metallic uranium, made by de Haen, and uranium made by Moissan's method.

Among the different experiments with uranium, described in the report, is the following in which the furnace, Fig. 2, described before, was used. Uranium was filled into the furnace in form of pieces of pin-head size. The furnace was heated in the saltpeter bath, which also contained an iron tube in which was the thermo-element in porcelain tubes.

An excess pressure of 125 kg per square centimeter was used with a gas mixture of three volumes hydrogen and one volume nitrogen.

Velocity of gas	Temperature	Percentage of NH ₃
32 liters per hour	570 deg. C.	5.65
25.7 liters per hour	595 deg. C.	6.54
9.5 liters per hour	496 deg. C.	9.1
2 liters per hour	503 to 493 deg. C.	11 to 11.9

In the periodic system manganese has as its left-hand neighbor chromium and as its right-hand neighbor iron. *Iron* is also a catalytic agent for ammonia formation. Below the iron in the same vertical column in the periodic system, we find ruthenium and osmium. Both can be produced in very finely powdered form by similar methods. Ruthenium, however, did not prove a very effective catalytic agent, while *osmium* is very effective. It is notable that in both the iron group and the chromium group of the periodic system the elements with the highest atomic weight—uranium and osmium respectively—are the most effective catalytic agents.

From the different osmium experiments described in the report only the following one may be mentioned, in which a osmium powder layer of 14 mm length and 4.5 mm diameter was used:

Velocity of gas	Pressure	Temperature	Percentage of NH ₃
20 liters per hour	156 atm.	592 deg. C.	4.75
10 liters per hour	156 atm.	572 deg. C.	5.91
1.5 liters per hour	174 atm.	550 deg. C.	7.9
1.5 liters per hour		521 deg. C.	9

The conclusion of the report is an account of a test of longer duration with gas circulation, made to show that the continuous operation of the process with liquification of the ammonia and with the return of the remaining gases into the furnace meets with no difficulties. The heat and cold regenerators proved very efficient.

New Polariscopes Tube.—The Bausch & Lomb Optical Company of Rochester, N. Y., has just placed on the market a new polariscopes tube, constructed after the design of Frederick Bates, of the U. S. Bureau of Standards. It is a scientifically designed tube suitable for nearly all polariscopic work and especially adapted to sugar polarizations. It has been adopted by the Bureau of Standards.

The Western Metallurgical Field

New Ore Discovery in Colorado

It has been so many years since a promising discovery of ore was made in a new district in Colorado, that the recent announcement of such a discovery in Eagle county has attracted more than usual attention. The scene of the new discovery is on Brush Creek, in the vicinity of the Mount of the Holy Cross, where an earlier discovery of gold and silver ore a number of years ago gave prominence to the camp of Fulford. On account of the difficulties encountered in prospecting and transportation, nothing of importance developed from the earlier discovery; but conditions are now more favorable, and it is probable that the coming spring and summer will find this district thoroughly prospected. At present the snow is too deep for widespread prospecting.

The ore is unlike anything thus far known in Colorado. In appearance it is a grey sandstone, containing minerals of silver which are variously reported to be chloride, bromide and sulphide. Assays indicate the presence of silver in large quantities, several hundred ounces per ton. The richest ore shows bluish spots which, when heated under the blow-pipe or in a muffle, yield metallic silver. A peculiarity of the ore is that some of it contains vanadium in small quantities, say about 1½ to 2 per cent V₂O₅. Should later development show that there is a considerable deposit of this kind of ore, a nice metallurgical problem will be presented to those who ultimately mine and treat it. Such a content of vanadium is not appreciably lower than that of some ores which are treated for this metal alone, and it should not be neglected in this high-grade silver ore. This is a condition which should be recognized before too much of the ore is smelted for the recovery of silver alone, in which event the vanadium would be lost in the slag.

Smuggler-Union vs. Liberty Bell

In June, 1912, the suit of the Smuggler-Union Mining Company against the Liberty Bell Gold Mining Company, both of Telluride, Colo., was tried in the United States district court in Denver. The plaintiff alleged wrongful extraction of ore by the defendant. The result of the first trial was favorable to the plaintiff company which was awarded damages in the sum of \$403,000. The case was appealed by the defendant, and was later heard by the United States Circuit Court of Appeals, sitting in St. Louis. This court affirmed the decision of the lower court on March 3, 1913. It is expected that this decision will be final unless the Liberty Bell company can show error sufficient to carry the case to the United States Supreme Court.

Copper Smelting Resumed at Tooele

During the labor strike in the Bingham district, Utah, last fall, the International Smelting & Refining Company ceased smelting copper at its plant at Tooele, Utah. This action was necessitated by the closing of several mines from which the supply of copper ore was received. Since that time the conditions have improved and are again normal, and the International company is again smelting copper concentrates, with no prospect of further interruption. In the lead smelting department of this plant, four blast furnaces are now in operation.

Consolidated Mercur to Close

For the past two years the annual reports of the Consolidated Mercur Gold Mines Company, Mercur, Utah, have foreshadowed the probable necessity of ceasing operations unless conditions improved materially. The shareholders were asked to authorize the directors to dispose of any or all of the property, which authority was granted. It appears that conditions are not improving and that the famous old mine and mill will be closed within a month or so. This act will close the career of a company which, through its technical staff, contributed greatly to the progress of the cyanide industry, particularly through the invention by George Moore of the vacuum filter.

Notes on Chemistry and Metallurgy in Great Britain

From our Special Correspondent.

Annealing Muffles

At the February meeting of the Birmingham Section of the Institute of Metals Mr. E. H. WALL read a paper on "Annealing Muffles," which dealt with the improvements made in muffles during the last fifteen years or so. The fire-boxes were now made smaller than they were formerly and the flues were arranged to distribute the heat better along the sides and under the beds, while the combustion was more perfect; and as the muffle was enveloped with heat there was an economy in fuel. Cross-fired muffles could be got to work more rapidly than end-fired muffles, and the temperature was maintained with greater regularity. The best arrangement for a muffle was one in which the flame, after passing the bridge and bed, went into a series of graduated portholes opposite to the fire-box; and, after passing these portholes, the waste gases entered a flue under the bed, traveled round the muffle two or three times, according to its width, before making their way into stack. There was still a great deal to be done in improving economical appliances, particularly with regard to gas-firing. At the present time 1.75 cwt. to 2 cwt. of coal was consumed in the gas producer for 1 ton of iron annealed, as against 1 ton of coal to anneal $1\frac{2}{3}$ ton in former times; but even now the theoretical heating was a long way from being attained. If it could be attained 1 ton of coal would heat 30 tons of iron to a welding heat. If the annealing point of copper were put at 1400 deg. Fahr., then 133 heat units would be required to raise 1 pound to that temperature, or 1 ton of coal would be enough to anneal 90 tons of copper. It was not probable that this ideal would ever be attained, but modern muffles had brought down the consumption of fuel by 30 to 50 per cent.

The Institution of Mining and Metallurgy

At a recent meeting of the Institution, papers giving a historical sketch of the Copper Queen Mines at Arizona and dealing with the varieties of ores and methods of treatment, as well as the geology of the Bisbee deposits, the power plants and the reduction works, were presented by Dr. JAMES DOUGLAS, Mr. ARTHUR NOTMAN, Mr. CHARLES LEGRAND and Mr. G. B. LEE.

From Mr. Legrand's paper it appears that the plant at Bisbee consists of five water-tube boilers, with super-heaters and economizers, each of 407 hp, which were fired with California crude oil the thermal value of which was about 18,500 B.t.u. per pound. The pressure of the boilers was 155 lb. per square inch, and the super-heater raised the steam to a temperature of about 440 deg. Fahr. There were three two-stage air compressors, two of which were driven by compound steam engines and delivered about 3500 cubic feet of air per minute, and the third was driven by a triple-expansion, four-cylinder engine and put out 7000 cubic feet per minute. The compressed air at 90 lb. per square inch worked five direct-motion hoists, two geared hoists, some hundreds of air drills and machine tools, some air-jet blowers and some small hoists. Three 500-kw vertical steam turbo-generators supplied three-phase current at 2300 volts, 60 cycles, and part of it was transformed down to 260 volts direct current by two 200-kw rotary converters. Several motor driven hoists were in use and all the pumps were worked electrically, while there were eleven locomotives underground, the largest of which weighed about 7 tons.

At Douglas eight boilers were of 500 hp each, with economizers similar to those at Bisbee; and four were of 520 hp, more than 400 yards distant from the boiler house, and utilized the waste heat from the reverberatory furnaces, working at 180 lb. per square inch to compensate for loss of pressure in the long steam main. There were eleven blowers driven by tandem compound engines, and six of these delivered 300 cubic feet per revolution, and the other five 200 cubic feet per revolution; five blowing engines for converters each delivered

6000 cubic feet of air per minute, and one engine 12,600 cubic feet per minute. There were four 400-kw, d.c., 260-volt generators which drove from 50 to 60 motors, 7 cranes and 14 locomotives; and two 750-kw mixed pressure, turbo-alternates supplied three-phase currents at 2300 volts, 60 periods, which was transformed to 44,000 volts to supply current to a mine 67 miles off.

Mr. G. B. Lee said that the original plant put down in 1902 at the reduction works at Douglas consists of five blast furnaces and four converters, and that plant was duplicated in 1906. The furnaces and converters were arranged on the opposite side of a 60-ft. crane-way, and the re-lining plant was at one end of the main building, which measured 900 ft. x 150 ft. The ores were stored in rock-walled pits, 800 ft. long, 38 ft. wide, and 11 ft. in depth.

Formerly the smelting mixture gave mattes with fully 45 per cent of copper, but gradually changes had been made and the matte was now only from 30 to 35 per cent. A further increase in the proportion of sulphide ores was indicated, and the smelting mixture was becoming finer and less suitable for blast furnaces. The altered conditions necessitated a change in the plant and an enlarged capacity, and consequently six 18-ft. McDougal roasters and two reverberatory melting furnaces, each 91 ft. x 19½ ft. were put down in addition.

Previously the whole of the smelting had been carried on in blast furnaces; and the fineness of the charge together with much alumina in the ores made treatment difficult, while large quantities of dust resulted from the disintegration of the pyrites and the fine concentrates. It was found that the loss of metal in the slag did not amount to the metallurgical loss; and the waste appeared to be due to dust going away with the smoke, but as the volume of the smoke was a million cubic feet per minute, bag filtration was impracticable. Experiment demonstrated that when the smoke was passed through a chamber at 150 feet per minute or less the fine dust settled at short distance, and by introducing wire screens 3 ft. apart an equal settlement was obtained at twice that velocity, and the section of the chambers could be considerably reduced. The screens were so contrived that there were no direct channels through the chamber; and the arrangement was the very opposite of the old system of passing the gases at a high velocity through long flues.

The Iron and Steel Institute

The annual meeting of this Institute will be held in London on May 1 and 2 at the Institution of Mechanical Engineers. The Bessemer medal will be presented to M. Adolphe Greiner, director of the Société Cockerill, of Seraing. Arrangements are in progress for holding the autumn meeting at Brussels.

The Society of Chemical Industry

Two departmental committees of the Home Office were appointed in 1911 to investigate the danger attaching to the use of lead compounds in paints for buildings, vehicles and other purposes; but up to the present they have not presented any report.

Professor H. E. ARMSTRONG and Mr. C. A. KLEIN have recently submitted a paper to the Society of Chemical Industry on "The Behavior of Paints under the Conditions of Practice, with special Reference to the Aspersions cast upon Lead Paints." The authors investigated the contention that illness resulting from living in freshly painted rooms and the bad effects felt by painters were due to volatile compounds of lead, and their experiments led to the conclusion that suspended dust and not lead vapor was the cause; and that earlier experimenters had failed to adequately safeguard against dust being carried forward. With regard to the statement recently made by Professor Baly that noxious organic vapors were emitted during the drying of paints, the authors were satisfied that the ill-effects generally experienced in freshly painted rooms were produced by turpentine vapor, and that symptoms commonly attributed to lead poisoning were simply the effects of turpentine. They were of opinion that the dangers attending the use of lead compounds in paints were merely mechanical dangers of which the chief arose from dust; and that painters ran very

little risk of harm if production of dust were duly prevented.

On the question of the comparative qualities of the materials used as white paints they considered that for outside use, where rain and the impurities always found in town air had access to the painted surfaces, lead paints only were suitable. Zinc paints were not adapted to outside work, but were very serviceable for interior use.

The Institute of Chemistry.

Mr. W. J. A. BUTTERFIELD delivered his second lecture on "Chemistry in Gas-Works" on January 31, when he dealt mainly with impurities, their effects on plant and fittings, and methods of purification. The question of by-products, manufacturing costs, work analyses, efficiency of burners for lighting and heating, further requisite chemical research and the qualifications and training of a gas-works chemist were also considered.

The obstruction caused by the deposition of one hundredth part of a pound of naphthalin in a main was experimentally demonstrated; and the processes employed for the recovery of ammonia, the removal of sulphuretted hydrogen, and cyanogen were described and materials exhibited. In connection with the elimination of sulphur impurities remaining after the removal of hydrogen sulphide, the lecturer referred to the reintroduction in London gas-works of the heated contact-substance process for separating carbon disulphide first proposed by Dr. A. Vernon Harcourt more than 40 years ago. Physical tests of tar and pitch were of little value because their physical properties altered with aging, and the nature and extent of the changes could only be ascertained by chemical analysis. The calorimetry and photometry of gas were demonstrated, and the evolution of the modern 10 c.p. Harcourt pentane standard lamp was illustrated by five successive specimens. The lecturer said that he had made comparative tests of about 150 Harcourt pentane lamps and had never found a greater divergence than 0.2 per cent, except where constructional faults existed. The disturbing effects of variations in atmospheric conditions on photometric measurements were discussed.

Transmutation or Expulsion

The general opinion among practical scientists is that Professor Sir J. J. Thomson has effectually disposed of the supposed conversion of hydrogen into helium and neon by Sir W. Ramsey, Professor N. J. Collie, and Mr. H. Patterson. There can be little, if any doubt about the correctness of Professor Thomson's view that the helium and neon are occluded in and are expelled from the electrodes, and possibly also the glass, because he found that these gases ceased to be evolved after a few days, but were again in evidence when fresh electrodes were substituted. If the transmutation claimants knew of Thomson's results before their papers were made public, it is rather surprising that they should not have withheld their communications until they could establish their case on a firmer foundation.

Engineering Imports and Exports

The returns issued by the Board of Trade for the month of January show a satisfactory all-round improvement in the trade in engineering materials and products as compared with January, 1912—the only decrease being in imports of electrical goods. Iron and steel, including manufactures, were imported to the value of £1,469,352, an increase of £390,759, and were exported to the value of £4,827,729, an increase of £473,491. Imports of other metals, including manufactures, amounted to £2,938,770, an increase of £184,530; and exports reached £1,272,040, an increase of £315,174. Imports of electrical goods totaled £136,192, a decrease of £3,804; while exports touched £368,618, an improvement of £93,527. Imports of machinery were of the value of £673,727, and show an increase of £158,780; and exports went to £3,112,474, an increase of £415,832. Imports of new ships are put at only £1,303, which amount however, is £1,246 better; and exports rose to £490,425, an increase of £307,203.

Market Prices, February, 1913

Copper opened at £69. The decline in price which has been

continued this month, has followed the same slope from Jan. 14th, dropping rather more sharply about the middle of the month. It was £66 on the 6th, £64 on the 18th, and since rather firmer, closes £64.17.6.

Tin opened at £230 and took a sharp and heavy drop to £220 on the 6th. It then recovered to £224.10 (10th) and fell away more slowly to £217 (18th), recovering to £219 on the 20th, but falling again to £213 by the 24th, touched £220 on the 26th, and closes £218.

Hæmatite, opened 81/3 and remained steady till the 14th, when it dropped to 80/6 in three days. Kept at this price till the end of the month.

Scotch Pig opened 71/3, and has been exceptionally uneven, being practically the same price on the 11th, and dropping to 65/- by the 18th, closed better at 66/7½.

Cleveland was 65/9 at the end of January and remained about 65/- till 13th, when it dropped away 4/- in five days reaching 60/3 on the 18th. It has since improved slightly, closing 61/-.

Lead opened £16.17.6 and remained firm till mid-month, after rather lower, £16.12.6 on 18th, recovering to £16.17.6 on the 20th, but closing £16.12.6.

	£ s.d.
Aluminium, per ton.....	89. 0.0
Alum, lump, loose, per ton.....	5.10.0
Antimony, per ton.....	36.10.0
Borax, per ton.....	17.10.0
Copper ore, 10 to 25 per cent, per unit.....	11/7½ to 12.1½
Copper sulphate, ton.....	23. 0.0
Carbolic acid, liquid, gal.....	1.5
Caustic soda, 70 per cent, per ton.....	10. 0.0
Ebonite rod, per lb.....	5.3
Hydrochloric acid, per cwt.....	5.0
India rubber, Para fine, per lb.....	4.0½
Mica, in original cases, medium, per lb.....	3/6 to 6.0
Petroleum, Russian spot.....	8½
Quicksilver, per bottle.....	7.15.0
Sal. Ammoniac, lump, firsts, delivered U.K. per ton..	44. 0.0
Sulphate of ammonia, per ton.....	14. 5.0
Sulphur, recovered, per ton.....	5.10.0
Shellac, cwt.....	4. 5.0
Platinum, oz.....	9. 5.0
Tin ore, 70 per cent, per ton.....	143 to 145. 0.0
Zinc, Veille Montagne, f.o.b., Antwerp.....	28.17.6

Differences Since January 31st

Higher	£ s.d.	Lower	£ s.d.
Shellac, cwt.....	12.6	Alum, ton.....	5.0
		Copper ore, unit.....	1.3
		Copper sulphate, ton.....	2. 5.0
		Carbolic acid, gal.....	4
		India rubber, lb.....	4
		Tin ore, unit.....	5. 0.0
		Zinc, ton.....	17.6
		Copper, ton.....	3. 2.6
		Tin.....	12. 0.0
		Lead, ton.....	5.0
		Iron Hæmatite.....	9
		Scotch Pig.....	4.7
		Cleveland.....	4.3

The new Van Ryn Deep mill on the Witwatersrand in South Africa, will be equipped with 80 stamps of 1000 lb. weight, eight tube mills each 16 ft. 6 in. by 6 ft., sand leaching tanks and a Butters slime filtering plant. The estimated capacity of the stamps is 20 tons per day, or 40,000 tons per month of twenty-five working days.

Mexican smelting plants of the American Smelting & Refining Company, at Matahuila and Velardena were compelled to close for a time during February on account of shortage of fuel, caused by interruption of traffic. The company's other smelters at Monterey, Chihuahua and Aguas Calientes have been running without interruption.

Recent Chemical and Metallurgical Patents

Electric Furnaces

Means for Joining Electrodes.—The managers of the electrode factory of Gebr. Siemens & Company, Lichtenberg, near Berlin, in Germany, EDUARD VIERTEL and HERMANN VIERTEL, have improved the method of joining two tapped electrodes together with true centers. Screw-threads can be made exactly in the electrodes with very great difficulty only and, as a rule, the screw-threaded nipples are made a little smaller than the appertaining screw thread. On account of the play between the nipple and the tapped end, the joined electrodes are frequently not in alignment. To improve this condition the ends of the electrodes are provided with cylindrical, conical or otherwise shaped centering means, for example, projections or recesses. Fig. 1 shows the wrong way of joining electrodes. Fig. 2 shows an annular ring projecting from No. 2 electrode and fitting into the corresponding recess of 1 electrode. In Fig. 3 both electrodes 1 and 2 have recesses, and a centering annulus is constituted by a separate member 4, which exactly fits the two recesses. These means allow to bring the electrodes into exact alignment. (1,049,624, Jan. 7, 1913.)

Carbon Electrode for Electric Furnaces.—Mr. BERTHOLD REDLICH, of the Planiawerke, Ratibor, Germany, has invented a new form of carbon electrode with metallic core. When metal, and especially iron, is inserted into the electrode before burning, it absorbs carbon and becomes brittle. A better contact is obtained by casting molten metal, or metallic alloys, into the hollow spaces provided in the electrode. In doing this care has to be taken to give the metal such a shape that the contraction during solidification does not separate it from the carbon. The drawings in Fig. 4 illustrate without

The process consists in subjecting the surface of the fabric or screen to the action of a sand blast, which, together with the material used, is so regulated that only the heavy exterior coating of the fabric is penetrated. The inventor claims that this process does not cause the deterioration in the fabric such as arises from the use of acid baths; also that the cost of renewals and repairs will be less than with the latter method.

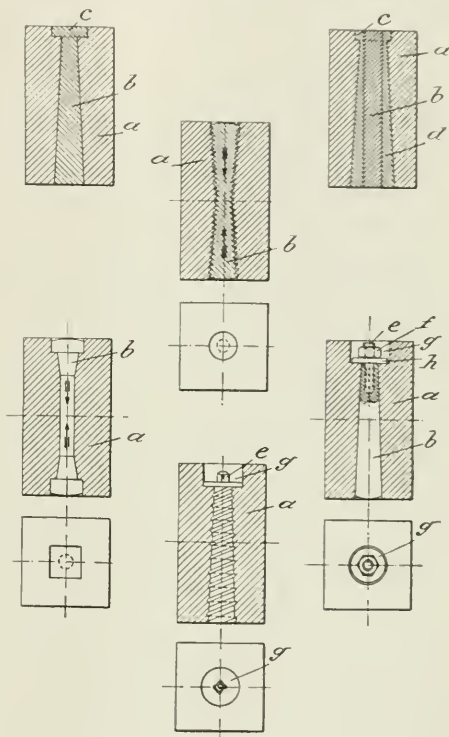
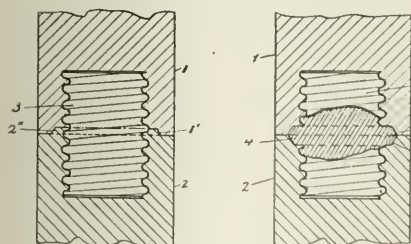


FIG. 4.—VARIOUS FORMS OF CARBON ELECTRODES WITH METALLIC CORE

Fig 2

Fig. 3



FIGS. 1 TO 3.—METHODS OF JOINING ELECTRODES

further explanation various ways of accomplishing this result. (1,048,581, Dec. 31, 1912.)

Gold and Silver.

Cleaning Filter Fabrics and Screens.—The adhesion or precipitation of solids on filter cloths and screens reduces their efficiency, and necessitates occasional cleaning. An improved method of cleaning these devices is proposed by Mr. MAYNARD J. TROTT, of Colorado Springs, Colo., the patent being assigned to the Dorr Cyanide Machinery Company, of Denver, Colo.

The method has been demonstrated on screens and filter fabrics in actual use. (1,052,101, Feb. 4, 1913.)

Lead, Zinc and Copper.

Roasting for Magnetic Separation.—The customary method of roasting mixed sulphide ores for subsequent magnetic separation of the iron, consists in freely exposing the ground ore to an oxidizing flame or heat for a period of several hours. In a process proposed by Mr. JAMES B. ETHERINGTON, of Winthrop, Mass., there is a radical departure from the old method, in that the ore is roasted in a practically non-oxidizing atmosphere, at a temperature that is precisely controlled, and for a brief time. All particles of the ore are subjected to the same degree of heat for the same time, and loss by volatilization is avoided. By avoiding a strong draft through the furnace, a minimum of dust is carried out of the stack.

The furnace is shown in Fig. 5. It consists of an unlined cylinder 7, approximately 6 ft long and 3 ft in diameter. It is practically surrounded by a hood 41, through which cooling air may be drawn by a fan not shown, but placed above the outlet 42. A hydro-carbon burner 10 is placed at the end of the furnace, through which oil and sufficient air to cause complete combustion, are admitted. A drying and preheating cylinder 24 is attached to the furnace, and both have baffles 22 on their interior surfaces for the purpose of causing the ore to fall in

a cascade through the flame. The products of combustion escape by the stack 32, and the roasted ore is discharged through ports 17 adjusted by gates 20. The entire mechanism is supported on rollers and adapted to be rotated by a worm gear.

In operation, the ore is fed through the hopper 30 and travels downward through the furnace owing to its inclination. On reaching the furnace chamber, the particles are subjected to a non-oxidizing heat caused by the blast of oil and air which

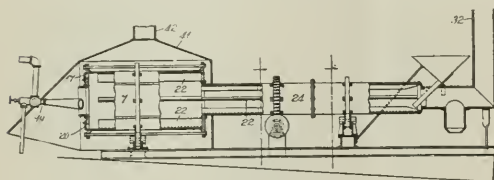


FIG. 5.—FURNACE FOR MAGNETIC ROASTING

is directed onto the wall of the furnace and caused to swirl therein. The rate of feed and size of ports are so adjusted that the ore remains in the furnace for only a few minutes, whereupon it is sufficiently magnetized to enable profitable magnetic separation to be made. The inventor claims that longer time or more oxidizing conditions are not necessary, and that if employed they only cause undue oxidation and volatilization of the constituents of the ore. (1,051,494, and 1,053,486, Jan. 28, and Feb. 18, 1913.)

Method of "Amphidizing" Copper Ores.—In describing the apparatus devised for carrying out his process of rendering the copper in ores soluble, Mr. CHARLES S. BRADLEY, of New York City, gives the following definitions: An "amphigen" is an element which combines with metals to form either an acid or a base, as for example oxygen. An "amphid" is a salt of an acid and a base, each of which contains an "amphigen," as for example copper sulphate, $\text{CuO} \cdot \text{SO}_3$. The apparatus which is used to form the "amphid," is called the "amphidizer."

The inventor proposes by amphidizing to convert the copper contents of ores into soluble form, such as copper sulphate, and then separate the latter from the gangue by solution, later precipitating the copper. His process of amphidizing is continuous in operation. The apparatus consists of a rotary cylinder with feed and discharge at the same end, the former being at the axis of the cylinder and the latter at the periphery. Heat is applied by any suitable burner placed at the end of the cylinder opposite the feed and discharge end. The

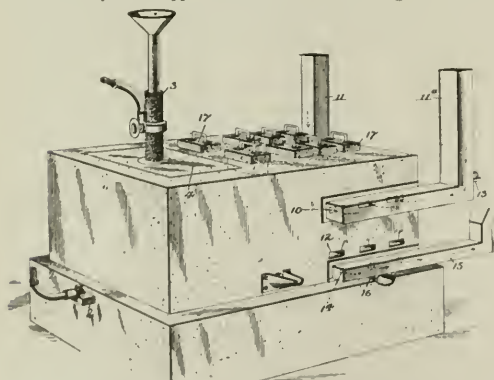


FIG. 6.—PERSPECTIVE VIEW OF ELECTRIC ZINC FURNACE

ore advances through the cylinder by one set of passages and returns by another. This is accomplished by subdividing the reaction space transversely into a number of reaction chambers, the alternate chambers being connected by bridging passages so inclined that the crushed ore advances by gravity as the drum rotates. The intermediate chambers are connected by

bridging passages inclined the other way so that the crushed ore returns by the action of gravity as the drum rotates.

By this arrangement the returning amphidized ore transfers its heat to the incoming ore, and a regenerative effect is produced. This results in an even distribution and regular heat which is so essential causing chemical reactions requiring a definite and steady temperature. In this case the temperature is maintained at about 550 deg. C., or between the dissociation temperatures of iron and copper sulphates. In some cases the heat of reaction will be sufficient to maintain the proper temperature, and it may be necessary to use artificial cooling to prevent the temperature rising too high. (1,052,793, Feb. 11, 1913.)

Electric Zinc Furnace.—In Figs. 6 and 7 are shown two views of a form of electric furnace for the reduction of zinc ores, devised by PETER E. PETERSON, of Butte, Mont. As Mr. Peterson is reported to have made some successful experiments on the electric reduction of the zinc ores of Butte, his furnace is interesting. The furnace consists of a smelting retort 1 in

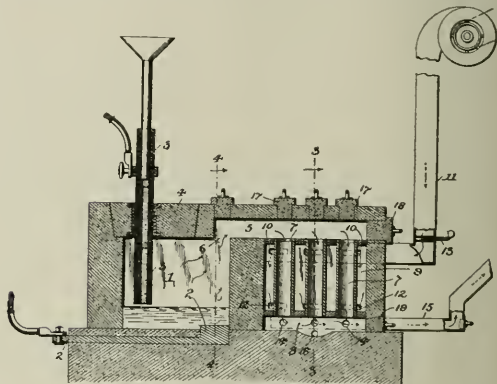


FIG. 7.—LONGITUDINAL SECTIONAL VIEW OF ELECTRIC ZINC FURNACE

which the material under treatment is volatilized by an electric current passing between electrodes 2 and 3. The latter is hollow and surrounds the feed pipe through which ore is introduced into the furnace. The condenser consists of a vapor chamber 5 connected with condensing tubes 7, which extend downwardly to a spelter chamber 8. The tubes are surrounded by a wind box through which cooling air is forced by a fan, entering both sides of the condenser chamber through ducts 11 and 11a. After circulating around the condenser tubes, the air is discharged through ports 12. Uncondensed fumes and vapors are discharged from the condenser through vents 14, leading into the duct 15. The spelter condenses in the tubes 7 and settles into the chamber 8, from which it may be withdrawn through a tap 16. Suitable covers for the condensing tubes are provided as at 17 and 18. It will be noticed that the zinc vapors and cooling medium pass in the same direction, an arrangement which the inventor says has been found highly effective. (1,051,512, Jan. 28, 1913.)

Chemical Industry.

Extracting Chlorine, Iodine and Potassium Hydroxide from Seaweed.—Messrs. FRANK K. CAMERON and RICHARD B. MOORE, of Washington, D. C., and Indianapolis, Ind., respectively, have patented a method of recovering useful products from the ash of seaweed and other marine forms of algae, and have permitted the use of the process by any person in the United States, without the payment of any royalty.

According to one method of treatment, the ash of seaweed is lixiviated with water, the solution drawn off from the solid matter and evaporated to a point where the less soluble salts begin to crystallize. The mother liquor then contains all the soluble iodides and some of the chlorides. It is then elec-

tolized in the anode compartment of a diaphragm cell, whereupon chlorine and iodine appear at the anode and potassium hydroxide at the cathode. The chlorine which escapes can be collected, and the iodic acid which is formed can be reduced to iodine by treatment with ferrous sulphate and sulphuric acid.

In a second method, the ash is stirred into water in the anode compartment of a cell. The soluble salts pass into solution, and the same changes occur on electrolysis as described in the first method. (1,051,984, Feb. 4, 1913.)

Iron and Steel.

Manganese Steel.—Mr. HENRY D. HIBBARD, well-known by his research work on impurities in steel, has been granted the following patent for making manganese steel. The object is to utilize manganese steel scrap without material loss of the manganese present in the scrap. If steel scrap containing from 11 to 14% manganese would be charged and worked in the ordinary way in open-hearth furnaces, nearly all the manganese would have to be oxidized before any carbon could be eliminated. This loss can be avoided by melting the manganese steel scrap while it is covered by a bath of decarbonized iron. The latter protects the iron from oxidation while fusion is proceeding. The process can be carried out in an electric furnace. The ordinary charge of an open-hearth furnace can, for instance, be worked down to 0.15% carbon, and the manganese steel scrap thrown into this bath, or the scrap can be heated in the furnace to not more than red heat and then be covered with molten iron of the above carbon contents. When using an electric furnace the latter will be provided with auxiliary heating means. The scrap which is again heated to not more than red heat will be covered with the molten iron, and the melt completed by the application of the electric heating element. When producing manganese steel the small unavoidable loss and the manganese required by the iron have to be added in the form of ferro-manganese. (1,051,840, Jan. 28, 1913.)

Tunnel Furnace.—Mr. ARTHUR RAMEN of Helsingborg, Sweden, applies certain improvements to tunnel furnaces of the canal type, such as are used for the burning of ore briquets at high temperatures and for other similar purposes. The wagons or movable hearths used to carry the material to be treated through these furnaces had a comparatively short life due to the disintegration of the covers and linings of the hearths. This trouble is overcome by providing, in the side walls of the furnace, overhanging portions beneath which the sides of the hearths pass in traveling through the furnace. These overhanging portions are made of hollow tiles and adapted to cooling. The details can be seen from the accompanying drawings and need no description. The tile 14 with a dividing wall 15 and in and outlets 16 and 17 respectively for passing a liquid or other cooling medium through is set up at suitable intervals and bears inclined side walls. By means of this it supports a longitudinal arch in the side wall of the furnace. The tile 10 runs all along the furnace and is cooled in a similar way. (1,051,634, Jan. 28, 1913.)

Process of Improving the Quality of Malleable Iron and Steel.—The late OTTO THALNER, formerly with Bismarckhütte, in Germany, and later with Electro Stahl, in Reinscheid, worked out shortly before his untimely death a new process for refining steel in an electric furnace on acid hearth. The patent claims that the steel treated according to the process described has a peculiar crystalline grain very similar to, or substantially identical with, the grain which has always been observed in crucible steel, and which apparently gives to such steel its great tenacity. The process may be operated in a hearth-furnace and more particularly in an electric arc furnace, on acid or basic hearth, the heat being furnished by gas, electricity or in any other suitable manner. The feature of the process is the addition to the slag of a metallic oxide capable of giving oxygen to the bath of liquid iron and also to add carbon to such bath. The carbon is best used as carburite, which offers no difficulty in providing carbon both in and below the slag. The amount

of iron ore should not be so much as to cause red-shortening or blowholes in the metal, it will generally be found that between 1 and 2½ per cent of the weight of the liquid metal will be sufficient, while the weight of carbon to be added depends on the amount already in the metal and the finished carbon desired. As an example, it is mentioned that a bath of 5000 kg. desired to be finished at 1 per cent of carbon received from a 15 kg. carbon in the form of carburite, 30 to 35 per cent sand, 30 to 35 per cent lime, 5 to 10 per cent manganese ore to increase fusibility, and 25 to 30 per cent of iron ore. A correspondingly different slag would be made up for a basic hearth. In several chemical equations of an entirely hypothetical character the inventor gives one way of explanation of the reactions which are liable to occur during the refining action. (1,053,454, Feb. 18, 1913.)

Iron-Nickel-Copper Alloy.—Mr. G. H. CLAMER of Philadelphia, Pa., has obtained a patent on replacing part of the nickel of nickel steels with low nickel contents by varying amounts of copper. The latter element is known to raise the elasticity and tensile strength of such nickel steels without diminishing the ductility. Carbon is allowed to be present up to 0.5 per cent. Alloys claimed by this patent contain nickel up to 6 per cent and copper up to 2 per cent. A commercially profitable way of producing these alloys is the use of metal metal or an alloy of 68 per cent nickel, 2 per cent iron and 30 per cent copper. (1,050,342, Jan. 14, 1913.)

Method of Briquetting Flue-dust.—Mr. GEORGE L. COLLARD, of Sharon, Pa., has invented a new method of briquetting the flue dust from blast furnaces or other finely comminuted ore. He mixes limestone, sulphuric acid, and water with the dust until a pasty condition is reached, and molds the plastic mass by hand or any of the ordinary forms of briquetting apparatus. In practice if the dust contains below 0.5 per cent of lime, as is frequently the case, 3 to 4 per cent of crushed or powdered limestone are mixed with it. Sufficient water is then added to moisten the mass and render it suitable for molding. It is then treated with the sulphuric acid which causes the agglutination of the gas and thereby produces the good result of mixing the plastic mass. It keeps the dust in a coherent form until the temperature is raised sufficiently high to melt the metal. (1,047,174, Dec. 17, 1912.)

Process of Oxidizing Ferros to Ferric Solutions.—Messrs. ALEXANDER McKECHNIE and F. G. BEASLEY, of Birmingham, England, have found that the oxidation of ferrous sulphate and chloride solutions into the ferric salts can be rapidly and completely accomplished by subjecting same at a temperature exceeding 100 deg. C. to the action of air under a pressure of 10 atmospheres or upward. The oxidation proceeds so rapidly to completion that large volumes of ferric solutions can be cheaply produced under commercial conditions, whereas the oxidation by atmospheric oxygen or by air in open tanks under normal temperatures and atmospheric pressure proceeds so slowly that the process is practically useless. (1,047,826, Dec. 17, 1912.)

Synopsis of Recent Chemical and Metallurgical Literature.

Gold and Silver

Zinc Wafers for Precipitation of Cyanide Solution.—At the January meeting of the Chemical, Metallurgical and Mining Society of South Africa, Mr. JONAS MACARTHUR read a paper dealing with the use of zinc as a precipitant for cyanide solutions, and proposing the substitution of small sheets or wafers of that metal for the customary filament or shaving. The objections to the use of shavings are obvious: the cutting of the filament weakens the fibrous structure of the metal, and causes fractures which are plainly visible under the microscope. These fractures result in a great quantity of "short zinc" and zinc slime which cannot be separated mechanically from the valuable precipitate, necessitating acid treatment. Bullion may contain from 50 per cent to 100 per cent of such material, necessitating elaborate methods of refining. Another

objection to zinc shavings is the difficulty of packing them uniformly in the box, giving rise to spaces of varying resistance to the flow of solution, and consequent imperfect contact and precipitation.

Mr. MacArthur proposes to use wafers of sheet zinc, cut into sizes varying from 1 in. to 2 in. long and $\frac{1}{4}$ in. to $\frac{1}{2}$ in. wide. There is no virtue in any particular size, but oblong pieces are found to pack better in the box than square ones. The resistance of these pieces to the flow of solution is greater than that of shavings, so that boxes must be made with twice the customary fall, and may be only half as deep.

At the Caveira mine in Portugal these wafers have been in use since 1907, and the bullion precipitate is found to contain only about 3 per cent zinc and 12 per cent of other foreign matter. The usual acid treatment is dispensed with. Wafer zinc is used at another mine in Italy and one in Mexico, and the results are said to be excellent. The zinc is found to dissolve evenly, becoming thinner and thinner, until at the end of three or four weeks the wafers disappear altogether without having their structural strength impaired, and without leaving distinguishable debris. The wafers pack in the box in an irregular overlapping fashion, and as their edges are slightly burled or distorted by the cutting process, they do not lie flat on each other, but provide narrow channels for the passage of solution.

A Small Pachuca Agitator for Testing.—In the Colorado School of Mines Magazine for March, 1913, Mr. JOHN GROSS describes a small air agitator which he uses for testing purposes. It consists of a riveted pipe, 24 in. long and 6 in. in diameter, joined to a 7-in. glass funnel. The joint is made with a melted mixture of scrap rubber and sulphur, which is poured into the space between tube and funnel. This mixture hardens quickly, makes a tight joint, but is not brittle. Air is introduced through a piece of glass tubing fitted into the neck of the funnel by means of a rubber stopper, or any other convenient method of holding the tube in place. The lift-pipe is $\frac{3}{4}$ in. in diameter and extends from near the apex of the funnel to within a short distance of the top of the agitator. This lift-pipe is supported in place by suitable means at top and bottom. A settling chamber can be formed at the top of the agitator for the purpose of drawing off clear solution, by inserting a short cylinder, say 4 in. in height and of smaller diameter than the main cylinder, and rising 1 in. above the level of the pulp in the tank. By means of a siphon, clear solution can be decanted as desired.

With slime pulp not thicker than 2:1, the agitation is perfect, and no settling or accumulation occurs in the funnel; but with thicker pulps some thickening and settling occurs. With reground sand this settling does not occur, as the sand seems to have a scouring action which keeps the funnel clear of accumulations.

Compressed air can be obtained from an ordinary tank and pump such as is used for assaying with gasoline. When pumped to 30 lb. pressure the air will last for 20 minutes. The apparatus will serve for the agitation of 32 lb. of dry ore and the necessary solution.

Recovery of Gold by Volatilization.—In the December, 1912, monthly Journal of the Chamber of Mines of Western Australia, Mr. BEN HOWE describes "a process, the application of which, at any rate, is new to the metallurgy of gold." (?) It consists in roasting the ore with salt, volatilizing the gold as chloride, dissolving the latter in a water spray and recovering the gold in a filter press. He states that the gold is suspended, "as a black powder" in the water and readily recoverable in the press. Salt is added in quantities varying from 1 to 4 per cent, and the ore is crushed to different degrees of fineness; the finer the ore the quicker the volatilization. A rotary furnace is used, the temperature being maintained at about 1000 deg. C. A higher temperature sinters the ore, and a lower one does not effect volatilization. Laboratory experiments can be made by mixing ore with salt and subjecting the mixture to muffle-heat. The percentage of volatilization is said to be usually over 90 per cent. This process is attracting attention in

Australia on account of its experimental application to refractory ores from the Gwalia Consolidated mines in the East Murchison gold district.

Copper

Notes on Smelting Practice at the Copper Queen Smelter.

—In a paper on the Copper Queen mines and works, Douglas, Arizona, collaborated by members of the technical staff of the company, the portion devoted to the reduction works is contributed by Mr. GEORGE B. LEE. The entire paper is published in Bulletin No. 101, Inst. Min. & Met.

The original plant was erected in 1902, and consisted of five blast furnaces and four converters; this was doubled in 1906. The furnaces and converters are arranged in parallel lines on opposite sides of a 60-ft. craneway. The entire plant is on a nearly level site, there being a slope of about $\frac{1}{4}$ per cent from the receiving end to the furnaces. This makes it possible to run trains by easy grades from the ore pits to the charging floor, the distance being about 2000 ft.

The system of bedding ores, which is so common in lead smelters, has not been customary in copper works, where the ores have usually been classified in separate bins from which the components of the smelting mixture were drawn. The bedding system is used at the Copper Queen smelter, all ores being bedded to form a smelting mixture, while fluxes and fuel are kept in separate bins and added to the charges in transit from the ore pits to the furnaces. The ore is measured, but the fuel and fluxes are weighed.

Owing to changes in the ore, causing a reduction in grade of matte from 45 per cent copper to 30 or 35 per cent, and the increase in quantity of fine ore not suited to blast furnace operation, it was necessary to make some changes in the plant. Accordingly, six 18-ft. McDougal roasters and two 91 ft. by 19½ ft. reverberatory furnaces were added. Changes have been made also in the converter practice, substituting basic lined for acid lined converters. The lining of the present converter plant is preliminary only to an entirely new converter plant which will consist of vertical converters, 12 ft. diameter.

The subject of dust losses has been investigated, with the result that the new flues for the calcining and reverberatory furnaces are equipped with wire screens to cause the settling of fine dust. The original experiments to determine the extent of dust loss were made by passing a portion of the furnace gas through a set of closely woven bags. The smoke samples averaged 18,000 lb. each, and the per cent of dust recovered varied from 0.06 per cent to 0.16 per cent, the larger amount corresponding to the larger number of furnaces in operation. The dust contained about 9 per cent copper, 8 per cent lead and 5 per cent zinc. By further experiments it was determined that this dust would settle in flues if the velocity of the gas was reduced to 150 ft. per minute, while by the introduction of wire screens at 3-ft. intervals in the flues, the dust would settle equally well if the velocity were twice as great. The new dust chambers are of novel design, having a steel frame with hollow tile for floor, sides and roof. The latter is suspended from trusses exterior to the chamber. Similar experiments conducted on gas from the converters resulted in the recovery of from 0.015 per cent to 0.035 per cent dust, assaying half as much in copper, but twice as much in lead and zinc as the furnace dust.

As the result of a long series of experiments conducted with a view of devising some suitable way of mixing fine dust with converter slag for subsequent smelting in the blast furnace, the following plan was adopted. The mixing device is a truncated cone set with its axis horizontal. It is mounted on rollers after the manner of a barrel converter, and is revolved at the rate of 5 r.p.m. by a chain and sprocket. Molten converter slag and fine dust are introduced into this revolving cone at its small end; the slag being poured from a ladle and the dust delivered by pipe from a storage tank. The effect is to form a porous, lumpy material which is well suited to blast furnace smelting. The proportions of slag and dust are regulated by one man stationed so that he can see the discharged product which is a guide to the proportions used.

Notes on Copper Smelting in Russia.—The copper smelting works of the Société Minière de Bogoslovsk, Perm, Russia, has been in operation intermittently for the last 150 years, but it is only of late years that modern methods have been employed. A description of present operations is given by Mr. RICHARD DAVEY, in *Bulletin* No. 101, Inst. Min. & Met. The company was one of the first in the Eastern hemisphere to adopt the bessemerizing process, the plant dating back to 1885. The ores consist of chalcocite, chalcocite, pyrrhotite, cuprite, malachite and chrysocolla, with a gangue locally known as augito-garnet, with some calcite and a little free silica.

The principal features of the works are sinter-roasting of fine ore and flue dust in pots, granulation of slag, and the concentration of the gold and silver in the converters. The last operation is performed as follows: Three successive charges of matte are made to a converter, the charge being blown to the slagging point after each addition of matte. After the slag is removed from the third charge, the converter contains white metal assaying about 65 per cent copper. This is then blown until the rods used for punching the tuyeres show signs of metallic copper. The converter is then quickly tilted and the white metal run into a ladle and transferred to another converter to be blown to blister for ingot copper. The copper which remains in the first converter is found to contain practically the whole of the gold and most of the silver contained in the original matte, and this is cast into anodes for electrolytic separation.

The blast furnace makes a matte assaying from 30 per cent to 40 per cent copper, and the slags average 0.28 per cent copper. An analysis of the slag shows, in round numbers, 30 per cent silica, 12 per cent alumina, 34 per cent ferrous oxide, 0.3 per cent sulphur, 13 per cent lime and 1.5 per cent magnesia. The sulphur on the charge is 10 per cent, and the quantity of coke used amounts to 7 per cent of the gross charge. The converter slag which is not required for fluxing purposes is added to the furnace hearth in such a manner that it mixes with the flow of slag and matte from the furnace.

Transformation Point Apparatus

We have just received from the Leeds & Northrup Company a copy of their bulletin describing a new and interesting line of apparatus for the location of the critical points of iron and steel. This bulletin outlines two sets of apparatus, both

placed in immediate contact with it. The material of this metallic body, or sample holder, as it is termed in the bulletin is so chosen that throughout the range wherein lie the critical points of iron and steel, the sample holder has no such changes.

Hence the sample and sample holder will heat at the same rate and remain sensibly equal in temperature until the sample undergoes a change involving the absorption or evolution of

heat. At this time the temperature difference between the sample and sample holder previously approximately zero, undergoes a sudden change. This change, even in low carbon steels, is definite and great enough to be readily seen.

The two main advantages of this method are patent. In the first place, fluctuations in the rate of heating affect both the sample and holder practically simultaneously and are, therefore, of slight effect. It will be remembered that the more commonly applied method of plotting temperature against time can

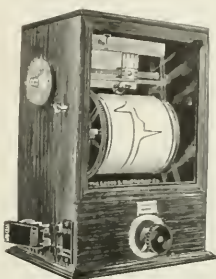


FIG. 2. — TRANSFORMATION POINT INDICATOR

only be used with satisfaction to determine cooling curves on account of the effect of fluctuations in furnace. Then, in the second place, the temperature—temperature difference method has the decided advantage of utilizing a quantity (the heat of transformation) whose actual magnitude is small to cause a large percentage change in a second quantity (the temperature difference) which is being subjected to a series of measurements.

To illustrate this last point, suppose a steel to be of such composition as to liberate heat sufficient to change its temperature 2 deg. F., as in cooling it passes 1435. To note by the time-temperature method this, the reading and plotting arrangements must be capable of showing a change of two parts in 1435. On the other hand by the difference method even though the sample and its holder differed initially by as much as 5 deg., a change of 2 deg. would be a change of two parts in five or about 40 per cent.

We reproduce in Fig. 1 heating and cooling curves taken from a series of steels ranging from pure iron to 0.6 carbon steel. The left-hand curve of each pair is the heating curve and the right-hand the cooling curve.

The steel sample should be about 1 in. long by $\frac{3}{4}$ in. in diameter. Where it is required, the holder may be arranged for flat stock.

The temperature and temperature difference measurements are made with thermocouples. The temperature measuring couple is read with a simple potentiometer and the temperature difference is read with a reflecting galvanometer. In one equipment the Leeds & Northrup potentiometer indicator is used and the readings which are taken practically simultaneously from the potentiometer and from the reflecting galvanometer are afterwards plotted in form of curves.

With the transformation point indicator, an illustration of which is shown in Fig. 2, the observer draws the heating and cooling curve while the process is going on, without taking any readings. In this instrument cross-section paper is wound on a drum which also has wound upon it the slide wire of the temperature measuring potentiometer. The reflecting gal-

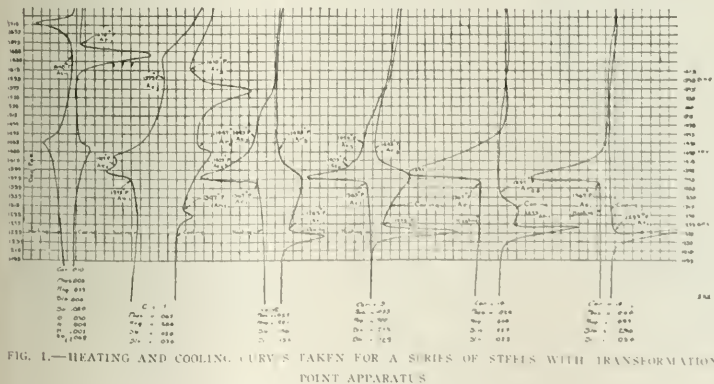


FIG. 1.—HEATING AND COOLING CURVES TAKEN FOR A SERIES OF STEELS WITH TRANSFORMATION POINT APPARATUS

fundamentally the same, but externally very different. We give herewith a reproduction of a series of typical curves taken from this apparatus.

In the curves plotted with this apparatus, the temperature of the sample as it heats and cools is used as one ordinate while the other ordinate is the corresponding temperature difference which exists between the iron or steel and a metallic body

vanometer of a difference couple circuit throws its beam upon a movable glass scale which also carries the pen which draws the record. As this galvanometer deflects, the glass scale is moved to keep up with the deflection, thus drawing the pen across the paper.

The galvanometer of the potentiometer circuit is also of the reflecting type and throws its beam upon a fixed scale. This circuit is held at a balance by turning the potentiometer drum thus drawing the line along the length of the paper. The resultant is of form similar to the curves shown herewith. Both motions are obtained by the rotation of shafts. Thus, at the completion of the test, the observer removes from the drum a finished transformation curve of the steel under test; he has drawn this curve by the simple process of turning two shafts in such a way as to maintain two spots of light on their respective scales. The manufacturers, the Leeds & Northrup Company, state that they have examined many varieties of carbon and alloy steels and that they have yet to find a steel which will not yield definite results on this apparatus.

Recording Differential Pressure Gauges

A comprehensive new line of recording differential pressure gauges has been developed by the Bristol company, of Waterbury, Conn. Some of these recorders have been in successful service continuously since the preliminary models were first sent out in 1908, and the design and construction of the line of these instruments now being placed on the market is based on results obtained in actual service during the last four years. These recording differential pressure gauges are designed for use in connection with venturi meters, Pitot tubes, orifices, combinations of orifices and Pitot tubes, etc., and thereby to record velocities and volumes of air, gas, steam, water and other liquids flowing through mains and pipes. These recorders may also be used to advantage for recording differences and variations of liquid level in steam boilers, pressure tanks, filter beds, process kettles, etc.

Important patents dated September 12, 1911, and September 17, 1912, have been issued to Prof. Wm. H. Bristol, president and founder of the Bristol company, covering novel features resulting from his pioneer work in developing differential recorders in addition to the other numerous lines of recorders which he has originated during the twenty four years which have elapsed since his first patent was allowed on a recording pressure gauge.

The fundamental principle employed in the construction of this differential pressure gauge is that one pressure is applied to the inside of the operating tube while the other is applied to the outside of the same pressure tube within a closed casing. In order to record the movement of the pressure tube it becomes necessary to transmit its motion to the outside of the pressure tube casing



FIG. 1.—RECORDING DIFFERENTIAL PRESSURE GAUGE.

As the differential pressure to be recorded is usually small as compared with the static pressure, the operative force is correspondingly small, and it is quite evident that it will be impractical to use a stuffing-box around a shaft passing through a pressure casing on account of the friction which would be produced. To avoid the use of a stuffing-box a unique frictionless

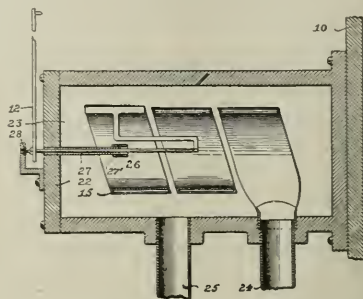


FIG. 2.—INTERIOR OF HELICAL PRESSURE TUBE TYPE OF RECORDING DIFFERENTIAL PRESSURE GAUGE

sealing device is employed as described in detail below.

Fig. 1 shows the exterior of one type of these recorders. Figs. 2 and 3 show part of the interior construction of the spring tube type differential recorder. In Fig. 2 pressure tube 15 is of the hollow helical type used with great success for many years in Bristol recording pressure and vacuum gauges. One of the pressures, the difference of which is to be recorded, is applied to the interior of this tube through the pipe 24 and the other pressure is applied to the exterior of this tube through the pipe 25. It will be noted that the pressure tube 15 is entirely enclosed in the pressure-tight casing 22. The movement of the pressure tube will be in proportion to the difference between these two pressures, and this motion is transmitted to the recording pen arm 12 (outside of the pressure casing) by means of small shaft 26 through the long tubular sleeve 27. The capillary action of the oil or liquid between the sleeve and shaft makes this patented joint both frictionless and pressure-tight.

In Fig. 3, 22 is the pressure-tight casing enclosing the diaphragm pressure tube 14, and in a similar way one pressure communicates with the interior and the other pressure with the exterior of this tube, and its motion is transmitted by means of the rotating shaft through the sleeve 21 to the recording pen arm 12. The length of this sleeve is many times the diameter of the shaft passing through it, differentiating it from an ordinary bearing.

This patented device permits of the recording of extremely small differences between the pressure existing respectively inside and outside of the pressure tube. It has been found that the simple frictionless sealing sleeve through which the pen arm shaft passes does not produce appreciable resistance to the rotation of the shaft, and at the same time capillary attraction and adhesion prevent leakage of even high pressures from the pressure casing.

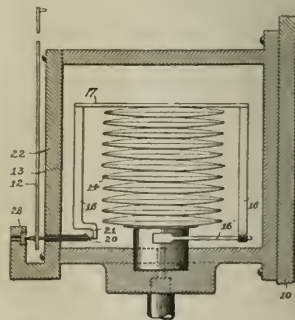


FIG. 3.—INTERIOR OF DIAPHRAGM PRESSURE TUBE TYPE OF RECORDING DIFFERENTIAL PRESSURE GAUGE

In Fig. 4 diaphragm tube 8 is shown in the pressure chamber of the casing 11 directly connected to one end of the rotating shaft passing through the patented pressure seal, and having its other end directly connected to the recording pen arm 9.

This diagram also shows a set of interlocking valves 17, 18 and 23, which constitute a device for adapting recording differential pressure gauges to practical operating conditions. Cross-valve 22 is shown in an open position connecting the two pressure pipes. Valves 17 and 18 in the pressure pipes are shown closed. Both of these valves 17 and 18 can be opened allowing the static pressure from either pipe to be applied simultaneously to the inside and the outside of the pressure tube of actuating mechanism. The interlocking member 21 can then be turned through an angle of 90 deg., thus making it possible to close valve 22 which completes the connections so that the instrument will record the difference of the two pressures.

Fig. 5 shows a patented safety device consisting of a U-shaped tube 30, partly filled with a suitable liquid such as mercury or water, this tube having enlargements 33 and 34, each having sufficient volume to accommodate the quantity of liquid contained in the U-shaped tube. The length of the U tube varies with the range of the gauge so that the greatest possible head of the liquid contained in the U tube corresponds with the total range of differential pressure that the gauge is designed to record. Should the full static pressure by accident be admitted to either side of the differential gauge, the liquid contained in the safety U tube would instantly be forced up into one of the enlargements, thus allowing the static pressure to be applied simultaneously to both inside and outside of the pressure tube and protecting it from being destroyed. There is a great need for these instruments in many different industries and the field of usefulness for these is rapidly growing. Further information may be obtained from the manufacturers.

Centrifugal Separation

By Leslie Griscom

The use of centrifugal apparatus in removing liquid from solids and in separating liquids of different densities has a continually widening application. In the textile and sugar industries they have been in very general use for many years. Of more recent application has been the milk-and-cream separator, the medical centrifuge, and the Babcock milk tester. To illustrate the wide scope of the application of the centrifugal machine we will note a few of the materials on which they are used: General laundry work, yarns, piece goods (in rolls), felt boots, rugs, rubber, sugar, hides, curled hair, various kinds of salts, milk, printers' ink, and in bacteriological work, for concentrating germs, crystals, etc., for observation under the microscope.

A form of centrifugal has also been used for extracting honey from the comb, leaving the comb intact, so that it can be filled a number of times by the bees. As a pound of beeswax is said to take several times as much of the energy of the bees as a pound of honey, the honey extractor has proved a great benefit to the bee industry. The amount of moisture left in the material by a centrifugal varies greatly with different types of materials. With certain kinds of chemical salts it will be as low as 2 per cent or 3 per cent, while with fabrics and hair it may reach 50 per cent to 100 per cent of the weight of the dried material, this residual moisture being held by capillary attraction. Possibly few realize the force developed in a centrifugal machine. If we take as an example a 40-in. centrifugal running at 1000 revolutions per minute, a 1 lb. weight at the periphery will exert a centrifugal force of

$$F = 0.000341 WRN^2 = 0.000341 \times 1 \frac{1}{2} \times 1,000,000 = 508 \text{ lb.}$$

in which

W = weight in pounds

R = radius in feet

N² = the square of the number of revolutions.

Taking the same example, if we have a wall of material in the basket 5 in. in thickness the centrifugal head will equal

$$\frac{V^2}{2g} - \frac{V_1^2}{2g} = \left(\frac{40\pi \times 1000}{60 \times 12} \right)^2 - \left(\frac{30\pi \times 1000}{60 \times 12} \right)^2 = 207.2 \text{ ft.}$$

For water this equals a pressure of $207.2 \times 0.433 = 89.7 \text{ lb. per sq. in. at the periphery.}$

The strain set up in the shell of the empty basket (not allowing for thickness of shell) under the above conditions would be $0.0000204 V^2 = 0.0000204 \left(\frac{40\pi \times 1000}{12} \right)^2 = 3224 \text{ lb. per sq. in. cross section of metal.}$

It is somewhat of a paradox that while a much larger proportion of the liquid can often be removed from chemicals than from fibrous materials the centrifugal machine has had a very general application in the textile trade, while its use for drying chemicals, coal, etc., appears to have been comparatively undeveloped. For this latter purpose a strong, well-designed and well-built machine is necessary to take care of the heavy loads. Further, it must be of a self-balanced type, with bearings well protected, and where acid materials are operated upon the basket, etc., must be constructed of material that will resist corrosion. The machine should be constructed to discharge the material through the bottom of the basket to facilitate emptying. It should preferably be of an under-driven type to give free access to the top of basket. A direct connected motor makes a very convenient drive, doing away with belts and shafting. With the impregnated windings now furnished the motors are able to withstand the corrosive vapors to a remarkable extent. Where there is much explosive vapor a steam engine can be used.

A centrifugal of this type, built by Schaum & Uhlinger, Inc., Second Street and Glenwood Avenue, Philadelphia, has been in successful operation for some years, in the drying of ammonium sulphate and for other purposes.

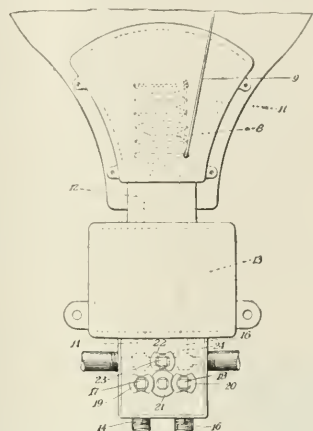


FIG. 4.—CONNECTIONS FOR DIAPHRAGM TYPE RECORDING DIFFERENTIAL PRESSURE GAUGE

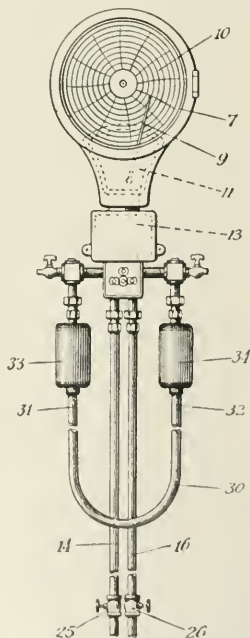


FIG. 5.—PATENT SAFETY DEVICE FOR USE WITH RECORDING DIFFERENTIAL PRESSURE GAUGE

These machines are substantially built of best material, and are being made to jigs and templates so the parts are interchangeable. The axle steel shafts are ground to gauge. Ample ball thrust bearings are provided which run in a bath of oil. Baskets are furnished either of steel or of material to resist corrosion. Curbs are made of heavy steel plates and lined when necessary to prevent corrosion. These machines are furnished for either belt drive or direct connected to motor through a flexible coupling. They also build a steam engine specially adapted to the operation of machines of this type.

Being under-driven the top of the machine is unobstructed, facilitating the charging and emptying of the basket. On account of the method of construction, and the heavy curb and base, these machines are comparatively free from causing vibrations in the buildings. Where desired, baskets having sloping

which have made possible the recovery of such waste products as wool grease at an actual profit.

We reserve a more detailed description of the equipment of the new Lummus industrial laboratory for a future issue.

"Semi-Producer" Furnaces

The recent rapid increase in the price of oil as fuel has brought various other processes prominently to the front, especially the use of powdered coal and the use of producer gas.

The W. S. Rockwell Company of New York City, has worked out an interesting solution of some difficulties experienced in the use of producer gas. Hot producer gas for average heavy work usually implies large, expensive flues, which soon become tar and soot clogged, and, in any case, entail the loss of a considerable portion of the sensible heat of the gas and take up a large amount of room. Cold producer gas means that moderate temperatures only are obtainable unless regeneration or elaborate recuperation is resorted to, since all the sensible heat of the gas has been lost, a portion of the volatile gases condensed to tar and there remains only a lean gas diluted with a large and worse than useless volume of nitrogen.

The solution of the W. S. Rockwell Company is very simple; it is to combine the furnace and gas producer; thus making a small, simple, self-contained, easily handled, efficient unit, covering within itself the entire heating operation from coal pile to heated product—having all the advantages of a hot producer gas furnace without the disadvantages of a separate producer and expensive connecting flues.

In this type of furnace the generation and distribution of the hot gas to the material in the furnace is accomplished without the loss of any sensible heat, without the condensation and

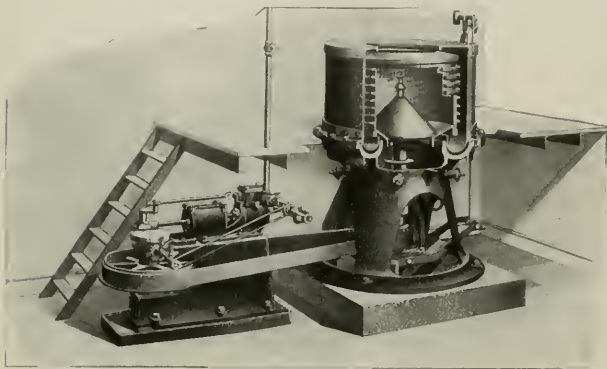
consequent loss of any of the tarry gases and with little loss of heat by radiation.

On these principles they have developed a variety of furnaces covering many classes of work. At all stages of the development the fact that good heating conditions must not be sacrificed in order to use a cheap fuel was borne in mind and the result has been a line of furnaces which involve such essential principles of good and economical heating as under-firing, recuperation, down-draft and absolute control of temperature.

While it has been impossible, of course, to cover all sizes and types of furnaces by this combination of furnace and gas producer, yet the fields which have so far been developed for this combination by the W. S. Rockwell Company, cover annealing, hardening, tempering, carbonizing, enameling, and other miscellaneous operations.

Metals and Alloys Free from Carbon Produced by the Thermit Process is the title of a very attractive and well written pamphlet of 30 pages, issued by the Goldschmidt Thermit Company of New York City. It contains concise technical information on the properties and uses of the following metals and alloys: ferrotitanium, chromium, ferrochromium, manganese and manganese alloys, ferromolybdenum, and vanadium.

Titanium in Steel—a Money Saver is the title of a very instructive and profusely illustrated pamphlet of 48 pages, just issued by the Titanium Alloy Manufacturing Company of Niagara Falls, N. Y., on the uses and benefits of ferrocarbon-titanium as a dioxidizer and scavenger of steel for various purposes, including, of course, rail steel. Particularly attractive and instructive are a series of microphotographs and a table giving the results of mechanical tests of steel treated with titanium. The pamphlet contains also the specifications which have been carefully worked out by the Titanium Alloy Mfg. Company to govern the addition of ferrocarbon-titanium for all kinds of steels.



CENTRIFUGAL MACHINE

bottoms are provided to facilitate emptying. They also provide a removable top to curb to facilitate removing basket, and provide hand holes in curb to render gutter accessible where the nature of the material handled requires frequent cleansing.

A New Industrial Laboratory and Demonstration Plant

The firm of Mr. Walter E. Lummus, of Boston, Mass., well known by his work in distillation engineering, has erected quite an elaborate industrial laboratory or rather demonstration plant, equipped to operate all kinds of processes involving distillation and evaporation on a manufacturing scale and an outline of some of the interesting work which is being carried out now in this laboratory is worth giving.

They are now building and testing apparatus for coating and drying fabrics with plastic compounds and recovering the solvents; also a plant for producing turpentine and other products from wood; further apparatus for extracting rubber from rubber-producing vines and shrubs.

There is further in course of construction a large continuous still for concentrating wood alcohol with recuperative heater, with a capacity of 10,000 gallons a day.

Nearly all the products of this firm are large apparatus for operation on a commercial scale, but they have under construction several interesting experimental apparatus. Among them is a new outfit for demonstrating fractional distillation, being built for Columbia University, further sundry experimental apparatus to be used, chiefly in connection with wood chemical and solvent extraction problems. Among these experimental apparatus is an extracting and distilling outfit which will be used by an important tropical trading company to de-resinate and concentrate sample lots of rubber.

A recent interesting achievement of Mr. Lummus is the development of remarkable economies in the concentration of extracted products by stage diffusion or extraction without heat

Personal

Mr. E. A. Austin, manager of the Guggenheim interests in the Iditarod district, Alaska, was a recent visitor at Stanford University, California, from which institution he graduated in 1906.

Mr. Herman Bacharach, until recently located at 722 Lewis Block, Pittsburgh, Pa., has moved into larger quarters at 1009 Hartje Building. He has enlarged the number of his specialties in scientific and industrial instruments, which now include smoke and dust recorders, specific gravity meters for gas and liquid, carbon dioxide recorders and engine indicators with automatic efficiency reckoner, etc.

Mr. Charles W. Badgley has resigned his position as chemist in the research laboratory of the American Smelting & Refining Company, at Perth Amboy, N. J., and has accepted the position of chief chemist for the El Paso Smelting Works, El Paso, Texas.

Mr. George H. Benjamin, consulting engineer, expert in patent causes, and solicitor of patents, announces the removal of his offices on April 15, 1913, from 45 Broadway to 66 Broadway (Manhattan Life Building) New York City.

Mr. D. W. Buckby has been appointed superintendent of the Stewart mill at Wallace, Idaho. For some time past Mr. Buckby has been engaged in experimental work for the Federal Mining & Smelting Company, of Idaho.

Mr. J. Parke Channing, consulting engineer for the Miami Copper Company, recently inspected the company's property at Miami, Arizona.

Mr. E. C. Engelhardt has left Garfield, Utah, to accept the position of smelter superintendent with the Idaho Smelting & Mining Company, at Clayton, Custer County, Idaho.

Mr. Le Roy Gordon has been appointed manager of the New York office of the Nelson Valve Company and Yarnall-Waring Company, with offices at 30 Church Street, New York City. Mr. Irving N. Beeler has been appointed district sales manager for central New York with offices in the Bastable Building, Syracuse, N. Y.

Mr. Franklin Guitermann, technical director for the American Smelting & Refining Company, with headquarters in New York City, was in Denver in March on a periodical trip of inspection.

Mr. A. B. W. Hodges, who left British Columbia three years ago to assume general management of the Cerro de Pasco company in Peru, has finished his contract and will return to the United States.

Mr. D. C. Jackling, general manager for several copper companies in Utah, Arizona and New Mexico, will change his residence from Salt Lake City, Utah, to San Francisco, Cal.

Mr. J. G. Kirchen was recently re-elected a director of the Tonopah Merger Company, and will continue as general manager of the company's operations at Tonopah, Nev.

Mr. W. A. Kunkle has resigned as superintendent of the Ajax mill at Victor, Colo., and will engage in consulting practice in Denver, Colo.

Mr. William D. Leonard, superintendent for the Garfield Smelting Company, Garfield, Utah, was a recent visitor in Denver.

Mr. Edgar Rickard, manager of the *Mining Magazine*, London, has been making an extended visit in the United States.

Mr. Haller R. Robbins has been appointed assistant professor of metallurgy at the State College of Washington, Pullman, Washington, succeeding the late R. E. Sampson.

Mr. J. F. Thorn has resigned as general superintendent for the Goldfield Consolidated Mines Company, Goldfield, Nev., and will be succeeded by Mr. Albert Burch.

Mr. William Wraith, general manager of the International Smelting & Refining Company, Salt Lake City, recently inspected the new electrolytic lead refinery built by his company at East Chicago, Indiana.

Obituary

Dr. Frederic Schniewind, the widely known pioneer of the by-product coke oven industry in the United States, died on March 12 in New York City. He was born in Germany 52 years ago and a graduate of the University of Heidelberg, where he received the degree of Ph.D. He came to this country in 1889 and by introducing the Otto-Hoffman coke oven he became the founder of the by-product coke oven industry in this country. At the time of his death he was president of the German-American Coke & Gas Company and its subsidiaries, the United Coke & Gas Company and the Coke & Gas Construction Company.

Mr. George E. Gunn, who has been a prominent figure in Western mining and metallurgical circles for the past twenty years, died suddenly of heart disease in his apartments in Hotel Utah, Salt Lake City, Utah, on March 11, 1913. Mr. Gunn was fifty years old at the time of his death. He had been prominently connected with the development of Western porphyry copper mines, having had his attention directed to their possibilities while engaged as field engineer for the American Smelting & Refining Company. He also acted as general manager for the company which developed the large lode copper mines and erected a modern custom copper smelter in the Mason Valley district, Nevada. He was associated with Mr. William B. Thomson in the Gunn-Thomson company which originally developed the Inspiration copper property in Arizona, later sold to the Amalgamated interests. The same company is owner of the Magna copper mines in Arizona. Mr. Gunn was a wealthy bachelor, and member of several clubs in Salt Lake City, on the Western coast, and in the East.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

694,349, March 4, 1902, Henry R. Cassel, of New York, N. Y. Relates to a process for recovering gold from refractory ores, such as sulphides, tellurides, etc. Such ores require roasting to set the gold free; by the present process, it is stated that roasting is avoided. The ore is finely pulverized and mixed with water containing about 3 per cent of salt to form a pulp. This pulp is agitated in an open tank (described in patent 694,350), having a lining of carbon connected as one electrode and carbon-coated stirrers connected as the other; the polarity of the electrodes being reversed at will. The current forms hypochlorites and oxychlorides, which oxidize the sulphur, tellurium, etc., setting free the gold. The gold and oxides of alkaline earth metals are precipitated upon the negative electrode as a black powder, from which it is scoured off by the pulp. The gold precipitate is dissolved by cyanide of potassium, which is finally added, and stirring continued; the gold is recovered from the cyanide solution by the usual method.

694,350, March 4, 1902, Henry R. Cassel, of New York, N. Y. Relates to apparatus for carrying out the process in patent 694,349. Within a tank suitably coated with cement, to prevent leakage, and lined with carbon blocks electrically connected as an electrode, there rotates an insulated metal shaft carrying radially projecting stirring-blades of metal and covered with carbon plates. The shaft and stirrers are electrically connected as the other electrode, and both connected to a suitable source of alternating current; or preferably to a continuous current which is periodically reversed. In operation, the ore, mixed with water containing enough salt to render it conductive, is charged into the vat while the stirrers are rotating. After the gold is set free from the ore, a solvent such as potassium cyanide is added to dissolve it, from which it may be recovered as usual.

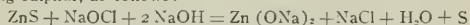
698,202, April 22, 1902, Edward D. Kendall, of New York, N. Y.

Relates to a process for separating metals from solutions, such as gold from cyanide. Within a suitable tank containing a porous cup is a mass of carbon presenting a large surface, and connected electrically to a source of current at about 15 volts. Within the porous cup is preferably a solution of caustic soda or potash, and a carbon plate connected as the other electrode. The mass of carbon is first connected as cathode and a gold-bearing cyanide solution allowed to flow continuously there-through, the gold being electrolytically deposited upon the carbon. The cyanogen liberated at the anode is absorbed by the caustic soda. When a considerable quantity of gold is deposited, the carbon plate is removed from the porous cup and a silvered copper plate inserted: the cyanide solution in contact with the mass of carbon is now replaced by a stronger cyanide solution and the electric current reversed, making the mass of carbon the anode and the silvered copper the cathode. The deposited gold, etc., is dissolved from the carbon and redeposited upon the silvered copper within the inner porous cup, from which it may be stripped from time to time. After the gold has been removed from the carbon, the strong cyanide solution is drawn off, the silvered copper plate replaced by the carbon plate, the electric current reversed, and the process of treating the gold-bearing cyanide solution repeated.

699,964, May 13, 1902, Frederic H. Long, of Chicago, Ill., assignor to Ross J. Beatty, trustee, of Muncie, Ind.

Relates to an improvement on his earlier patent 650,646. In operating the apparatus described in that patent it was discovered that gases were liberated in the cathode compartment and collected under the diaphragm, thereby increasing the electrical resistance. In the present patent a vent-pipe projects into the cathode compartment to carry off the gases, and also provides means to condense and collect any vapors of acid, etc., entrained by the gases.

700,563, May 20, 1902, Samuel S. Sadtler, of Philadelphia, Pa. Relates to the extraction of zinc from ores, etc., and from zinc scrap. The ore is pulverized to about 200 mesh and treated with sodium hydrate and hypochlorite of soda in a suitable tank; this dissolves the ore forming zincate of soda and separating sulphur, as follows:



The solution is allowed to settle and the clear liquid run into the cathode compartment of an electrolytic cell, the zinc being deposited upon a suitable cathode. The resulting sodium hydroxide is run into the anode compartment where it may be converted to the hypochlorite by the addition of a suitable quantity of sodium chloride and then used to treat a fresh quantity of zinc sulphide. Zinc scrap, tin scrap, zinc crusts, etc., may also be treated by this process.

700,941, May 27, 1902, Nathaniel Shepard Keith, of Arlington, N. J.

Relates to a process for electrodepositing copper from its solutions, preferably the sulphate. The electrodepositing cells are connected in series both electrically and hydraulically; the electrodes are so proportioned that there is a higher current density per square foot of electrode in the cells at the beginning of the series, where the electrolyte enters than at the end where it leaves. The current density decreases as the copper content decreases, thereby producing a uniform reguline deposit of copper in each cell. The exhausted electrolyte is used to treat a fresh supply of ore and produce more electrolyte.

702,153, June 10, 1902, Jarig Philippus van der Ploeg, of the Hague, Netherlands.

Relates to the extraction of antimony from its ores, etc., and consists in mixing the finely powdered antimony ore with a suitable quantity of quicklime and a monosulphide or polysulphide of calcium or magnesium according to the quantity of antimony present. When water is subsequently added, soluble double sulphides of antimony and calcium or magnesium are formed, whereby all the antimony may be removed from the ore. The solution containing the antimony sulphide is electrolyzed, or may be treated with an acid, such as hydrochloric acid, to precipitate the antimony sulphide.

Book Reviews

The South African Mining Directory. A monthly handbook. Published by the South African Mining Directory, Ltd., 119 Exploration Bldg., Johannesburg, Transvaal, South Africa. 208 pages, 5¼ x 8; price, £2 10s.

This is a new publication, issued for the first time in November, 1912. Among mining directories it should hold a premier position for accuracy by reason of its frequent publication.

The customary objection to all such directories is that they soon become out of date on account of the changes in the personnel of the business and technical staff of the companies. As this directory is to be issued monthly it will be free from that objection. The first number is quite complete, giving data on the organization of the companies, and a roster of the principal officials. All the mining industries in South Africa and Rhodesia are covered, comprising gold, copper, tin, coal and diamond mines. Statistics on the annual purchases of South African mines, and an alphabetical index of mine officials, complete the volume.

* * *

Refractories and Furnaces. By F. T. Havard, E. M. 6 x 9 in. (15 x 23 cm.), 356 pages, 123 illustrations; price \$4.00. New York: McGraw-Hill Book Company.

The author was in practical work as superintendent of smelting furnaces, and some notes on furnace linings made at that time served as the nucleus of this book. By collecting and adding to these the scattered data on refractory materials, heat conductivity of materials, and heat losses by radiation and convection, the work has been completed.

It discusses refractories primarily and furnace construction incidentally.

The treatment is highly scientific yet intensely practical throughout, and it is just the kind of book which was needed, both for the university and the works. We advise everyone having anything to do, or to work out, in this line, to get the book, and to get it quickly, for while a better book may be written in the future, there is no work at present in print which covers the subject so satisfactorily.

* * *

Electroplating. A Treatise on the Electrodeposition of Metals, including Metal-coloring and Bronzing. By Wm. R. Barclay and Cecil H. Hainsworth. 12mo. (12 x 18 cm.), 399 pages, 62 illustrations. Price \$2.10. London: Edward Arnold. New York: Longmans, Green & Company.

This is primarily a handbook for the practical electroplater, to help him drop rule-of-thumb and to work scientifically. The book is an amplification of lectures delivered at the University of Sheffield by Mr. Barclay. While giving data in ounces Troy and avoirdupois, yet the book lays its main emphasis on using the metric system, and thus tries to lead the electroplaters to the simple, modern decimal system.

The writers not only describe present electroplating methods, but they look ahead and foresee advances in the art. For example, speaking of lead, tin, and antimony, they say that "at the present moment, and writing from an electroplating point of view only, the three metals dealt with in this chapter * * * are of comparatively little interest for the practical worker. It is quite possible and even probable, however, that the immediate future will witness an increase of the commercial possibilities of electroplating with these metals, and some little space should therefore be devoted to an outline of the principal methods of their deposition." Very well said; a book written in such a spirit cannot fail to be valuable and inspiring.

The April meeting of the American Electrochemical Society will show how accurate was the foresight, at least as regards "lead." The field of the electroplater is rapidly widening, as the point of view and education of the electroplater are broadening.

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The Coming Denver Meeting of the American Electrochemical Society

The next meeting of the American Electrochemical Society will be held in September in Denver, Col. During the first eleven years of its life the American Electrochemical Society has become truly international in membership, activity and prestige. The coming years, with the Denver meeting this fall, the proposed meeting in Birmingham, Ala., in the fall of 1914, and a San Francisco meeting in 1915, will give an opportunity to broaden and strengthen the national scope of the society.

In view of such a nation-wide activity the question may be asked: What can electrochemistry do for the country? With respect to the Denver meeting the question becomes more specific: What can electrochemistry do for Western metallurgy? It is natural that the discussion of just this question will mark the keynote of the Denver meeting, a symposium of papers being arranged discussing broadly the potentialities of the applications of electrochemical methods and principles of physical chemistry in Western metallurgy.

If the mountain does not come to the prophet, the prophet has to go to the mountain, and no particular power of prophecy is needed to say that the meeting will be interesting. Electrochemists are born good mixers. Electrochemistry is a borderland of sciences and a borderland of industries. In our age of specialization the scientist and the engineer each uses his own jargon, and so does the physicist, the chemist, the metallurgist and all the otherists of man-made professional distinction, and in this one respect at least modern methods of progress often resemble now the building of the tower of Babel that those who work on the promotion of civilization, each in his own way, no longer understand each other's language.

To remove such man-made gulfs or at least to bridge them firmly, this has been one and perhaps the most glorious function of the American Electrochemical Society. The society has brought together engineers and scientists, to look at each other no longer with distrust, but with mutual respect. Those of its meetings which were devoted especially to particular human mixing reactions, like the Pittsburgh meeting and the recent Atlantic City meeting, have been eminently successful. The Denver meeting should become another milestone in the career of this society.

Electrochemistry and the Electroplating Industry

The symposium on electrodeposition of metals at the Atlantic City meeting of the American Electrochemical Society will add considerably to that Society's already well-deserved reputation as a meeting ground on which men interested respectively in the practical and theoretical aspects of industrial science come together on a basis of

mutual assistance and respect. Such mutual attitude is unfortunately not too commonly met with in the field of applied chemistry, possibly owing in part to the patronizing attitude which too many well-meaning official or academic chemists adopt towards those engaged in industries of which electroplating is typical, in which full advantage is decidedly *not* taken of current scientific knowledge.

The imaginative common sense of the retiring President of the American Electrochemical Society, Professor W. Lash Miller, led him to try the experiment of bringing electrochemists and electroplaters into personal contact and mixing them vigorously in the hope that a solution—or at least an emulsion—might result. If the immediate outcome of the experiment was to demonstrate the surprisingly high values of the surface tensions involved, this merely indicated the desirability of continuing experiments with the view of reducing these. The appointment of a committee to maintain coöperation between the Electrochemical Society and the Electroplaters' Association was the outcome with (appropriately enough) that master of emulsification, Professor Wilder D. Bancroft, at its head.

This seems to us a decided step in the right direction. It is often too readily assumed, when the application of current scientific knowledge is absent from such an industry, that this is because the industrial men do not want it, and that such application is an easy matter. Both these ideas received a violent refutation at the Atlantic City symposium. The interest and enthusiasm of the attending electroplaters, and their obvious desire to absorb anything electrochemistry could give them were remarkable; if they don't get much benefit from electrochemical science it isn't at any rate because they don't want to.

Even more notable was the obvious gulf between the two parties to the discussion. Not only did they not, as both admitted, "speak the same language"; it was hardly possible for them to come into contact on any point. This gulf cannot be airily disposed of by telling the electroplaters they are on the wrong side; it has to be bridged, and to this end the best joint efforts of both parties on the committee may well be directed. The strongest promise of success in this aim is offered by the spirit of coöperation in which the two organizations have approached the matter.

The first step in the direction of improvement is, however, one which those electroplaters who have not yet taken it can best learn from those who have, viz., the rigid application of method and the systematization of all procedures. We are acquainted with electroplating plants in which methodical supervision and close record-keeping have led by themselves to extraordinary improvements in working. It need hardly be pointed out that this is nothing more or less than the application of scientific method, and that without it an electroplater will look in vain for assistance from scientific knowledge.

Moving Pictures for Industrial Purposes

The photograph has long been recognized as a valuable adjunct to oral and written description. In many cases the simple projection of a lantern slide, or the printed reproduction of a photograph, conveys quickly and definitely an idea which might require a long, involved and less

satisfactory verbal explanation. The general adoption of this method in educational and commercial work proves its value. More recently the development of the moving picture has called attention to possibilities in this line which have only begun to be realized. In the field of metallurgy there is an opportunity that should appeal to both educational and commercial institutions, to use moving pictures for more clearly presenting and explaining the operation of many machines which work continuously or in a cycle of intermittent steps.

It is customary in engineering schools for the instructors to illustrate some lectures by means of stereopticon views. The method is excellent, but might be vastly improved by the use of moving pictures. Consider the number of metallurgical operations that could well be illustrated by the latter method: Crushing and grinding, classification of sand and slime, sizing by various forms of screens and the screenless sizer, sinter roasting, concentrating, vacuum filtration by the movable leaf or revolving drum types of filters, handling and conveying materials, charging and discharging blast and retort furnaces by hand or machinery, bessemer converting, dressing amalgamating plates, and so on. The operation of an entire metallurgical plant could be presented on one film, taking the different steps in the process in proper sequence, giving in effect a trip of inspection through the plant.

For the manufacturer of metallurgical machinery, the moving picture may be an adjunct in advertising, promoting and selling. Recently one manufacturer resorted to this method with good effect. His machine was new and little known or understood; it embodied some principles that were novel, and its operation was not easily described; catalogs were inefficient in conveying an idea of its full value. A few hundred feet of film, however, showing the machine in operation, demonstrated quickly and convincingly the nature and scope of the machine and its method of operation. In such cases, simple photographs fail to convey the idea of operation, and leave too much to the imagination. The moving picture, on the other hand, gives the effect of an actual demonstration, and is far more effective than photographs or verbal description. In view of the widespread use of machines for projecting moving pictures, films can be shipped to any important point for demonstration.

Cleaning Filter Cloths.

The prevalent method of treating filter fabrics for the removal of incrustations and deposits of lime and other earthy salts, is with a solution of hydrochloric acid. The method has its disadvantages and is not wholly in favor; filter cloths suffer some deterioration under the action of the acid, and additional tanks may be required. In our issue for last month we noted a patented process for cleaning filter fabrics and screens by the use of a sand-blast. By this method the outer surface of the dried fabric is subjected to the action of the abradant. If the size and hardness of the material used in the blast is carefully selected, and the pressure of the blast properly regulated, it is claimed that the fabric suffers less deterioration than when treated with acid, and that the removal of the incrustation is very effective.

The use of the sand-blast for this purpose is not entirely new, especially in other countries. We believe the idea was first applied in Kalgoorlie, Western Australia, in connection with filter pressing. A great deal of trouble was experienced there with calacorous deposits from the water, which was heavily loaded with salts. Incrustations formed on the plate-and-frame presses to such an extent that after breaking a press it was almost impossible to make tight joints between the members when setting the press up for another run. The deposit resisted removal by scraping or other means, and the sand-blast was called into use with good effect.

From Western Australia the practice spread into New Zealand, where it was used on filter fabrics with more or less success and satisfaction. From New Zealand the method was taken to South Africa. In the latter country the practice is in vogue at one or two mills with fair success. The sand is fed into an open conical hopper which forms a reservoir. A pipe for compressed air extends downward to a point near the apex of the cone, and from the latter point a flexible hose and nozzle conduct the blast to the filter fabric. In the United States the process is in use at the Golden Cycle mill.

Still another proposal for cleaning filters is the use of steam under about ten pounds pressure, assisted by a subsequent wash of hot water. This method has been very effective at one of the cyanide mills in the Black Hills of South Dakota, having been introduced there by Mr. F. C. Bowman. The filter is of the drum type. When the cloth needs cleaning, steam is forced through successive sections of the filter by means of the pipe ordinarily used to conduct compressed air to dislodge the cake. The effect of the steam is to immediately loosen and disintegrate the deposit of lime, after which it can easily be washed out by hot water, leaving the meshes of the cloth clean and free.

Rolled Steel and Cast Iron

It is interesting to note that common knowledge and such statistics as can be cited unite in testimony that the iron casting is decadent as compared with its vigorous rival in the ferrous material field, rolled steel. Continually one hears of new uses to which steel is put, and practically all these uses involve rolled steel. Metal furniture employs rolled steel, the automobile carries much rolled steel, a little forged steel and an occasional steel casting, but a negligible amount of cast iron. Sheet piling, concrete reinforcing bars, relatively new products, are of rolled steel. The cast-iron column is decadent in building construction, while the skeleton structure of rolled steel thrives amazingly. The cast-iron carwheel is losing ground to steel, though in this case the steel product is either a real forging or is rolled in a manner which does not place it within the category of the ordinary rolled-steel product. In the majority of forms of hardware rolled steel has long been replacing cast iron.

A few references like these are amply suggestive of the rapid march of rolled steel, and of the relative, if not absolute, decadence of the iron casting. Inasmuch as the total production of ferrous material, measured by the production of pig iron, which is a fair index despite the use of a moderate amount of old material, has been increasing

rapidly, doubling in a general way every decade, it requires close scrutiny to determine whether the decadence of the iron casting is absolute or merely relative to rolled steel. There have never been statistics of the production of iron castings, but the production of foundry iron should furnish a valuable clue, for the consumption of cast-iron scrap in the foundry cannot have increased very rapidly. The full statistics of pig-iron production by grades in 1912 are not yet available, but there are statistics which segregate Bessemer, basic and manganese bearing metals from other pig iron, and these show that the production of foundry, malleable, forge and miscellaneous grades was slightly less than 6,500,000 gross tons in 1912, whereas in both 1910 and 1909, the next best years in total pig-iron production, the output of these grades was about 6,750,000 tons. The total production, however, has been: 1909—25,795,471 tons; 1910—27,303,567 tons; 1912—29,727,137 tons. Here was a large increase in total pig iron, and a decrease in the foundry grades.

The statistics of the market seem to bear the same testimony. In the general revival in the iron and steel industry in 1912, rolled steel products advanced until they reached a certain level, early in October. Since then the level has been maintained, there being practically negligible advances and declines. Foundry pig iron, on the other hand, after reaching a high point in December, has steadily declined in price, and in the past month the declines have been particularly rapid. With quoted prices unchanged, the steel mills have month by month been realizing better prices on their shipments, as old and lower-priced contracts were worked out, while at the same time pig iron, which has certainly been subject to no adverse influence from the steel trade, has been declining.

It is one of the adages in the iron and steel trade that in case of a general improvement in the industry the foundry trade finds its improvement about six months later than the steel trade, lagging behind it by about this distance. Before this time, therefore, there should have been a very distinct improvement in the foundry trade, which one would naturally expect to find in its reflection in the market prices for pig iron.

It is sometimes suggested that the iron casting is yielding to the steel casting. As a generalization, this is altogether a misconception, though it is undoubtedly true in a few specific instances. One must compare totals to reach a proper conception. As long ago as in 1906 the production of steel castings amounted to 773,705 gross tons, while in that same year the production of forge pig (of which but a very small tonnage really reaches the puddling furnace), foundry iron and malleable iron, exceed 6,000,000 tons, and making allowance for the consumption of scrap the production of iron castings undoubtedly much exceeded this figure. Yet it is a fact that, with 1912 still to be reported upon, there has been no year in which the production of steel castings has reached a million tons, showing that the tonnage gain since 1906 has been altogether insignificant. Thus the steel casting has not replaced the iron casting to any appreciable extent. The statistics for 1912 when presented will undoubtedly show that in point of percentage the steel ingot has grown much more since 1906 than the steel casting.

Readers' Views and Comments

"Thermistry" and the "Thermist"

To the Editor of Metallurgical and Chemical Engineering:

Sir:—What shall we call the man who has charge of the heat treatment of steel or for that matter of any metal or alloy? He is hardly a metallurgist or metallurgical engineer although those terms are sometimes applied. Neither is he a chemist, although the chemist often has charge of heat treatment; yet the man in charge of the latter may know very little chemistry.

Here then is a distinct position or grade which is becoming more and more important as we more fully understand the possibilities of this branch of metallography. Then why not call it by a distinct name?

Thermistry then, is the art and science of heat treatment and heat measurement and the practitioner would be a thermist. The name is descriptive, dignified, concise, and much less awkward than "metallurgist in charge of heat treatment" or "engineer in charge of heat treatment."

H. M. BOYLSTON.

Cambridge, Mass.

Volatilization Processes in the Metallurgy of Gold

To the Editor of Metallurgical and Chemical Engineering:

SIR:—On page 220 of the April, 1913, issue of METALLURGICAL AND CHEMICAL ENGINEERING, there appears a review of an article originally published in the monthly *Journal* of the Chamber of Mines of Western Australia regarding the recovery of gold by volatilization. In the original article the author, Mr. Ben Howe, states that his process merely takes advantage of a fact long known to metallurgists of the old chlorination school, viz., that when gold ores are roasted with salt the precious metal is volatilized in greater or less degree, according to the conditions of temperature, time and composition of the roasting mixture. By determining the proper conditions for the roast, Mr. Howe finds that he can volatilize the gold to such an extent as to constitute an efficient method of extraction. He further observes that this is "a process, the application of which, at any rate, is new to the metallurgy of gold."

It has been remarked that methods and machines that are old and well known to one generation or to one part of the world are rediscovered and revived by later investigators or in another section. The present instance is a case in point. About 1897 or 1898 Mr. Edwin C. Pohlé and Mr. Stuart Croasdale, then respectively chief assayer and chief chemist for the Globe Smelting & Refining Company, Denver, Col., devised and patented a process based on the same principle as that suggested by Mr. Howe. Mr. Pohlé was first led to investigate the volatilization of metals as a result of serious losses encountered in assaying high-grade silver ores from Aspen, Col. When an assay was made with the customary "salt cover" wide variations occurred in the results, which, however, were more uniform when no salt was used. The two gentlemen named then began a serious investigation of the subject, with the idea of using volatilization in the recovery of metals, particularly from refractory ores.

They found that gold, silver, lead, and copper could be almost completely volatilized from most ores. It was necessary to have in the ore some form of sulphide, in proper proportion to react with the added sodium chloride. The reaction liberated nascent chlorine which combined with the metals to form volatile chlorides. If the ore contained a high percentage of sulphide it was first partly roasted; if it contained none, or not enough, raw pyrite was added. The laboratory tests were simple so far as volatilization was concerned, but recovery was a different matter. The process was finally worked

in an experimental plant in Denver, and later a commercial plant was erected at Mayer, Ariz. The latter was never operated, as far as I am aware.

It may be surmised that the stumbling block to success was the recovery of the volatilized chlorides. As one metallurgist remarked, it was difficult to see the wisdom of converting the metals into a less tangible condition than that in which they already existed in order to attempt their recovery. Bags, sprays, solution towers, etc., were tried at the recovery end, but without success. The problem was never satisfactorily solved. The volatilization feature, however, was attractive in regard to refractory ores, for they usually yielded to the treatment.

There is one possible solution of the recovery problem which has occurred to me since the development of the Cottrell method of fume condensation by means of a high-potential electric discharge. The volatilized metal chlorides might be suitably condensed by the Cottrell process, forming a concentrate of mixed chlorides, which could then be separated by chemical means. In the absence of any information as to how the Australian metallurgist is to recover his chloride fumes, it will be interesting to await further developments. His process, however, certainly is not "new to the metallurgy of gold."

Denver, Col.

H. C. PARMELEE.

Simple Rules for Temperature Conversion

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I am interested to read Dr. Hering's communication in your April issue (p. 176), on mental calculations from the Centigrade to Fahrenheit thermometer scales and vice versa. He omits, I think, one for which I claim no originality, but find convenient. It is not approximate, but exact.

In either case, whether Centigrade or Fahrenheit, add 40 to the reading, then if Fahrenheit take five-ninths and subtract 40, or if Centigrade take nine-fifths of the sum and subtract 40.

My attention was called to this by one of my students, and I find it a simple method.

D. H. CHILDS.

Technical High School,
Buffalo, N. Y.

* * *

To the Editor Metallurgical and Chemical Engineering:

Sir:—As many of your readers know, the conversion of daily temperatures in Centigrade degrees into Fahrenheit degrees is the bug-bear of Americans traveling in Europe, and who has not seen the letter published daily in the Paris edition of the New York *Herald* about the following effect?

"Dear Editor:

Can you furnish me with a simple rule for converting centigrade temperature into Fahrenheit?

"Most sincerely yours,

"YOUNG PHILADELPHIA LADY."

Your last number contained a gallant attempt at a solution of this perennial problem, signed oddly enough "Philadelphia," but alas it was not intended for use in these latitudes, and we hope and trust that the "Young Philadelphia Lady" may never experience, here or hereafter, the temperatures for which Dr. Hering gave his approximate but very convenient rules. No; for purposes of discussing the weather, the problem has not yet been solved, except by the time-honored expedient of memorizing a few fixed points, such, for example, as that 10° Centigrade is 50° Fahrenheit and every 10° Centigrade more or less is 18° Fahrenheit higher or lower than 50° Fahrenheit, every 5° Centigrade is 9° Fahrenheit, and every 1° Centigrade is 1.8° Fahrenheit. This simple mental exercise solves all the weather problems.

South Bethlehem, Pa.

JOS. W. RICHARDS.

Alumina from Clay by the Fluoride Process

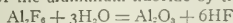
To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have been much interested in the symposium of papers on alumina in your March issue and would like to add something to the discussion.

When a mixture of aluminium fluoride and clay, in such proportions that atoms of silicon and fluorine are to each other as one to four, is heated to a red heat in the presence of water vapor, silicon fluoride passes off and alumina is left.

The only surprising thing about this is the completeness with which this occurs, and the small excess of fluorine needed to secure the complete elimination of silica. In a previous paper (Eighth International Congress of Applied Chemistry, Inorganic Section, Original Communications, Vol. II, p. 67, and Vol. XXV, p. 115), under the title "Action of Some Fluorine Compounds on Clay, Water, etc." I gave results of tests made on some of the details of the process of producing alumina from clay in the above manner. The resulting silicon fluoride reacts with steam or a water spray, the silica is separated by centrifugal action, and the gaseous hydrofluoric acid passes up through a tower filled with clay, over which a small stream of water is flowing. The acid solution from the tower is treated with excess of mildly calcined clay, precipitating the excess silica, much of the iron and titanium, and regenerating the aluminium fluoride.

Two important questions—after the one, "Will it work?"—are, first, the cost of the alumina, and second, the purity. And in that order, because two-thirds of the alumina comes from the calcination of the aluminium fluoride by the reaction



and that fluoride of aluminium can be made of good purity by proper neutralization with clay.

While no claim is made that metal for electrical conductors can be made by this process because small amounts of impurities so greatly affect the conductivity, yet the other uses are for which such metal cannot be properly recommended.

Further on the subject of purity, I find that half the titanium and more than half the iron of the mixture are volatilized with the silica in the calciner. As to costs the first point must be the recovery of the fluorine. From 80 per cent to 90 per cent has been recovered in the laboratory, and probably that figure will stand until some larger tests are made.

On the 80 per cent basis, calculations show a cost per ton, including the factors of labor, power, fuel, acid and fluorspar in the neighborhood of \$17. Appreciating the doubt that properly clouds all calculations based on laboratory tests, it is nevertheless interesting when those calculations give a low value.

We are now building a calcining and absorbing set to try this out on a larger scale.

Attempts have been made to recover the needed fluoride from the gases which are given off in the manufacture of phosphate fertilizer.

These attempts have been frustrated so far by lack of settling space for the dust arising when the ground phosphate is dumped into the bins with the acid. Very little work has been done on this, however.

Technical High School,
Buffalo, N. Y.

D. H. CHILDS.

Segregation and Deoxidation and the Efficiency of Aluminium, Silicon and Titanium as Deoxidizers.

To the Editor of Metallurgical and Chemical Engineering

Sir:—To supplement the notes on the discussion of sound

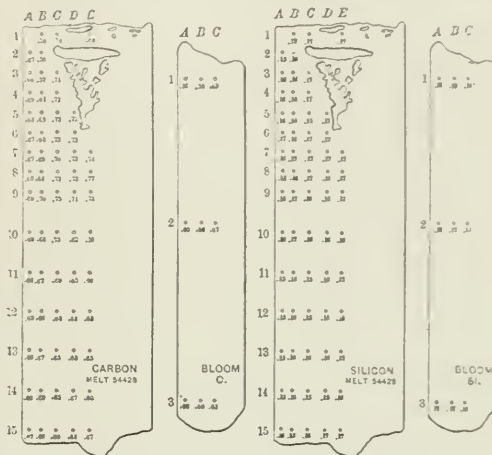


FIG. 3.—CARBON-DIAGRAM OF INGOT AND BLOOM.

FIG. 7.—SILICON-DIAGRAM OF INGOT AND BLOOM.

FIGS. 3 AND 7 OF DUDLEY PAPER

ingots before the American Institute of Mining Engineers, as reported in your March issue, p. 151, the following remarks which I made in the discussion of Dr. Dudley, seem pertinent.



SULPHUR PRINT STANDARD
OPEN-HEARTH

SULPHUR PRINT TITANIUM
OPEN-HEARTH

In reading the very interesting paper of Dr. P. H. Dudley I noticed on the carbon diagram of an ingot the following carbon determinations:

In Fig. 3, line 9, a normal segregation of from 69 to 73; line 10, a negative segregation of from 69 to 50; line 12, a negative segregation of from 69 to 63; line 13, a negative segregation of from 68 to 65; line 14, a negative segregation of from 68 to 60.

I will endeavor to explain why this negative segregation occurs in the lower half of the ingot and submit as a partial explanation the fact that the segregation is induced by the oxides in solution upon which the carbon of the steel reacts.

In examining the silicon diagram of the same ingot as shown in Fig. 7, it will be found that the silicon is nearly uniform. If we admit for a moment that there are some oxides in solution in the center of the ingot and I will try to show later that it is proper that we should, why were these not reduced by the remaining silicon which is supposed to be such an efficient deoxidizer?

This brings us to a discussion of the comparative efficiency of the three deoxidizers mentioned by Dr. P. H. Dudley and Mr. Talbot, viz.: aluminium, silicon, and titanium.

Aluminium as a Deoxidizer.—Aluminium is the most powerful of all known deoxidizers. It will reduce the oxides of both silicon and titanium, but unfortunately by oxidation it will be converted into alumina, which melts at 2010 deg. C. and consequently is infusible at the temperature of the bath. This fact has received only slight consideration in many quarters.

This infusible alumina will remain in the steel and if some part of it by chance is near the top of the ingot it will cover the surface of the pipe and shrinkage cavity, eliminating any advantage which could be gained from immediate rolling, as suggested by Mr. Talbot. It will be noticed that Mr. Talbot has suggested in "well made steel" an addition of 2 oz. of aluminium per ton, or in the case of a 12,000-lb. ingot an addition of 12 oz., which by oxidation will produce 24 oz. of alumina per ingot; this most infusible material will be disseminated throughout the mass in small shot, which later on will be rolled into streaks.

When such streaks of alumina are present between the metallic crystals of a steel rail they destroy the continuity of the metal, having no strength in themselves they are very readily broken and cracks started from this cause are bound to spread through the adjacent metal when it is subjected to the severe stresses and strains of the heavy wheel loads passing over the rail.

Silicon as a Deoxidizer.—Silicon, as already mentioned, is not a powerful deoxidizer and when added to steel it leaves a remainder in it. Usual determinations of silicon give no clue as to the state in which it is found in the metal, whether as silicides, silicic acids or silicates, but if it exists in either of the two latter forms its presence is exceedingly harmful.

Microscopic examination of rail steel discloses small globules of foreign matter consisting of sulphide of manganese and also of oxides and slags. It has been noticed that very often a steel containing as much as 0.10 per cent of silicon in the form of iron silicide will be wild when poured necessitating an addition of aluminium as a deoxidizer in the molds.

Titanium as a Deoxidizer.—Titanium added to the steel in the ladle, after the silicon, will complete the deoxidation. Heats with an addition of titanium of from 0.05 to 0.10 per cent will not require any additional deoxidizer in the molds.

It is the only deoxidizer known which can be used without danger of leaving any of the products of its oxidation in the bath of steel, as pointed out in Dr. P. H. Dudley's paper.

By oxidation titanium produces titanic acid which will flux the slag in suspension in the steel and if sufficient time be given for the reaction to be completed in the ladle, after its addition, all of the products of its oxidation will rise to the surface leaving a clean steel.

If in the Talbot process such a deoxidizer be used to make pipeless ingots, the pipe being clean will weld more readily when the ingot is compressed.

If a comparison were made between two sulphur prints from the cross section of two rails coming from the same part of two different ingots; the rails being of substantially the same chemical composition but the one having been properly treated with 0.10 per cent of metallic titanium, the other being a plain open-hearth steel, the former would not show any slag inclusion and only a very slight segregation, while the latter would in many instances show a dirty steel "full of slag" and an excessive amount of segregation. This is evident from the adjoining comparative sulphur prints.

It will be noticed that the untreated rail has a silicon content

of 0.15 per cent and it is a fact that a few pieces of aluminium were used in the mold in the production of this steel. If, therefore, silicon is a powerful deoxidizer why should there be such an amount of oxides and slags and consequently so much segregation in this steel? The sulphur print of the rail treated with ferro-carbon-titanium does not show anything of the kind and it is therefore reasonable to assume that the elimination of the oxides will result in diminished segregation.

From the foregoing my conclusion is that for the manufac-

ture of good rail steel silicon, a cheap deoxidizer, ought to be used to a limited extent, but that deoxidation should be completed with titanium, which as mentioned by Dr. Dudley will not only react on the oxides, fluxing the slag, etc., but will also reduce the iron silicates formed in the steel by the previous introduction of ferrosilicon.

As to the impression that rail steel treated with ferro-carbon-titanium pipes, I will say that good steel always pipes and in experimenting along this line I have found, although I cannot as yet give a positive explanation for the fact, that silicon and titanium are antagonistic and that the silicon content in rail steel treated with titanium should be reduced within the limits of from 0.06 to 0.10 per cent, for if the silicon is so reduced it will be found that the shrinkage cavity will be similar to the one shown in Dr. Dudley's paper (see adjoining illustration) and that the cavity will be clean and free from all silicates, slag or alumina so as to increase its welding properties when the ingots are immediately



SPLIT INGOT

bloomed as suggested in Dudley's and Talbot's processes.

The drop and exhausted ductility test, for basic open-hearth steel rails is excellent and we are indebted to Dr. P. H. Dudley for its introduction. Very often, however, a rail rolled from a heat treated with aluminium, in accordance with the ordinary practice or by the Talbot or Hadfield process, might pass the drop test to the entire satisfaction of both the railroad and the mill, but as already explained when I mentioned aluminium as a deoxidizer, such a rail containing streaks of alumina, after a certain length of time in service will develop defects which are very often the real origin of broken rails and consequent wrecks.

I wish to state, however, that I do entirely approve of Dr. P. H. Dudley's specification for the manufacture of basic open-hearth rails but will add, as already explained, that if no aluminium be used (which transformed into alumina will coat both the shrinkage cavity and the pipe) the steel bloomed under its equalized initial heat will be rendered pipeless by the usual discard of from 8 to 10 per cent.

Niagara Falls, N. Y.

N. PETINOT.

Nichols-Hesse Dinner

On the evening of Saturday, April 19, a complimentary dinner was given by the chemists of the United States to Dr. William H. Nichols, the president of last year's International Congress of Applied Chemistry, and to Dr. Bernhard C. Hesse, the Secretary of the Congress, as a mark of their appreciation and affection. The dinner was held at the Chemists' Club in New York City, and as some 225 men had come to do honor to the two men who more than any one else bore the brunt of the battle and made the Congress a success, the large lecture room (Rumford Hall) had been chosen for the occasion. It was not by any means merely a New York City affair. The names of the diners as well as of the speakers showed that American chemistry was broadly represented.

That the Chemists' Club as a social institution is fully up to date in every respect, was evidenced by the fact that just before the beginning of the dinner the extra waiters which had been engaged for the evening went on a strike. But it is also a fact that the House Committee settled the strike quickly and that very few of those who sat down to dinner in the prettily decorated hall realized what a narrow escape there had been

on his success. As Mr. Little explained this was one of the rare occasions of this kind which ended with a surplus instead of a deficit. This result was mainly due to Mr. Matheson's highly effective work as treasurer. At the concluding meeting of the organization of the Congress, held just the night before, \$2,000 out of the surplus had been granted to the Chemists' Club.

Mr. C. O. Mailloux read some twelve official letters of congratulations to Drs. Nichols and Hesse from French and Italian societies.

Then the speech making proper began. Dr. Wilder D. Bancroft, of Cornell University, spoke as Past President of the American Chemical Society and of the American Electrochemical Society. He spoke exceedingly well, both concisely and to the point. Starting with a funny church collection story, he passed to a review of the success of the Congress—for it was a great success—and of the men who made it such. Morris Loeb lost his life at it. Dr. Rosengarten nearly did. Mr. Matheson produced a surplus and Dr. Hesse gave up a year and a half of his life to produce thirty volumes of Transactions and a great many tons of literature. And as to Dr. Nichols'



WILLIAM H. NICHOLS



BERNHARD C. HESSE

from a somewhat awkward situation. Well, everything went along lovely. The dinner was excellent and greatly enjoyed. It was simply representative of the potentialities of the cuisine and the wine cellars of the Chemists' Club on one side and of the congeniality of American chemists on the other.

After the dinner was over, the assemblage was graced by the presence of a number of ladies gathering on the balcony of the banquet hall. The speechmaking began and lasted till after midnight.

Dr. Chas. Baskerville, of the College of the City of New York, for the Committee of Arrangements, introduced Dr. Edward W. Morley, of West Hartford, Conn., as the presiding officer of the evening. Dr. Morley who had been the Honorary President of the Eighth International Congress, made a felicitous little speech and concluded by introducing in turn Mr. Arthur D. Little, of Boston, Mass., as the toastmaster of the evening.

Mr. Little spoke humorously and feelingly of the achievements of the Congress and of the rôle which the personal element had played in it, of the work of Nichols and Hesse, and of the tragic death of Morris Loeb.

At the conclusion of his introductory speech Mr. Little read a long series of messages, including letters from Dr. Chandler, Dr. Remington, Mr. Kuttroff, Mr. Adamson, Dr. Bogert, Dr. Cameron, and Mr. Wm. J. Matheson. Mr. Matheson had been the treasurer of the Congress and a toast was drunk to him

work as president, if we should ever go again through such a Congress in our life-time (which Heaven forbid) and if we should be forced to get along without Dr. Nichols as president (which Heaven forbid)—we might think in poetry:

Twinkle, twinkle little star—
Riding on a trolley car—
Trolley car went off the track—
Gee, I wish we had our Nickels back.

The next speaker was Dr. William Brady, of the Illinois Steel Company, of South Chicago, Ill., who represented the American manufacturers. He spoke of the American manufacturer as a business man. His first concern is the quality of his product and he educates the American public in quality. Then he looks for quantity and systematizes and organizes. Now he is very much interested in his men and their safety. If it is possible to go to extremes, the American manufacturer is willing to go to extremes in the matter of safety of his employees. Dr. Brady concluded with a happy reminiscence from early life in a country school.

Mr. Little then introduced Dr. Leo. H. Baekeland, of Yonkers, N. Y., as Past President of the American Electrochemical Society and of the American Institute of Chemical Engineers ("the only aristocracy among American chemical bodies"), as the discoverer of velox paper and the "anonymous discoverer of bakelite," "but the discovery which has most endeared him to his friends was his discovery of Mrs. Baekeland."

Dr. Baekeland in his happy speech asked: "Was it worth while?" Answers to questions as to worthwhileness depend on the point of view. If different viewpoints were not possible, if viewpoints could not be exaggerated, honest men could not differ in discussions at all. But as applied to the past Congress, different viewpoints were hardly possible. He pitied anybody for whom the Congress had not been very much worth while. Of course, those will be disappointed who want to get more out of the world than they put into it. The man behind the machinery and organization was Dr. Nichols. Dr. Nichols' work at the Congress has been a school for every one of us. "I had a bully good time."

Dr. David T. Day, of Washington, D. C., spoke of the Congress as a lightning express on its way to Russia. Hesse was at the switch and got the proceedings of the Congress out before the Congress began to proceed. It was a wonder what it is possible to get out of a Hessian crucible. The calm spot in the center of the cyclone was Dr. Nichols. The inspiration came from him. He put the different men together and made

Dr. B. C. Hesse had before him a complete bound set of the *Transactions* and told something of the troubles which he had to meet as the secretary of the Congress. But from his statistical figures it seemed that out of eight there were seven men satisfied and only the eighth one was disappointed. The documentary evidence produced by Dr. Hesse as to the reasons why some were disappointed was truly amusing. "Get the eighth fellow to do as the rest of us do." Dr. Hesse concluded with hearty words of thanks to Dr. Nichols and to the chemists of the United States.

A toast drunk to the president of the Ninth International Congress of Applied Chemistry, Dr. Walden, in St. Petersburg, concluded the evening, which will long be remembered.

In the following we give the alphabetical list of those who participated in the celebration in honor of Dr. Nichols and Dr. Hesse:

L. Albrecht, C. Amend, O. P. Amend, T. Armstrong, F. W. Atkinson, W. S. Ayres, L. H. Baekeland, F. H. Baldwin, W. D. Bancroft, C.



PHOTOGRAPH TAKEN AT NICHOLS-HESSE DINNER

them work, and while he worked very hard in doing so, he was not used up himself. That is, he was the catalytic agent. Nickel as a catalytic agent may tire at times. Nichols does not, being a plural and representing a polymerization product, Dr. and Mrs. Nichols. In order that Dr. Nichols and Dr. Hesse should take with them into later life the memory of the appreciation and affection of their American fellow chemists Dr. Day presented, in the names of the American chemists, to Dr. Nichols a counterpart in silver of the desk set in the Royal Treasury in London and to Dr. Hesse a silver loving cup.

Dr. William H. Nichols, in his gracious words of appreciation, pointed out as one great lesson of the Congress the enormous amount of ability and chemical knowledge stored up in our universities, colleges, laboratories all over the country, and the enormous amount of hard and untiring work done there in a quiet and unostentatious way. He pointed out that this work needs more encouragement. He pledged his aid to render it possible in future to make the most of this work of college professors for the good of their students and the good of the world. This being a true problem of "conservation," nothing of these vast intellectual resources of the country should go to waste.

Baskerville, H. P. Bassett, G. C. Bauer, H. V. Berg, W. D. Bigelow, M. W. Bingham, H. B. Bishop, F. Black, V. G. Bloede, E. Boise, S. Bookman, F. C. Bowman, W. Bowman, C. S. Bradley, W. Brady, A. A. Breneman, J. H. Brickenstein, A. von Briesen, T. I. Briggs, C. A. Browne, H. Buel, C. F. Burgess, C. F. Chandler, S. R. Church, C. B. Clark, V. Colbentz, E. J. Cole, A. H. Cowles, H. N. Cox, F. D. Crane, A. S. Cushman, W. H. Davis, D. L. Davoll, D. T. Day, E. deMeuse, C. Dill, R. E. Doolittle, C. A. Doremus, J. Douglas, L. A. Dreysfus, H. Dullois, T. R. Dugan, B. W. Dunn, A. W. Ellens, W. H. Engels, W. H. Erhart, W. J. Evans, H. A. Fales, W. C. Ferguson, C. G. Fink, T. B. Freas, H. H. Fries, M. M. Garver, J. F. Geissler, A. E. Gibbs, L. Gifford, T. T. Gray, W. S. Gray, E. V. Grano, A. Gref, A. P. Hallock, C. Hancr, E. Hart, J. Hasselacher, F. R. Hazard, F. Henningway, E. Hendrick, J. H. F. Herreshoff, C. H. Hervey, A. E. Hill, N. Hill, Jr., H. W. Hilbery, K. J. Holliday, F. Holzrichter, W. D. Horne, J. W. Hornsey, H. Howard, A. Humphreys, H. A. Huston, A. von Isakovics, M. Ittner, F. Jager, A. B. Jones, C. M. Joyce, J. Kahn, W. C. King, P. C. Kingsbury, E. D. Kingsley, E. C. Klipstein, D. P. Knowland, A. E. Kraemer, O. H. Krause, H. J. Krebs, G. F. Kunz, A. Kuttroff, K. B. Lamb, N. Lane, J. Langlois, A. C. Langmuir, N. A. Laury, A. D. Little, E. G. Love, F. S. Low, B. A. Ludwig, C. S. Lukins, K. G. MacKenzie, C. O. Mallow, J. R. Marble, S. C. Mastick, W. J. Matheson, H. F. Merriam, H. A. Metz, F. J. Metzger, D. Miller, W. L. Miller, H. I. Moody, R. W. Moore, S. R. Morey, J. L. Morgan, L. Morgan, F. W. Morley, R. K. Murphy, J. J. R. Muirling, W. S. Myers, H. S. Neuman, A. C. Neish, C. W. Nichols, F. H. Nichols, W. H. Nichols, J. H. Nield, S. J. Norris, G. Ornstein, G. Palmenberg, T. J. Parker, C. J. Parsons, E. D. Pearce, G. H. Pearson, J. C. Pennie, C. Pickhardt, W. P. Pickhardt, L. Putkin, A. Plaut, C. E. G. Porst, M. R. Pouchet, F. H. Pough, T. W. Prichard, J. de Puligny, A. M. Purves, J. W. Richards, C. F. Ruge, A. E. Rising, A. Rohl, E. F. Roeder, A. Rogers, G. D. Rosengarten, C. Roth, W. E. Rowley, H. D. Ruhm, L. Saarbach, M. F. Schaak, E. Schill, J. D. Schwentzman, R. W. Seabury, H. C. A. Seeborn, F. Secler, R. A. Shaw, A. Smith, E. E. Smith, T. E. Smith, E. A. Sperry, J. Speyer, J. W. Starr, S. Steele, G. C. Stone, I. F. Stone, E. H. Strickler, F. Stutz, O. Sussman, C. E.

Thies, B. F. Thomas, G. W. Thompson, M. Toch, C. Travelli, S. A. Tucker, H. C. Tuttle, A. P. Villa, T. B. Wagner, L. I. Waldman, W. H. Walker, J. M. Weiss, E. Weston, M. C. Whitaker, L. R. Whitcomb, C. T. Whittier, F. G. Wiechmann, H. Wigglesworth, H. A. J. Wilkens, G. F. Willett, F. Wyatt, W. D. Yearseley, C. B. Zabriskie.

International Geological Congress

The Twelfth International Geological Congress will meet in Toronto, Canada, from August 7 to 14, 1913. The headquarters will be at the University of Toronto.

The membership fee is \$5.00.

Titles of papers and proposals to be presented should be submitted to the General Secretary, R. W. Brock, director of the Geological Survey of Canada, Ottawa, Ont., Can., as early as possible. Contributions must be received by May 1st if they are to be printed in time for distribution before the meeting.

The official language of the Congress is French, but contributions may be submitted in French, English or German.

The following topics have been selected by the executive committee as the principal subjects for discussion: Coal resources of the world; differentiation in igneous magmas; the influence of the depth of the character of metalliferous deposits; the origin and extent of the pre-Cambrian sedimentaries; the sub-divisions, correlation, and terminology of the pre-Cambrian; the extent of which the ice age was broken by inter-glacial periods; the physical and faunal characteristics of the palaeozoic seas, with reference to the value of the recurrence of seas in establishing geological systems.

A great many different excursions have been arranged, preceding the sessions of the Congress, during the sessions, and after the sessions. Details concerning the same may be found in the second circular issued by the Congress. This circular as well as all other information may be obtained from the Secretary of the Twelfth International Geological Congress, at the Victoria Memorial Museum, Ottawa, Ontario, Canada.

Electrochemical Spies in Niagara Falls

A mild sensation has been recently caused in Niagara Falls electrochemical circles by the capture of a Norwegian engineer in the act of making drawings and notes in one of the larger plants. An examination of the man's effects brought to light data from a number of plants in Niagara Falls and elsewhere.

His mode of procedure was to get a job in a repair gang thus securing access to more departments than would be possible in special work. Two accomplices went about the country with this chief, following the same lines.

A confession by a compatriot in one of the plants brought out the fact that money was plentiful and was freely used where necessary. From the same source it was hinted that large foreign interests were responsible for the presence of these spies and financed the operations liberally. One of the names used at Niagara Falls by this man was Kverne, those of the assistants, a Swede and a German, not yet having been obtained.

Publicity is given these facts as a warning to manufacturers, as the considerable ability of this engineer, evidenced in his notes, makes his presence dangerous in plants where secrecy is considered essential.

Closing of Norwegian Electric Iron Ore Reduction Plant

The Electric Iron Smelting Company Hardanger, of Norway, has been forced, after three-quarters of a year's operation, to close down its factory, according to information published in Scandinavian newspapers. The iron smelting process used by the Hardanger company was the well-known electric shaft furnace process developed by the Swedish engineers, Grönvall, Lindblad and Stalhane, and which is in successful operation at different steel plants in Sweden.

The news that the Hardanger plant has been closed may appear startling, contrasted with the successful operation of furnaces of the same type in Sweden. The reason is due to inavailability of suitable raw materials in Norway. The

Swedish electric-blast furnace has to be charged with charcoal, which is readily available in Sweden, while the Norwegian plant was using coke, since charcoal was not obtainable at the plant at a reasonable price.

During the experimental work of developing the electric-shaft furnace at Trollhattan coke was found unsatisfactory in the operation of the furnace, so the closing of the Hardanger plant only confirms earlier results.

The Hardanger company owns the water-power which was used for the electric smelter, amounting to 14,000 hp, and have leased it to a French company, which will utilize it for electrochemical industries.

The Non-Ferrous Metal Market

Since our last report the non-ferrous metal market has been affected by industrial and political conditions both in this country and abroad. Floods and the prospect of tariff changes have combined to affect the domestic spelter market; Mexican revolution has resulted in reduced exports of lead from that country to Europe, with a consequent advance in the foreign market, which was later accentuated by labor strikes in Australia; the tin market has been subject to wide fluctuations, but the copper market has become steadier.

Copper.—The depression which was manifest in March was removed in April. The influence of heavy buying in Europe was felt in the domestic market, where most of the business was for foreign trade. The statistics for March, which were published early in April, were rather favorable, and the situation is considered much improved. The last quotations are 15.35@15.55 cents for Lake, and 15.20@15.30 for electrolytic.

Tin.—Although the foreign market was subject to wide fluctuations in the latter part of March and the first part of April, the publication of the March statistics produced an optimistic tone in the market, and prices have become firm at about 48½ cents, New York.

Lead.—The foreign market has been very firm on account of conditions reported from Mexico and Australia, but the domestic market is quiet and unchanged at 4.30@4.35 cents, New York, and 4.15@4.20 cents, St. Louis.

Spelter.—Sales have been effected only at concessions, and the market has declined. Production has been heavy, and plenty of metal is available for sale. A large tonnage has been sold. The last quotations are 5.70@5.80 cents, New York, and 5.55@5.65 cents, St. Louis.

Iron and Steel Market.

The iron and steel market lost tone quite decidedly in April. Conditions had been practically stationary for an unusually long time. Prices of finished steel products had ceased to advance early in October and after a gradual speeding up the mills reached a state of operating at full capacity at about the same time. The placing of contracts for forward delivery reached a high point in October, dwindling thereafter, while during the first three months of this year contracting was uniformly light. In the late months of last year specifications against contracts were extremely heavy, greatly exceeding the shipments, while during the first three months of this year they ran quite uniformly at a rate approximately equalling the shipments.

Thus since October as to prices and the productive rate, and since the beginning of the year as to contracting and specifying, the steel situation had shown no change. In April a definite departure occurred, but the departure was of such a nature that we describe it as merely a loss in ton. Production continued at the full rate. Prices did not recede. Specifications decreased, falling definitely, though but slightly, below the current shipments. Contracting for forward delivery ceased almost entirely.

The mills had a very large amount of business on books in the form of actual specifications, and the decreased market activity of April did not impair their position, as to the rate of production or as to prices. Continued for two months longer.

the condition probably would do so. In some finished products the mills are booked far beyond midsummer, but without an improvement over April conditions the situation would be impaired by July 1 in some directions.

Whether this relative dullness will continue for a considerable time is naturally dependent upon the causes which produced the condition. In the trade, the common theory is that the tariff discussion is responsible, and if so a definite improvement could hardly be expected for several months. A closer analysis would suggest that there is at least one other cause, that being a psychological one, but one nevertheless susceptible of analysis. In the long run materials make the market, but the market as one finds it from week to week is made by men, not by materials. It proceeds by great movements, for a period upward and for a period downward. The upward movement occurred last year. Then came an anomaly: for six months there was substantially no change in prices, and very little change in general conditions. So long a period of unchanged prices was abnormal, a definite change being overdue. There was not enough latent strength in the situation to produce a fresh upward movement, hence it was natural that the market should lose tone instead. The upward movements are assisted by the price advances, which encourage buyers to contract ahead; with unchanged prices they are not so moved.

While the market has lost ground, it stands in no great danger. The mills are well booked ahead, as to the total tonnage, though here and there are mills with only a little tonnage. There is no danger of cancellations, for nothing new has occurred. A year ago well posted men were convinced that the Democratic party would be successful, and that it would revise the tariff this year, substantially along the lines now proposed. If there is no material improvement from the market condition of April, a slight decrease in production will occur during the midsummer months, but a relatively high rate of activity is assured until almost the close of the year. The serious doubt is as to next year, rather than as to the next six or eight months.

Pig Iron.

The dullness in the pig-iron market continued through April, and was accompanied by somewhat more rapid declines in prices. These declines began late in December, activity having decreased in October and November, but in the first three months of the year the pig-iron market as a whole, averaging all grades and all districts, lost in prices by only about 25 cents per ton per month. In April the losses were at double or triple this rate. The pig-iron decline has been due to psychological rather than material causes. In the first place, producers marked prices up too rapidly, in the late months of last year, having regard to the rate of demand and the large amount of idle capacity. Production by the merchant furnaces increased rather rapidly from October to February inclusive. Once it became established that prices were not to advance further, buyers adopted their usual waiting attitude, and a period in which buyers bought more than their current consumption required, making purchases for further forward delivery, was succeeded by a period in which they bought practically nothing. The declining prices have confirmed them in their attitude. No new thing occurred in April, for as early as February it became certain that a fresh buying movement would not occur until there had been the usual test of strength, by prices declining to a point which would find some furnaces going out of blast or would find some buyers forced to purchase. That test has not yet occurred, but it will probably occur soon. Production is stationary, consumption is undiminished and if stocks as a whole are increasing they are increasing very slowly, for in some districts, notably the Pittsburgh-valley district they are lower than for a long time. Prices are down nearly to the cost of production at many furnaces now operating. We quote: Bessemer, \$17; basic, \$15.75; malleable, \$15.50; No. 2 Foundry \$15.25; forge, \$14.75. f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh; foundry iron at Philadelphia, \$17.25; Buffalo, \$15.50. Cleveland, \$16.25; Chicago, \$17; Birmingham, \$12.50.

Steel.

The floods of late March in the Central West caused a total loss in steel output of more than 250,000 tons, which fell largely in mills producing billets and sheet bars, as a result of which the supply in April was poorer than ever. Prices are nominally unchanged at \$29 for billets and \$30 for sheet bars and rods, at maker's mill, Pittsburgh or Youngstown. In resuming after the floods the blast furnaces led the steel works, so that the steel works are in comfortable position as to pig iron. About one-half of all the sheet and tin mills were closed on account of flood, and their resumption was retarded by poor deliveries of steel. A Youngstown interest has sold several lots of sheet bars, aggregating 25,000 tons or more, for second half delivery at \$27.50, maker's mill, but this seems to exhaust its offerings, and the only market quotable is that for relatively early delivery, given above.

Finished Steel.

Premiums for finished steel products for early delivery are quite unusual, except that plates usually command a premium of \$1 to \$2 a ton and merchant steel bars bring prices up to 1.75 to 2 cents, depending on quantity. Regular mill prices are given below. Under date of April 12 the National Tube Company issued a new list, advancing steel pipe by about \$1 a ton, and the new list was adopted by independents, eliminating the slight concessions which had been given from the old list by several mills. Slight shading in sheets and tin plates disappeared in April, on account of the flood decreasing production, but on galvanized sheets the shading reappeared, to the extent of \$2 a ton.

Regular prices at Pittsburgh, unless otherwise stated, are as follows:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.
Plates, tank quality, 1.45 cents.
Shapes, 1.45 cents.
Steel bars, 1.40 cents, base.
Steel hoops, 1.60 cents, base.
Common iron bars, 1.65 cents, Pittsburgh; 1.67½ cents, Philadelphia; 1.57½ cents, Chicago.
Wire nails, \$1.80 per keg, base; plain wire, 1.60 cents.
Sheets, blue annealed, 10-gage, 1.75 cents; black, 28-gage, 2.35 cents; galvanized, 28-gage, 2.40 to 2.50 cents; painted corrugated, 2.55 cents; galvanized corrugated, 3.55 cents.
Tin plates, \$3.60 for 100-lb. cokes.
Merchant steel pipe, ¾-in. to 3-in., 70½ per cent off list.
Steel boiler tubes, 3½-in. to 4½-in., 70 per cent off list.
Standard railroad spikes, 1.80 to 1.85 cents.
Button-head structural rivets, 2.10 to 2.20 cents; cone-head boiler rivets, 2.20 to 2.30 cents.

The Western Metallurgical Field

Labor Strike at Globe Smelter

The beginning of April witnessed a labor strike at the Globe plant of the American Smelting & Refining Company, Denver. The Western Federation of Miners induced the men to walk out and demand more pay and better working conditions. The strike continued for a week or ten days, during which time the smelting furnaces were banked or run at such capacity as could be maintained by the technical staff of the plant. The smelting company refused to treat with the Western Federation, but offered to listen to a delegation of the strikers. After being out about ten days, the strikers returned to work, and the plant is again running at usual capacity.

An investigation of the conditions at the smelter indicated that they were not as bad as claimed by the Federation, neither were wages as low as stated. Another rumor and press report, which proved to be untrue, was to the effect that the strikers who returned to work did so at reduced wages.

Prospecting for Potash in Death Valley, Cal.

Preliminary results of the Geological Survey's drilling in Death Valley, California, have now been reported, and the following brief summary is issued:

Reports state that practically the entire area in Death Valley has lately been located as "potash" claims, presumably as association placers, but in blocks of very large area. It is said that a single tract of 17,120 acres, known as the Kali property, was located in May to July, 1912, on the lowest part of the valley. Other groups both north and south of this tract have subsequently been taken up. A vast amount of saline matter is accumulated on and beneath the floor of Death Valley, and it is logical that this area should have been one of the first to attract attention in the search for soluble potash salts.

Four wells have recently been drilled by the Geological Survey to test the general character of the deposits that lie beneath the floor of this valley. Three of these wells were placed in the area of smooth salt in the lowest depression of the valley. The fourth was located in the area of rough salts about 20 miles north of the others. The records of all of these holes are very much alike. These borings are believed to be the first information that has been obtained concerning the character of the deposits underlying the floor of this valley.

Eighty-three samples of muds, salt crystals, and waters or brines have been preserved from these borings for further chemical examination. The preliminary results, given in the accompanying table, include only potash determinations made on natural brines collected from the beds that yielded the principal flows. The analyses are by W. B. Hicks and R. K. Bailey.

Description of Sample	Depth in Feet at Which Sample Was Taken	Total Salts in (Ignited Residue) Percent- age of Original Sample	Potash (K ₂ O) in Total Salts Expressed as Percent- age of Ignited Residue	Potassium Expressed as Percent- age of KCl in Original Solution
Ground water in the salt crust at the "sink".....	.5	28.19	3.43	1.53
Water in open "pothole".....	.5	27.47	1.20	.52
Water in open "pothole" (same).....	9.5	27.48	1.18	.51
U. S. G. S. well, No. 1.....	6.0	27.87	2.85	1.27
U. S. G. S. well, No. 1.....	24.0	28.64	2.22	1.01
U. S. G. S. well, No. 1.....	29.0	28.96	2.35	1.09
U. S. G. S. well, No. 1.....	52.0	28.66	2.01	.91
U. S. G. S. well, No. 2.....	32.0	28.33	1.54	.69
U. S. G. S. well, No. 2.....	38.0	29.16	1.78	.82
U. S. G. S. well, No. 2.....	70.0	29.96	2.48	1.18
U. S. G. S. well, No. 3.....	1.0	27.78	2.05	.90
U. S. G. S. well, No. 3.....	30.0	27.91	1.68	.74
U. S. G. S. well, No. 4.....	32.0	28.77	2.23	1.02
U. S. G. S. well, No. 4.....	38.0	28.73	2.12	.97
Average.....		28.42	2.08	.94

A vast amount of saline material is accumulated in the bottom of this valley, but the mode of its deposition probably is not favorable to selective crystallization on a large scale. Segregation of potash or any other portion of the soluble constituents of the waters may have taken place to a slight extent in the individual salt crust layers, but under the conditions described any such differentiation is likely to have been restricted to the individual layers as units, and therefore has occurred on a scale so small as to be of doubtful practical importance. It seems evident that unless a vast body of saline material has been deposited at one time during a single period of desiccation that there would be little chance for the various dissolved constituents to become segregated one from another on a large scale. There is no record of the drying up of a single large lake of saline waters in Death Valley. Although it is possible that the shores of such a lake might have been completely buried, the assumption that this may have happened must be purely a matter of speculation. Similar reasoning may apply to many other areas in the desert basin region. Great interior lakes have existed in certain areas and may have dried up under such conditions that the salts they contained were deposited in a great body and that the potash and other minerals in the waters were to a certain extent segregated in some portion of the deposits by crystallization. Searles Lake contains a deposit of salt evidently produced in this way, and others may be found, although they are not now exposed at

the surface. It does not appear that there is much theoretical justification for the belief that such deposits must be present in Death Valley. It is, of course, possible that the present conclusions are based on insufficient negative evidence, and for this reason any further drilling that may be carried on by private claimants in the Death Valley region should afford a record of much importance and interest in this connection.

Tendencies in Concentration

In a paper recently read before the Colorado Scientific Society, Mr. Edward S. Wiard, of Denver, gave the following classification of concentrating ores, based on the degree of crushing required to unlock their constituent minerals:

Class A. Ores yielding on a coarse initial break, clean concentrate and clean tailing for the large size (between 1¼ in. and 10-mesh), and similar products in the sand and slime sizes.

Class B. Ores yielding on a coarse initial break, large size clean concentrate, large size clean tailing and large size middling.

Class C. Ores yielding on a coarse initial break, large size clean concentrate and a corresponding middling.

Class D. Ores yielding on a coarse initial break, large size clean tailing and a corresponding middling.

Class E. Ores yielding on a coarse initial break, nothing but large size middling; and as a sub-class, ore yielding two or more grades of large size middlings.

The definition of the first three classes indicates a brief outline or flow-sheet for the treatment of a majority of ores. Examples of Class A ores are very rare; but recently a good example has come to light in an ore containing 13 oz. zinc per ton, 12 per cent lead and 32 per cent zinc. On crushing coarsely to a limiting size of about 10 mm., screening and classifying, the screen sizes were jigged and the classified products treated on tables. The jigs yielded clean lead concentrate of 63 per cent lead and zinc concentrate of 48 per cent zinc. The jig tailing contained only 1 per cent lead and 6 per cent zinc. Similar products were obtained from the tables, the slimy discharge from which contained only 1.2 per cent lead and 13 per cent zinc, amounting to only 3 per cent of the total mill feed. The approximate percentage saving for the whole operation was 88 per cent of the lead, 90 per cent of the zinc and 85 per cent of the silver.

In large size mills, say above 100 tons capacity, there is a tendency to abandon the familiar trommel line in favor of one or more sets of splitting screens which divide the feed into oversize and undersize for direct feed to jigs. The reason for this trend is found in the excessive cost of maintenance of trommel lines, the time lost in repairs, and the inefficiency of the device under large loads, with the accompanying tendency for the feed to build up at certain points in the system.

In jigging practice the criterion has been the production of concentrate, and it has been commonly believed that unless a shipping product could be made, jigging was not profitable. It is possible, however, by jigging, to produce an enriched portion for subsequent fine regrading without very much loss, and the enriched portion could be concentrated to the same tailing that would obtain if all the ore had been ground fine before separation. Jigging in such a case would give a greater total saving, and the required equipment would be less.

Company Reports

The American Smelting & Refining Company's fourteenth annual report, for the calendar year 1912, contains little or no matter of general interest. From the financial figures submitted it appears that the company had a profitable year, but the balance of the report lacks detail and definiteness. As an incentive to good service on the part of employees, the company appropriated the sum of \$307,822 for distribution as a cash bonus. A new pension system has been adopted whereby male employees who have reached the age of sixty-five years and female employees who have reached the age of sixty years, and who have been continuously in the employ of the company, or of acquired companies, for twenty years, may be retired. The

amount paid to pensioners will be based on an allowance of 1 per cent for each year of service of the average annual pay during the ten years next preceding retirement, but no pension will exceed \$200 per month or be less than \$20 per month. A sum of \$500,000 will be invested to meet the requirements of this system.

The Mount Morgan Gold Mining Company, Ltd., Queensland, Australia, has made public the report of the general manager, Mr. Benjamin Magnus, for the half year ended Nov. 30, 1912. The operations in the mine were very satisfactory and were attended by a gradual reduction in cost. The copper smelter treated the following tonnage of ore:

	Tons treated	Copper produced tons	Gold produced oz.
Copper ore	93,783	3,276	42,744
Gold ore	19,895	530	18,831
Many Peaks ore.....	54,394	894	413
Purchased ore	178	23	9
Miscellaneous	4,173	282	555
Total.....	172,433	5,005	62,552

The permanent closing of the Mundic works has caused the smelter to handle the silicious ore formerly treated there. This has increased the cost of producing copper, but the increased gold value of the blister copper gives greater net returns on the operation. A new smelting plant has been started, entirely independent of the old plant, and will embody the latest ideas in modern copper smelting. This new plant is warranted not only on account of the increased net return which will result, but also by the life of the mine. In the meantime improvements are being made at the old plant so that costs there will be cut down as far as possible. Mr. Robert Sticht is assisting Mr. Magnus in the design of the new plant.

The Goldfield Consolidated Mines Company, Goldfield, Nev., effected material reduction in costs during the year 1912, according to the sixth annual report of that company. The ore produced from the mine had an average value of \$19.77 per ton, while the average gross realization from ore treatment was \$18.40 per ton. Costs were reduced by \$1.36 per ton; the noteworthy items of reduction being, milling, 28 cents per ton; marketing concentrates, 61 cents; marketing bullion, 8 cents; general expense, 10 cents. The total expense of milling was \$1.61 per ton; concentrate treatment, \$0.38; marketing bullion, \$0.07. The net realization from operations was \$11.75 per ton. This amounted to 59.44 per cent of total production. The operation of the concentrate treatment plant increased net profit from milling operations approximately 50 cents per ton of ore milled.

The Miami Copper Company's annual report for 1912 states that the sixth and last section of the mill was started about the last of February and that the average tonnage treated per section per day was 489 tons of ore. The ore has proved hard to crush, and rigid rolls have been replaced by heavy spring rolls, and the Chilean mills by Hardinge pebble mills. Two important experiments were conducted during the year. The first showed that no greater extraction or reduced cost could be obtained by using the so-called "roughing" process. The second, which is still in operation, indicates that a re-treatment process, consisting in regrinding the coarsest sands from the sand-floor reject and middling, followed by classification and table treatment, will give an increased saving of about 1½ lb. copper per ton of ore. The net smelter returns of refined copper amounted to 32,832,609 lb. Based on this return, the cost of refined copper in concentrate on board cars at Miami, was \$0.06477 per lb.

The Old Dominion Copper Mining & Smelting Company has issued its annual report for 1912. The following items relate to metallurgical operations. Concentrating ore treated in 1912, was 166,870 tons containing an average of 3.37 per cent copper, as compared with 130,230 tons of 3.99 per cent grade in 1911. The costs for 1912 were, nevertheless, lower than for 1911, being \$0.753 and \$1.012 per ton, respectively. Smelting costs for 1912 were \$2.304 per ton of new charge smelted, as

compared with a similar cost of \$2.568 in 1911. The cost of converting per ton of fine copper was \$9.45 in 1912, and \$8.26 in 1911. Included in the 1912 cost is an item of \$13,500 for improvements. The old practice of carrying converter fumes directly into the furnace flue-dust chamber was abandoned, and a new flue-dust chamber was erected for converter gases. In six months the recovery of dust has amounted to 34,000 lb., averaging 45 per cent copper. Over 87 per cent of this was finer than 200-mesh, and would have been lost under the old system. The first unit of a basic converting plant has been installed, being of the Great Falls type, and it is expected that converting costs for 1913 will be materially reduced.

The fifth quarterly report of the Chino Copper Company, covering the fourth quarter of 1912, shows a total production of copper for the year of 29,237,966 pounds. Each succeeding quarter of the year showed an increase in tonnage treated and copper produced. During the fourth quarter the quantity milled was 452,650 tons of ore, averaging 1.841 per cent copper, yielding 10,781,621 lb. copper. The grade of ore was lower than for the preceding quarter when it averaged 2.33 per cent copper. The total net profits from all sources for the year was \$2,352,822. The earnings for the fourth quarter were calculated on an average price of 14.532 cents per pound of copper. The cost per pound of net copper produced, after making smelting deductions, was 8.49 cents per pound, as compared with 6.57 cents for the preceding quarter. This increased cost is due partly to the ore treated, which contains a large percentage of iron, thus giving a lower ratio of concentration and lower grade of concentrate. The ratio of concentration was 15.14:1, the concentrate averaging 18.03 per cent copper as compared with an average of 25.02 per cent for the preceding quarter. The average extraction for the quarter was 64.58 per cent, compared with 65.15 per cent in the preceding quarter. The fifth section of the mill was put in operation November 28, 1912, but practically only four sections have been in operation during the fourth quarter on account of making improvements.

The Ray Consolidated Copper Company's report for the fourth quarter of 1912 shows that the gross production for the year was 35,861,496 lb. copper. The average grade of ore treated during the fourth quarter was 1.6737 per cent copper, which is about the average for the year. The extraction was 68.102 per cent, compared with an extraction of 68.2783 per cent for the whole year. The average cost per pound of net copper produced during the quarter was 9.828 cents, as against a cost of 10.047 cents for the first three quarters. During the quarter six sections of the mill at Hayden were in operation. The seventh section is now available, and the eighth is nearing completion.

In the annual report of the Arizona Copper Company for the year ended September 30, 1912, the general manager gives the following data regarding the operation of mill and smelter. During the year 782,100 tons of ore were concentrated, the ratio being 6.74 tons into 1. Improvements in the concentrating plant raised the daily tonnage from 907 tons per running day in the previous year to 1573 tons last year. Concentrate re-dressing plants were installed at both mills last year, for the purpose of making a higher grade, cleaner product. At the oxide ore concentrator and leaching plant 108,230 tons of ore were treated, yielding 89,800 tons of tailings which were treated in the leaching plant. This treatment yielded 0.71 per cent of the total copper production.

In the smelting department the average yield from ores and concentrates was 12.03 per cent copper. The elimination of silicious matter in the re-dressing plant at the concentrator reduced the amount of flux required, increased the capacity of the furnaces and effected an economy in smelting. In the converter department one converter was run with basic lining, the operation being so successful that the remaining converters have been changed to conform to that practice.

The erection of the new smelting works is progressing rapidly, and it is expected that operation will commence in late summer or early autumn. The prospective date of completion and operation is considerably earlier than expected.

Evolution in Methods of Handling Slime—II

By H. N. Spicer

New Zealand is an interesting country from a metallurgical point of view, particularly on account of its contribution of the tall, pneumatic-slime-agitation tank to the cyanide industry. This device is the invention of F. C. Brown and S. D. McMiken, of the Komata Reefs mill, where the first of these tanks was built and used for the agitation of ore slime. Owing to the fact that this tank was first adopted in North America at Pachuca, Mexico, it has come to be more commonly known as the Pachuca agitator, thus in a way detracting from the credit due to its inventors and the country of its origin.

It will be apparent from the following notes that the methods of handling slime in New Zealand are much more advanced than those described in the preceding paper on India, in which country the crudest methods have prevailed up to the present time. As in other countries, certain typical features have been

ders which carry off the clear water. In the center of each tank is a vertical shaft with a heavy wooden arm attached at the bottom, the whole being slowly revolved by an overhead gear. Near the periphery of the bottom of the tank is a 4-in.



FIG. 2.—REINFORCED CONCRETE PACHUCA TANKS, WAIHI MILL

valve which is opened at intervals of half an hour for the discharge of accumulated thick slime. This product contains about $1\frac{1}{2}$ parts of water to 1 of solids. It is then mixed with cyanide solution and pumped to Pachuca agitators.

These agitators are worth more than passing mention, as they are built of reinforced concrete, 55 ft. high and 13 ft. diameter, inside measurements. The walls are 5 in. thick; but similar tanks at another mill have been built with only 4-in. walls, and have proved satisfactory. The reinforcement is old rails and wire rope, the latter being woven horizontally through the former which stand vertical. The cost of these tanks was appreciably lower than the bids for similar steel tanks, indicating that under certain conditions where cement and labor are comparatively cheap, it may prove profitable to build concrete agitators. A closer view of the agitators is given in Fig. 2.

A different procedure obtains at the Waihino mill of the same company, shown in Fig. 3. Sand and slime are separated in spitzkasten, the former being shovelled into filter-bottom concrete tanks and leached with cyanide solution. The slime from the dewatering cones, containing about $1\frac{1}{2}$ parts water to 1 of solids, is further dewatered by vacuum filters which reduce the moisture to about 30 per cent. These filters are of the stationary Moore type. After a cake has been formed, the vacuum is cut off and the slime allowed to fall back into the tank. If necessary, compressed air is blown into the leaves to aid in dislodging the cake. The tank has a V bottom, terminating in a goose-neck through which the sludge is periodically withdrawn into a mixer where cyanide solution is added.

The slime is then elevated to steel Pachuca tanks by means of two-stage tailing wheels. The agitators are without the



FIG. 1.—WAIHI MILL, WAIHI GOLD MINING COMPANY

developed in New Zealand cyanide practice, which are common to all the mills and distinguish them from others.

The Waihi Mills

The Waihi mine is the largest in New Zealand, and the ore is treated in two large mills. The Waihi, of 90 stamps, and the Waihino, of 200 stamps. At both mills the ore is crushed in water, thus introducing the necessity of dewatering before cyaniding.

At the Waihi mill, shown in Fig. 1, the stamp pulp, containing from 7 to 9 tons of water per ton of ore, is elevated by wheels to spitzkasten, 4 ft. square and 4 ft. deep, two of these being arranged in series for each twenty stamps. The separated sand is reground in tube mills, and after amalgamation and concentration, is classified in other spitzkasten. The sand pulp is run to circular collecting vats previously filled with water. Any slime accompanying the sand flows out of the vats and is returned to the main separating cone. When the sand vats are full they are allowed to drain, after which the sand is leached with cyanide solution.

The slime, 90 per cent to 95 per cent of which will pass a 200-mesh screen, flows to a set of dewatering cones, 5 ft. square and 5 ft. deep, in which the pulp is thickened to a consistency of three or four parts of water to one of solids. This thickened slime is further dewatered in a device peculiar to this plant. It consists of three flat-bottom settling tanks, 32 ft. diameter and 18 ft. deep, provided with annular overflow laun-



FIG. 3.—WAIHINO MILL, WAIHI GOLD MINING COMPANY

customary central lift-pipe, but are found to give good results. The pulp is agitated for about 7 days, with air at 40 lb. pressure. The great length of time required for agitation is mainly to effect solution of the silver.

Moore type vacuum filters are used at both these mills. Their construction suggests an interesting departure from the usual custom, corrugated steel sheets being substituted for the ordinary cocoa-matting center.

Concentrates from these mills are treated by cyanidation after being reground impalpably fine. The pulp is agitated for fourteen days in high, narrow tanks, and then filtered in plate and frame presses. The extraction is about 94 per cent.

Waihi Grand Junction Mill

The present practice at this mill represents an interesting development from the separate treatment of concentrate, sand



FIG. 4.—WAIHI GRAND JUNCTION MILL

and slime, to a system in which all the ore is treated as slime. With this development came the adoption of crushing in cyanide solution, this being one of the few mills in New Zealand to adopt this practice. Preliminary dewatering of the slime is thereby avoided, and the time of treatment is shortened by the more rapid solution of the precious metals during the crushing and grinding process. The mill is shown in Fig. 4, at the extreme right of which can be seen the tall agitators.

The ore treated here is practically the same as at the Waihi mill, and the process consists in crushing by stamps, separating sand from slime in cones, regrounding the sand in tube mills which work in a closed circuit with cones, settling and agitating the slime, and filtering by vacuum. The underflow of the cones is run over the Willey tables formerly used as concen-



FIG. 5.—TALISMAN MILL, KANGAHAKE, NEW ZEALAND

trators when concentrate was treated separately, but now used merely as classifiers to separate some fine sand for return to the tube mills.

The slime separated by the cones contains about 10 parts cyanide solution to one of solids, about 95 per cent of the latter being finer than 200-mesh. It is settled in cone-bottom tanks, 22 ft. diameter, the clear overflow from which is returned into the mill system. The thickened pulp is drawn continuously from the bottom of these settlers, having a consistency of about $1\frac{1}{2}$ parts solution to 1 of solids, and is

pumped to a series of flat-bottomed storage agitators fitted with stirring arms revolving at 4 r.p.m. From these the pulp is pumped to the high agitators, which are without the usual central lift-pipe. Air is used at 38 lb. pressure, and the time of agitation is 18 hours, the latter having been reduced from a period of several days. This is an improvement which may be ascribed, in part, to the practice of crushing in solution. Vacuum filtration in a Moore filter completes the slime treatment.

The Talisman Mill

The principal features in handling slime at this mill are the use of diaphragm cones for classification, and huge spitzkasten for dewatering. The latter may be seen in Fig. 5, at the top of the lower building. The ore is stamped in water, and classified into sand and slime in diaphragm cones, the diaphragm being near the bottom, with a $\frac{1}{2}$ -in. space all around. These discharge a thick pulp, about one to one, which is ground in tube mills, combined with the overflow of the cones, and again classified in a second set of diaphragm cones, the underflow of which is returned to the tubes. The overflow from the second set of cones is amalgamated, whereby from 30 per cent to 35 per cent of the valuable metals is recovered. Concentration on vanners is the next step, the concentrate being shipped to smelters.

The combined vanner tailings run into steel vats, using Butters and Meins distributors. The vats are originally filled with water through which the sand settles, while the slime overflows.



FIG. 6.—KOMATA REEFS MILL

When a vat is full of sand it is drained and leached with cyanide solution. The slime is pumped to the large spitzkasten previously referred to, there being two sets of nine pyramidal boxes; each 6 ft. square and 6 ft. deep. Limewater is added to facilitate settling. The underflow from these spitzkasten passes to rectangular settling tanks, 20 ft. by 10 ft. area and 7 ft. deep, each having two pyramidal hoppers in the bottom, 10 ft. square and 7 ft. deep. The overflow from these passes in succession to two sets of flat-bottom circular vats to effect further settlement before flowing to waste.

Slime for agitation, consisting of water and ore in the ratio of 2:1, is drawn from the settling vats and pumped to tall agitators which, like those at the Waihi, have no central lift-pipe. When an agitator tank is filled within 18 in. of the top, the flow is diverted and the slime allowed to settle for an hour or more. The clear water is decanted, cyanide solution added to form a pulp of one part solids and one part solution, and agitation proceeds for 24 hours. Air is used at 40 lb. pressure. Filtration in a Moore fixed-frame type of filter completes the treatment of slime at this mill. At the present time the Talisman company has a new mill under construction, the operating costs of which are expected to show a considerable reduction in comparison with those of the mill described above.

Komata Reefs Mill

The view of the Komata Reefs mill, Fig. 6, is interesting as showing the first tall air-agitation tanks ever erected for the treatment of ore-slime. These are the two at the left, behind which are two others all resting on the elevated platform.

The other tanks at the right, with continuous sides resting on concrete foundations, were erected later. These agitators are equipped with central lift-pipes, which are an essential part of the original invention of Brown and McKen. Another important contribution to the mechanics of cyanidation originated with Mr. Brown, namely, the iron plate-and-rib liner for tube mills.

The practice at this mill conforms in some respects more nearly to modern ideas about handling slime. The ore is crushed in water and classified in spitzkasten, the sand being reground in tube mills. The tube mill discharge is classified in other spitzkasten, the sand passing to leaching vats, and the slime to Dorr continuous thickeners. The latter continuously discharge a pulp containing less than 40 per cent moisture, which is pumped to the agitators and mixed with cyanide solution to a consistency of about 65 per cent solution and 35 per cent solids. Intermittent charges are agitated for seven or eight days, after which the pulp is drawn off to a Dorr thickener which removes clear cyanide solution and gives a pulp containing about 45 per cent solution, which is finally displaced in a Moore "turnover" type vacuum filter.

Waihi-Paeroa Tailing Mill

The practice at this mill is a good example of what can be accomplished in re-treating tailings from other mills when favorable conditions exist. The various mills at Waihi, Karangahake and Waitakauri discharge tailings into the Ohinemuri River, which has a fall of about 20 ft. per mile to a point about 6 miles above its junction with the Waihi River. Here is a stretch of more nearly level country where the stream deposits its burden of sand, while the slime is carried on to sea. It has been estimated that the mills on the upper reaches of the river have crushed from 5,000,000 to 6,000,000 tons of ore, and have discharged this quantity of tailings into the stream. The sand has accumulated in the natural settling basin referred to; and an estimate of the quantity of such material deposited is about 2,000,000 tons. To this must be added the current annual contribution of the upper mills, amounting to about 500,000 tons of sand and slime.

The project of the Waihi-Paeroa Gold Extraction Company, Ltd., involved dredging this sand from the river bed, grinding in tube mills in cyanide solution, thickening, agitating and filtering the slime thus produced. This has been successfully carried out in a 500-ton mill.

The dredging machinery, consisting of oil engine, air compressor, centrifugal pump and air-lift, is mounted on a pontoon. The lift-pipe is telescopic, and can be lowered to any desired depth. The centrifugal pump is used to force water through a series of jets near the bottom of the air-lift to thin the sand to the desired consistency for pumping by compressed air. The sand is pumped into barges and towed to the wharf.

Charcoal and river sand are removed by treatment on vanners, after which the valuable residue is ground in tube mills working in a closed circuit with classifiers. The tube mill product is all finer than 150-mesh. This is elevated by two-stage tailing wheels to large settling cones from which the thickened pulp is discharged at a consistency of about 11 to 1. It is pumped to tall air-agitators which are without central lift-pipes, and agitated for forty-eight hours.

The Moore type of vacuum filter is used, the tanks being

circular and cone-bottomed. One tank is used for loading and one on either side of this for washing and discharging respectively. A disintegrator is placed in the bottom of the discharging tank, for the purpose of churning the caked slime with water before finally discharging it to waste.

Blackwater Mines Reduction Works

Stamping in cyanide solution is practiced at this mill, which is illustrated in Fig. 7. The view shows plainly the tall agitation tanks, dewatering cones, Moore filter and sand vats. The

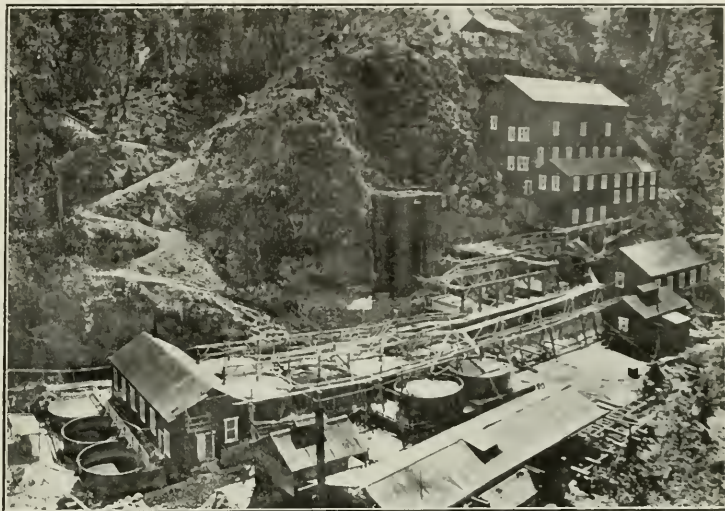


FIG. 7.—CYANIDE MILL, BLACK WATER MINES

dewatering cones and Moore filter tanks are of wood; the agitators have no central lift-pipe.

The first separation of the stamp pulp into sand and slime is made in the leaching vats, the overflowing slime and solution being pumped back to a series of large wooden dewatering cones. The latter thicken the pulp to a consistency of about two to one, after which it is agitated. Vacuum filtration completes the slime treatment.

Summary

The characteristic features of New Zealand practice in handling slime may be summed up as follows: Crushing in water prevails at most of the mills. Intermittent or charge agitation is customary, the continuous method having been tried and abandoned on account of the tendency of the coarse solids to build up in the tanks. All New Zealand mills use the tall air-agitator of Brown and McKen, but without the central lift-pipe, except at the Komata Reefs mill, where the agitator was invented and first used. It is generally claimed that the use of the central tubes give no better results; and as thus apparently the only feature of the agitator covered by patent it is generally omitted. High air pressure, 40 lb. per square inch, is customary, being about double that used in America.

Tailing wheels still hold their own as elevators in New Zealand mills. The last to be erected are at the Waihi-Paeroa, where a two-stage lift is adopted. The Brown ribbed liner for tube mills, which is being more generally adopted in other countries, was originated in New Zealand, and is generally used there. The Moore filter has a strong hold, all the mills of any consequence using it. One of the peculiarities of construction of filter leaves is the substitution of corrugated sheet steel for cocoa matting in the center. It is claimed that this type of leaf filters just as well as the other, is lighter and dries more quickly; also that it is more readily cleaned if leaks occur in the outer canvas.

Denver, Col

Carbon Electrodes for Electrolytic Cells

By John Hårdén

A good deal of artificial carbon electrodes are to-day used for electrolytic work, and the subject of their manufacture is still rather important, although it may perhaps be right to say that the bulk of carbons made at present and more so in the future are used for electrothermal purposes. A few data regarding the "electrolytic carbons" may, therefore, be of some interest.

In the United States the graphitized electrode, first introduced on a larger scale by Dr. Acheson, whose name has rightly become famous in connection with this industry, has gained a well-deserved market. In Europe these electrodes have also obtained due recognition, but here the conditions are considerably different so that common amorphous electrodes are still to a considerable extent in use.

This may be explained by relating the writer's experience in some electrolytic works in Russia some years ago. These works produced on a large scale chlorine and caustic soda from ordinary sodium chloride obtained from the Black Sea and from the Caspian. Having been engaged previously as head of the testing department of a large carbon electrode factory, the writer was given ample opportunity to compare and to judge the particular "pro's and contra's" in question.

Since a heavy duty was levied on the finished article, it was found uneconomical to import the electrodes from abroad, and no factory was in existence in the country itself. The preparing of the raw material, however, is of great importance, as will be seen later. Therefore the ready prepared mass was obtained from a foreign carbon factory on the other side of the frontier, transshipped as raw material, consequently free of duty, and pressed and calcined into proper electrodes at the works. This realized a considerable saving and made the large outlay of a complete electrode plant with expensive machinery and large stock of materials unnecessary.

In spite of this, the cost of electrodes was a heavy toll on the works' budget. The cells consisted of an earthenware top in which seven round electrodes, about 4 inches in diameter by 2 feet 6 inches in length, were fastened. Diaphragms of asbestos separated the two compartments, anode and cathode room, and as the electrodes were worn off, the whole set of cells, six in each batch, had to be dismantled when repairs were to be made.

In order to lengthen the life of the cell, some graphitized electrodes were tried, with partial success as was expected. In those days, however, such electrodes imported from U. S. A. became very expensive, not only on account of their initial considerably higher price, but the exorbitant freight and unavoidable loss in breakage. It was generally found that the outlay for such electrodes, under the prevailing circumstances, was not in any way justified by their longer life, since the local labor was so inexpensive that it did not come into play, and the saving in material, although decidedly marked was not sufficient to compensate the other outlay.

This state of affairs is no doubt responsible for the fact that non-graphitized are still being used to a great extent, notwithstanding that the graphitized material is superior as regards actual wear and tear. In order, however, to test the matter more fully, the firm decided to graphitize its own electrodes on the spot, thereby utilizing a method worked out somewhat earlier by the author, and which differed in various respects from the Niagara practice. Some of its points have been briefly described in an article in *METALLURGICAL AND CHEMICAL ENGINEERING*, vol. IX, p. 130, March, 1911, headed "Some Electric Furnace Notes." A furnace was built and put into operation and after a few failures and alterations we succeeded in obtaining a continuous working, producing some 12 to 16 graphitized electrodes per day.

But even then, the economy was very problematic. Although the cost of power was not excessive, since it was produced by means of cheap local coal, the saving generally was not so great as at first expected. The electrodes prepared in this

way, though fully graphitized, showed a marked tendency to scale off in the bath, an experience, by the way, which has been since made also in electric furnace work, when the higher conductivity of the graphitized carbon has been made use of in order to increase the current density, thus decreasing the running costs of electrodes generally.

The graphite scales thrown off lodged in the bottom of the cell and clogged this up to such an extent, that it had to be opened and cleaned long before the electrodes were worn out, thus rendering the apparent saving illusive. This was due, to a great extent, probably to the fact that the material from which the electrodes were made was not quite suitable for this particular purpose; if a carbon factory proper had been connected with the works, it is quite possible that the fault may have been remedied to a greater extent. This problem of scaling off is one which the manufacturers of graphitized carbons should bear in mind; a material which, when graphitized may serve very well for one purpose may be quite unsuitable for another; in this case probably because every trace of the carbonaceous binder was thoroughly burnt out and converted into graphite as well as the main body, a fact which was fully shown by means of microphotographs, taken by the author and published in *Zeit. f. Electrochem.*, 1901, p. 324, and also by Sproesster in No. 71 of the same journal.

Large grains with adequate binding material, pressed under moderate force, which gives a very good result for furnace electrodes when used as amorphous electrodes, showed this scaling off in a marked degree when graphitized, although the porosity generally is much lessened. Finer grain, with less binder, stamped hard, calcined and graphitized, showed a better result in this respect, though the resistancy against the chemical action of the electrolyte is then generally lessened.

For the same reason it was also found that material which had been previously graphitized and then ground to suitable size, gave with an adequate binder of tar and high pressure as well as good calcination a comparatively better result. The economy of this working, however, was less satisfactory.

The Testing of the Electrodes for Electrolysis

The testing of the electrodes is of great importance, both for the manufacturer and the user. For the former, as it will tell him about the raw material and its preparation and also the subsequent manufacture so as to avoid complaints, etc.; for the latter to make sure that he has the right kind of material, since a fault may cost him a good deal if it is found out only when the cells have been built in. Generally, it should be emphasized that an electrode should be tested in the solution and under conditions as near as possible to those in actual use.

Preliminary, the following observations should be made:

- (1) Homogeneity and grain.
- (2) Absence of cracks and flaws.
- (3) Porosity and specific weight.
- (4) Contents of inert material.
- (5) Electric conductivity, and last but not least—
- (6) Resistivity to corrosion.

(1) Homogeneity and grain should give an idea of the uniformity of the raw material; if too fine and of a deep black luster, too much dust and "lampblack" material has been used. Such an electrode is likely to be less strong mechanically and will often cause larger lumps to drop off in the cells. A grey structure with light, shining points or globes indicate the presence of a large quantity of petroleum-coke, which if used in excess may behave similar to the former kind.

As a rule, the structure of a good, durable, amorphous electrode should be of even grain, not too fine, the grains being well and evenly united by the binder of a dark greyish color. Each particle of the mass should be thoroughly enveloped in a film of the cement or binder before pressing, and it is this part of the preparation of the raw material which calls for the utmost care and attention, and which, besides the choice of the material and the calcining, is the mainstay of the durability of the electrode.

Ingenious machines, the description of which space forbids,

have been invented and put into use to produce this effect, and money spent on this by the manufacturer is well invested.

(2) The absence of cracks and flaws is of course equally essential for the same reasons. These may be caused (and are usually caused) by inadequate treatment of the raw material, but also through false methods of calcining. The firing usually takes place in a gas-fired "ring" furnace, in which the electrodes are placed, embedded in coke dust, etc. If this packing is uneven, the electrode will "hang" and edge cracks are the result.

Such cracks are bad enough in electrodes for electrolytic work, since the increased resistance of the carbon may cause a poor distribution of the current and also cause the faulty electrode to drop off, but they are distinctly bad in the case of furnace electrodes, where a broken electrode may mean a whole smelt to be spoiled. Such occurrences are now-a-days rather rare, where proper "ring" kilns for calcining are employed, but a less scrupulous foreman may cause faulty products even with the best kilns.

A good, sound amorphous carbon should, when hanging free in a sling, and tapped with a steel bar, give a clear, distinct ring like a bar of hard cast iron, and endure nearly the same rough handling as such. A graphitized carbon again should give a deep "thud," yet clear and sound, be capable of polish by means of a brandishing iron and be soft enough to be cut with a knife, while the amorphous carbon is hard and should not permit cutting with other tools than the chisel and emery wheel.

(3) The porosity and specific weight are also both of importance. A certain amount of porosity is of advantage, as it will increase the active surface of the carbon; but if in excess, the bath liquid will enter the pores and cause a rapid development of gas which tends to break off small chips, especially at the less porous edges and thus shorten mechanically the life of the electrode. A good practice, gained by comparing a new carbon of unknown merit with a test piece of a well-known quality, whereby a medium strong magnifying glass should be used, will be found helpful. The pores should be even and not too marked, resembling those on a hard sugar loaf, the calcined binder causing the surface to appear slightly glazed.

The general way in which the specific gravity is determined may be shortly described.

A piece of the carbon, not too small, preferably cut out to regular size of a few cubic centimeters, is boiled in alcohol, dried in the oven, where it may cool and the weight is taken (*a*). Its volume (*v*) is then determined in a measure, the water is wiped off quick and the wet weight (*b*) is taken. The mean specific gravity is then $\frac{a}{b}$ and the real specific gravity is

$$\frac{a}{v - (b - a)}$$

It may also be determined by means of the method of suitable heavy liquids, such as a mixture of bromoform and chloroform of known specific gravity.¹

The relative specific gravity varies between 1.4 to 1.6 while the real specific gravity is generally between 1.6 to 1.8, and the porosity $\frac{100(a-b)}{a}$ is ranging from 11 to 22.

(4) With inert material is understood such contaminations which do not act as electrode material proper, but may be due to interior purification of the raw material. This is determined through ash-analysis, i.e., combustion in oxygen and air, whereby both the CO₂ evolved and the remaining ash should be weighed. Good carbons made from petroleum coke and lampblack have usually only about 0.3 to 0.5 per cent of ash, while anthracite electrodes may have 3 to 4 per cent.

Acheson graphite electrodes show as a rule a very low content of ash, due to the high temperature at which they have been treated. Electrodes for electrolysis ought not to have more than 2.5 to 3 per cent ash, while 4 to 5 per cent, as is sometimes found, is distinctly disadvantageous. The

carbons for electric furnace work should suit the conditions for which they are to be used, i.e., for carbide furnaces a content of 3-4 per cent lime is quite tolerable, for steel furnaces 1 to 3 per cent iron, etc., but the life of the electrode will suffer from a higher content of these substances.

The microscopic examination may also be of considerable use for determining the quality with a skilled observer, but it would carry us too far to enter further into this without the reproduction of the various sections. The author has taken a number of microphotographs of carbons manufactured in different ways and he may state as a general rule that the microphotograph will show clearly if rapid consumption of the carbons is due to insufficient calcination of the binder or if the main raw material was of an unsuitable nature. Again in this respect, the graphitized electrode gives a distinctly superior result, as the sections taken prove that practically all of the binder is converted into graphite carbon.

(5) The electric conductivity is naturally of considerable importance, for the more voltage we must apply to a cell, the greater is the dead cost of the electrolysis. That this is a weighty question is obvious, if we remember that a cell is usually operated with several thousands of amperes and a slight increase in resistance will soon run up the bill. This is also, though not to such a degree, of importance in furnace work, as may be clear from the following experience of the author.

A set of metallurgical furnaces was operated with carbons of 14 in. diameter at 4000 amp. The stumps were screwed on to the new electrodes by means of threaded plugs and graphite paste. A firm offered a brand of new carbons for which they claimed superior quality at a somewhat lower price than the usual cost of amorphous electrodes. The average resistance of a new carbon not in use was found to be within the proper limits, and a couple of the carbons were put into use as a test.

After a few runs it was found that the carbon scaled off in a remarkable way and one of the electrodes, having been jointed in the meantime, became red hot in the joint, in spite of the fact that the joining was made with special care and with the material specified by the maker. The voltage drop in a 5-ft. unjointed and in a 6-ft. jointed was taken and found to be 4.8 volts in the former and 6.9 volts in the latter at 4000 amp., consequently 19.2 kilowatts were wasted in the former and 27.6 kilowatts in the latter which, of course, made the use of these electrodes an impossibility, since the reduction in initial cost was in no way equal to the loss in power. It may be added that inductance and leakage was duly considered, in any respect it was of little importance, since the periodicity was fairly low, only 22.5 per cent.

The specific resistance is

$$\text{Sp. R.} = \frac{r q}{l}$$

where *r* is the measured resistance of a strip in ohms, *q* is the cross-section in square millimeters and *l* the length in meters. It ranges usually for amorphous carbons from 58 to 75, for graphitized electrodes from 6 to 12 and for charcoal electrodes up to 150.

As it is, of course, of great importance that the fastenings of the carbons in an electrolytic cell have as low resistance as possible, it is found advantageous to provide the ends of the carbon slabs with a good coating of graphite paste in all such cases where graphitized carbons cannot be used for one reason or other. The author prefers, where the conditions of the cell permits, to cast a shoe around the upper end of the carbon bar of a type of metal which will contract strongly over the carbon, yet be soft enough to avoid an undue strain on the bar. An alloy of lead, antimony, and bismuth in proportions tested for different brands of carbons has been found very suitable, and as only a thin coating is required, the expense is amply justified. Copper plating is usually found obnoxious, even if the nature of the electrolyte is such as to not corrode the copper, since it is found impossible to eliminate all traces of the acid of the copper bath from the pores of the carbon and the plating will invariably peel off and cause poor contact in time.

¹Z. f. El. Chem., 1901, p. 973.

(6) The resistivity to corrosion is the vital point in question, as has already been mentioned. If a carbon is used as anode in a cell, the carbon-block is invariably disintegrated, more or less, depending upon the nature of the carbon and also of the solution forming the electrolyte. Some of the products, in the case of an alkali cell are gaseous, some go into solution and the bulk is setting off as a slime. Oxygen-giving solutions are more active in this respect, but chlorine is also an enemy to the electrode.

If amorphous carbons are used, one may frequently find a product formed called mellogen, which may be characterized by the formula $C_{12}H_2O_4$, which is soluble in alkali and in concentrated sulphuric acid. This will be changed into mellithic acid ($C_{12}H_2O_{12}$) at ordinary temperatures. The same may be observed in using retort carbon, while graphitized electrodes² will form graphitic acid $C_7H_4O_5$.

The most reliable way of testing the carbons in this respect is to demand from the maker two or three pieces cut out from the electrodes he intends to deliver and subject these to electrolytic test under such conditions as those prevailing in an ordinary cell, i.e., in a salt solution for chlorine-alkali electrolysis, for instance.

The author used for factory testing the following arrangement: 6 to 12 cells consisting of alkali- and acid-proof enamelled cast-iron vessels were connected in series, the cathodes being sheet iron bent into suitable shape. The carbon anode was suspended in a chamber made of earthenware, provided with asbestos diaphragm and open at the bottom. The solution was 20 per cent sodium chloride, which was kept constant, both to concentration, volume and temperature. The current density was also held constantly at 0.1 ampere per square centimeter by means of meters and regulating device.

A clock was so arranged that a series magnet would permit the clock to go as long as the cells were in operation, but arrested the work automatically when the current was shut off from the cells; in this way an absolutely reliable time record of the duration of the test was obtainable.

The gases evolved in the anode chamber were sucked through a weighed potassium hydrate solution, in order to determine the quantity of carbonic acid produced.

The carbons were cut into regular shape and size, $3 \times 1 \times 10$ cm, brushed, boiled in water, dried in warm alcohol and hot air, and, after the connection lead had been cast on, carefully weighed. It was then immersed in the cell, a soft rubber packing being used to exclude leakage of gases.

The current was then turned on and allowed to act during a given time, usually 48 hours. The electrode was taken out at regular intervals of 12 hours, boiled in water and dried in alcohol and weighed as before. In this way a curve was plotted, showing the deterioration as it proceeded, and the life of the carbon could in this way be easily predicted, if compared with the curve of a standard electrode of well-known wearing quality.

It was very interesting to observe how in many cases the curve was at first very flat, showing a slight wear only, but as the test proceeded the curve rose rapidly, owing to the fact that the outer layers were more calcined than the inner, also showing greater porosity, whereby the carbon particles were thrown off more readily. By comparing the voltage required by the cell during the test one could very well foretell when it would be more economical to exchange the carbons, even if they were not worn down totally, similar to the critical point of a carbon filament lamp.

One might object that this test was more destructive than the actual conditions in a regular cell, and therefore not conclusive. This, however, is counteracted by the comparison with the standard carbon, tried before in actual use and from time to time submitted to a blank test in series with the new specimens to be tested. The variations observed were so slight, that during the $3\frac{1}{2}$ years the author used this method for determining the wearing quality of such carbons, the variations could

very well be neglected and the test was found sufficiently accurate for decision, both for manufacturing and ordering.

A few figures gathered by the author may serve to give an idea of the losses under above conditions: the figures are given in grams per 10 amperes per hour, 20 per cent salt solution and 0.1 ampere per square centimeter density.

1. Conradty	5.17
2. Le Carbon	2.80 (Graphitized)
3. Plania works	1.90 (special composition)
4. Schiff & Co. "X"	5.65
5. Retort Carbon	2.92
6. Acheson Graphite	1.06
7. Author's Graphite	2.02

As will be seen from the above, the graphitized electrode is generally superior to the ordinary carbon, but, as already stated, local conditions at the works may under circumstances be such, that one has to use ungraphitized electrodes on account of costs, etc.

The specimen No. 3 shows that it is also possible to produce an amorphous carbon with a wearing quality practically as good as the graphitized one, but it is only just to add, that, owing to its special nature, the price of this was almost the same as a graphitized one. Apart from this, the specific resistance was about eight times higher, and the material very hard and brittle, wherefore it could not be recommended as a good substitute for the graphitized brand.

It is quite clear from the above, that the judging of the merits of a good carbon for electrolytic work requires a good deal of consideration and patience, if a fully economical working result is to be arrived at. It should be borne in mind that the general points set forth above do not by any means exhaust the subject of the testing, but they will serve to give an idea of the faults and merits to bear in view.

Luton, England.

The Fourth International Congress on School Hygiene will be held in Buffalo, N. Y., from Aug. 25 to 30, 1913, under the patronage of Woodrow Wilson, President of the United States. Delegates will attend from all the leading nations, from every college and university of note in this country, and from various other educational, scientific, medical and hygienic institutions and organizations. The congress is further open to all persons interested in school hygiene. Membership may be secured on the payment of a \$5 fee. Applications should be sent to Dr. Thomas A. Storey, College of the City of New York, New York City.

The American Metal Company, of New York City, has purchased 2500 acres of Pittsburgh coal lands and 350 of surface land at Purgettstown. On this surface land the company will commence to build a zinc and sulphuric acid plant. A subsidiary company, known as the American Zinc & Chemical Company, is being organized with a Pennsylvania charter to operate the new plant, and Mr. N. L. Heinz, of LaSalle, Ill., one of the best-known zinc and sulphuric acid experts in the United States, will be the general manager and has been made a member of the board of directors. What is known as a one-unit plant will be erected and will have a capacity for producing 40,000 tons of sulphuric acid and 20,000 tons of zinc annually. The plant will be so designed that it can be enlarged rapidly, and it is the intention to build additional units until the annual capacity amounts to three or four times the original.

A shaft will be sunk and coal mining operations will be commenced coincident with the opening of the plant. The initial investment in this new plant will be not less than \$2,000,000, and it is expected that when the entire plant is completed and in full operation, which will probably be within three years, it will represent an investment of approximately \$5,000,000.

The new plant will give employment to 500 men at the start and as the plant is enlarged this number will be increased until about 1500 men will be employed.

²See Zellner, Künstliche Kohlen.

A Modification of the Jäger Method of Gas Analysis

By S. H. Worrell

In the November, 1911, number of METALLURGICAL AND CHEMICAL ENGINEERING, there was an article on a simple method of determining methane in producer gas and enriched water gas. The method was that made use of in ordinary combustion analysis of carbon compounds; namely, the use of copper oxide as a source of oxygen. The method is as old as Fresenius, of course, who absorbed the products of combustion and weighed them. In the article just referred to I made the statement that a fairly thorough search of the chemical journals failed to reveal anything published on the subject. I might have added also that the method was not to be found in any text book on the subject of gas analysis that is at hand in our library. Shortly after the article appeared, in looking up another reference, I found that E. Jäger had made use of the method and published it in *Journal für Gasbeleuchtung* in 1898. However, he went a step further, and by first heating the tube containing copper oxide to 250 deg. C., he removed the hydrogen, and then by igniting to redness burned the methane. His method in a modified form was at once put to use in this laboratory.

It was difficult to understand at first why a method so simple and seemingly so meritorious should not be in more general use or at least have found a place in textbooks on the subject of gas analysis. The reason is probably twofold. In the first place, Jäger's method of placing the KOH pipette beyond the copper oxide tube may be a source of error as will be shown later, and then the method of standardizing is tedious.

Unsatisfactory Methods for Carbon Monoxide and Hydrogen

In gas analysis, the absorption of carbon dioxide with KOH, and thereby the determination of its percentage, is a simple matter. The same is true with reference to the determination of the illuminants (unsaturated compounds) with bromine water, and oxygen with stick phosphorus. However, the absorption of CO with cuprous chloride, and hydrogen with palladium tube is sometimes less successful. At least the results obtained in the absorption of hydrogen with the palladium tube were never very satisfactory in the work in this laboratory extending over quite a number of analyses of a series of coal gas samples. Burning both the hydrogen and methane together was resorted to for awhile; the loss in volume being hydrogen which when burned condenses to water. But this method is open to the objection that there is no check on the carbon monoxide absorption with cuprous chloride; for, the CO present, if there is any, is burned to carbon dioxide and is absorbed and measured along with that due to methane. Furthermore, if there is any ethane present, which is possible, it increases the residue in volume when burned and thus decreases the apparent percentage of hydrogen. However, with Jäger's method, the hydrogen is first removed. Then when the residue from the hydrogen is burned, the increase if any in volume is due to ethane. In this way the ethane can readily be determined.

Of course no distinction is made in this case between ethane and any higher member of the series, which if present would be measured as ethane.

In using Jäger's method, a pipette containing water slightly acid with sulphuric acid, was always used instead of KOH as he directs. It was soon observed that the residue left after burning out the hydrogen at 250 deg. C., frequently contained a small quantity of carbon dioxide. At first it was thought that this small quantity of CO₂ was due to a partial combustion of methane, caused perhaps by its reacting with a small quantity of copper oxide dust that might be present, a very high temperature perhaps not being necessary to bring this about. However, prolonged heating at 250 deg. C., and repeatedly passing the gas did not cause the formation of

more carbon dioxide. As a matter of fact, this carbon dioxide when present was due to the burning of the small residue of carbon monoxide which had failed of absorption with the cuprous chloride, either through lack of thorough shaking or too weak a solution.

As a result of this observation, the cuprous chloride pipette has been discarded entirely, and after removing the oxygen with stick phosphorus, the procedure is simply to burn the hydrogen to water and the total of the carbon monoxide to carbon dioxide in one operation at a temperature of 250 deg. The loss in volume is the percentage of hydrogen present. The CO₂ formed is then absorbed by passing into KOH and the loss in volume is the carbon monoxide that was pres-

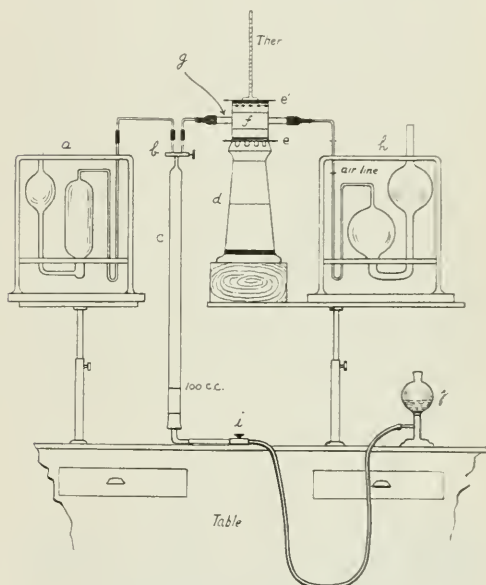


FIG. 1.—APPARATUS FOR GAS ANALYSIS

ent. The copper oxide is then heated to redness and the residual gas passed. The volume of gas is noted before and after burning. If there is an increase in volume it is due to ethane, and one half this increase is the percentage of ethane present. The total carbon dioxide formed is then absorbed and the residue noted. The loss in volume, accounting for the ethane first, is the percentage of methane present. The residue is the percentage of nitrogen direct.

Description of Apparatus

There follows a description of the apparatus necessary and a detailed statement of the method of using it with the precautions to be taken.

For the method in detail, reference is made to the accompanying illustration. All glass connections are of capillary tubing and the rubber should be thick-walled, tight-fitting. Referring to Fig. 1:

- a—Semi-permanently attached KOH pipette (500 gms. to the liter).
- b—Two-way stopcock.
- c—Gas burette.
- d—Cradlock burner for convenience. Any burner would do of course.
- e—Asbestos board, 1 1/2 in. in thickness, solid.
- e'—Asbestos board, 1 1/2 in. in thickness, with a hole for a thermometer.
- f—A 4-in. asbestos air bath (Eimer & Amend Catalogue No. 2373).
- g—Silica tube, 1/4 in. bore, 7 in. long.

h—Pipette filled with water, slightly acid with sulphuric acid.

i—One-way stopcock.

j—Leveling bulb filled with 1% to 2% sulphuric acid.

Method of Procedure

The solution of KOH in *a* having previously been drawn over nearly to *b*, the stopcock is closed to *a* and opened to the air. Before the portion of the apparatus shown from *d* to *h* is placed in position, the burette is filled with gas to be analysed until the water sinks about one inch below the 100 cc. mark. The stopcock *b* is then closed and the leveling bulb *j* raised until the gas under pressure has a volume of 100 cc. The stopcock *i* is then closed and *b* opened, whereby the gas is brought under atmospheric pressure and exactly 100 cc. has been taken. The gas is then allowed to pass into *a*, when its carbon dioxide is removed. The water in the burette is now caused to flow out to the air until the entire system including the connection of capillary tubing is filled with water. The bromine pipette containing water saturated with bromine to which a few crystals of KBr have been added is then attached on the right. The gas, returned from *a* to *c*, and back and forth a few times if necessary, is measured by bringing the liquid in the leveling bulb *j* to a level with the water in the burette. Stopcock *i* is closed and the burette is read. The loss in volume is the percentage of carbon dioxide present.

The gas is then passed back and forth from *c* to the bromine pipette until bromine fumes are distinctly visible in the gas on returning to *c*. It is then passed into *a* to remove these fumes and upon being brought back to *c* is measured. The loss in volume is the percentage of "illuminants" (unsaturated carbon compounds) that were present.

The phosphorus bulb is then substituted for the bromine bulb and the oxygen determined in a similar manner; that is, by loss in volume.

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Precautionary Measures

The apparatus as shown in the illustration is then placed in position. Care must be taken to have the right hand opening of the two-way stopcock *b*, as well as the capillary tubing leading from it, entirely free of KOH solution. In fact, *not one drop of KOH should ever be allowed to enter the burette* if it is possible to avoid it, for the reason that one drop can absorb many times its volume of CO_2 , causing an error in the hydrogen determination. If any KOH has entered, the gas should be passed into *a* after the oxygen is determined, and the passage way to the air thoroughly washed with the acid solution in *c* and *j* until all KOH is removed or neutralized.

The apparatus having been placed in position as shown, the air line on the capillary tube of *h* is noted. With a little care it can always be brought to the same place. The burner is then lighted and turned down until the bottom of the board *c* glows in one spot in the center from 1 to 2 inches in width. This will generally be found sufficient to bring the temperature of the air in the bath to 250 deg. C. which is the required temperature; from 250 to 260 deg. is sufficient. The silica tube in the air bath is filled in its central portion, about 4 in., with copper oxide, "wire form," crushed until it will pass a 20-mesh screen. The dust formed in crushing should be screened out and none passing 80-mesh should be placed in the tube since it clogs the passage of the gas. The remainder of the tube is filled with an asbestos fiber plug, about one-half inch on each side of the oxide, and then glass wool to each end of the tube. The glass wool is not necessary, except to make the air content smaller, and to help retard the passage of the water if it should by accident be allowed to enter the silica tube on either side. Care should be taken *not to pack* the copper oxide or it will expand on heating and crack the tubing. On the other hand it should fill the tubing entirely if possible, to prevent an empty passage along the top of the

tube, and so permit the gas to pass without coming into intimate contact with the copper oxide. A simple way to accomplish this is to place one plug of asbestos in position, then pour in the oxide and tap it gently with the finger to settle it. Then the other plug can be gently pushed into place. The proper place can be seen by holding the tube to the light.

Advantage of Silica Tube

In the former article alluded to above, a hard glass tube was recommended. It was soon discarded in this laboratory for the silica tube which does not bend on heating repeatedly, being more refractory; and, what is of more importance, it does not break if water is allowed to enter it from either side or is condensed from the water formed in combustion. The rubber tube connections to the capillary elbows at either end are of thick-wall "antimony" tubing. No difficulty is experienced from this tubing being burned, as it is protected by the asbestos air bath, through the walls of which the silica tube passes by means of two holes on opposite sides, just large enough to admit the tube and so placed that the center of the tube is about the center of the bath.

The temperature of the air bath having risen to from 250 to 260 deg. C. as shown by the thermometer, the leveling bulb *j* is placed on a shelf above the level of *f* and the cock *i* opened slightly to allow the gas to pass slowly through *g*. It must be passed back and forth until it no longer loses in volume. This means that it will have to pass through the silica tube six to eight times. Then, the light being extinguished, and the thermometer and *e* being removed, the apparatus soon cools to the temperature of the room. Then the gas is returned entirely to *c* until the water rises in the capillary tube of *h* to the former air line. The cock is then closed and the loss in volume noted. This loss in volume is the hydrogen present in the gas plus the oxygen of the air in the silica tube. All this oxygen has been used up of course. This volume of oxygen has to be subtracted from the loss in volume due to the hydrogen burning, and the difference is the percentage of hydrogen in the gas. (A simple way of determining the air volume of the silica tube or rather of its oxygen content will be given further on.) The residual gas is then passed into bulb *a* and the carbon dioxide formed by the combustion of the CO while heating to 250 deg. is then measured directly by the loss in volume due to the absorption of the carbon dioxide formed. To accomplish this thoroughly, the residue in *c* after being deprived of its CO_2 content is again passed through the silica tube, which of course contains CO_2 , and then the whole is again passed into *a*. By repeating this fractional removal three or four times, the CO_2 is reduced to an amount too small to be measured.

The reaction will be found to be complete at the temperature mentioned, provided the gas has been passed a sufficient number of times to burn the hydrogen completely. This can easily be determined as noted above, by passing it back and forth until it no longer loses in volume.

The asbestos board *e* is now removed, the burner again lighted and the tube heated to redness. It may be found necessary to place a single sheet of iron-wire gauze (the sort usually placed under beakers) on top of the bath to heat the tube sufficiently. But this is to be avoided if possible because it may cause heat enough to scorch the rubber connections.

The gas should be passed back and forth about the same number of times as was found necessary in the case of the hydrogen, but in this case can be passed much faster. The light is extinguished, the apparatus allowed to cool and the volume noted. It should be the same as before burning, or at least no less. If less, it shows that the hydrogen and possibly the CO were not completely burned, or that KOH was allowed to enter the burette and up into the capillary tube leading to the silica tube. Any increase in volume is due to ethane or a higher member of the series. This increase in volume, if due to ethane alone, may be read at once as the percentage of ethane. The gas is again passed into *a*, the CO_2 removed and the residue measured. The loss in volume is the percentage of

methane if there was no ethane. If ethane was present, then to determine the amount of methane, the percentage of ethane is doubled and subtracted from the total amount of carbon dioxide formed. For example suppose a mixture of the following gases in the residue: 8cc. of nitrogen, 20cc. of methane and 7cc. ethane, with a total volume of 35cc. Upon burning, the nitrogen, of course, is unaffected; the methane burns to CO_2 but has the same volume as before, but ethane doubles its volume on burning. The total volume will be 42cc.: 42-35 gives 7, the percentage of methane. When passed into KOH, the loss in volume will be 34cc.: 34 less 14 (double the percentage of ethane) gives 20, the percentage of methane. In removing the carbon dioxide, the same fractional method has to be resorted to as in the case of the carbon monoxide determination mentioned above. The sum of the various constituents determined thus far, is subtracted from 100 and the residue is nitrogen; or, the residue after removing the last of the carbon dioxide is the nitrogen percentage when the oxygen content of the silica tube has been added to it.

Correction for Oxygen in Air in Silica Tube

To "standardize" the silica tube and all that portion of the apparatus between *b* and the air line in *h*, the following procedure will be found convenient: Take any gas, such as average city water-gas or coal-gas, known to contain methane to the extent of say 5%, but no ethane, (no sample of gas has been analyzed in this laboratory in the past year that contained more than a "trace" of ethane) and remove the carbon monoxide and hydrogen by prolonged heating at 250 deg. as outlined above. The residue of methane and nitrogen is passed into *a*. The bulb *h* is removed, the burner again lighted, and the cuprous oxide and copper are again oxidized to black oxide of copper by repeatedly passing air in and out. The apparatus is then allowed to cool to the temperature of the room again, and air is once more passed in and out to be sure that the silica tube contains air only. The bulb *h* is then attached as before and the residue of methane and nitrogen burned as outlined above, by passing the gas back and forth two to four times. It is then allowed to cool, returned to the burette and the loss in volume carefully noted. This loss in volume is that due to the oxygen of the air content of the silica tube and accessories in place. This then is the amount to be deducted from the hydrogen percentage and added to the residual nitrogen to determine the percentage of nitrogen. The copper oxide in the silica tube can be reoxidized and used over and over again by passing the air as just described. After it has been used many times, however, it becomes more or less solid on the bottom. It then has to be punched out or dissolved out with acid and renewed. Incomplete combustion of the hydrogen on repeatedly passing it through, marks the stage when renewal is necessary.

Comparative Results

Following are the results obtained in a number of analyses by this method alone, and when combined with the cuprous chloride method.

Gas No. 1.			
1st sample		2nd sample	
CO removed with CuCl	4.6	CO removed with CuO only.....	14.0
CO removed with CuO	9.6		
14.2			
Gas No. 2.			
1st sample		2nd sample	
CO removed with CuCl	11.2	CO removed with CuO only.....	21.9
CO removed with CuO	10.6		
21.8			
Gas No. 3.			
1st sample		2nd sample	
All CO removed by CuO	16.8	All CO removed by CuO	17.00
Gas No. 4.			
1st sample		2nd sample	
All CO removed by CuO	15.5	All CO removed by CuCl	15.0
Gas No. 5.			
1st sample		2nd sample	
Carbon dioxide	0.2	0.2	
Hydrogen	2.0	2.0	
Oxygen	1.4	1.3	
Hydrogen	45.1	45.1	
Carbon monoxide	19.3	19.4	
Methane	24.4	24.6	
Residue	7.6	7.4	

The results given under No. 5 are those obtained in the course of the analysis, without applying the correction for the oxygen in the silica tube. To do this and obtain the correct results, 1.3 must be taken from the hydrogen in each case and added to the nitrogen.

In conclusion, attention may be called to the fact that in the Jäger method as outlined in the reference, if the CO was not completely removed it would place an error on the hydrogen due to the use of a KOH pipette where bulb *j* is placed in the accompanying illustration. The method of calibrating the silica tube as outlined is simple and accurate.

It may be that the determination of carbon monoxide by this method has already been done by someone else, but if so I have not seen its publication. At any rate, it has not found its way into the various textbooks on gas analysis. After repeatedly experimenting with this method for the past two years, the writer has a decided preference for the method as outlined above over the cuprous chloride, palladium oxide and explosion method, both from the standpoint of accuracy and also of convenience.

Chemical Laboratory of the Bureau of Economic Geology and Technology, University of Texas, Austin.

Utilization of Wood Waste—III STEAM DISTILLATION

By John E. Teeple, Ph.D.

The distillation of wood waste with steam, to recover turpentine and pine oil was at one time quite an industry, and what is more there were certain periods in its history when it was fairly profitable. The rapid development began about 1902 and reached its maximum in 1910 when a very considerable number of plants were in operation. Almost every plant operating during 1910 was able to show a profit for that year and the first few months of 1911. The fall in price of turpentine since then has made the business unprofitable. There is not today a single plant operating by steam distillation alone, excepting where its raw material is sawmill waste, and there are



FIG. 1.—STEAM DISTILLATION PLANT FOR RECOVERY OF TURPENTINE AND PINE OIL

only one or two of these. Since it is an industry that has completed its life course in that form, we can sketch its history with safety.

The process of extracting volatile products from pine wood with steam either saturated or superheated and with either excess pressure or reduced pressure, was patented in 1864 and 1865 by Hull, Leffler, and Emery, and has been repeatedly patented since in its various phases. It apparently did not become a profitable industry, however, until soon after 1900 when turpentine began to stay uniformly above 50 cents per gallon at Savannah.

Then the Krug patent was exploited to a considerable extent in Georgia and Florida. The novel idea of the Krug patent was two outlet pipes, one to remove the turpentine and pine oil, and another to remove the products of destructive distillation. The retort used was the regular horizontal type of hardwood distillation retort such as is commonly used in Pennsylvania

and southern New York. Experience showed that the second outlet for destructive distillation products which was the prime feature of the Krug patent was useless because with this system the plants never proceeded far enough to get destructive distillation products.

Nevertheless, several plants were installed using the general type of hardwood retort with only one outlet. Many improvements in methods of handling the wood, systems of conveyors, systems of discharging, apparatus to empty the retorts, convenient bins and chutes for filling them, enabled some of these plants to continue operating for several years. In this system the wood was brought in usually in 4-ft. lengths, and passed through an edging grinder, commonly known as a hog, to reduce it to fair-sized chips and later in many cases shredders were installed. These, with the engines, retorts, condensers and still house equipment were the largest items of cost.

Another type of retort used was a vertical one holding from



FIG. 2.—LIGHT WOOD IN THE WOODYARD OF A STEAM DISTILLATION PLANT

one to three cords. In some cases the steam entered at the bottom and passed upward through the wood. In other cases it entered at the top and passed downward. Each of these types of retorts variously patented by Gardiner, Acheson and Snell, were exploited to a considerable extent, especially in Florida. The chief differences were in the direction of passing the steam, modes of agitating and discharging chips, and variation in type of the lower door. All used saturated steam without more than 20 lb. or 25 lb. pressure.

In addition to the horizontal and vertical retorts another variety was set obliquely, consisting of a cylinder holding less than one-half cord set at an angle of about 60 deg., charging at the top and discharging at the bottom. Two plants of this type were installed in Florida.

Another type which was exploited to a considerable extent was the Bethune system, being a type of vertical retort using superheated steam. Some five or six plants were installed from North Carolina to Florida, but the superheaters usually failed on account of insufficient control and were gradually abandoned.

Still another type was the revolving retort suspended on trunnions, the steam entering through one trunnion and the volatile products passing out through the other. This type seemed to offer certain advantages for charging and discharging, and for agitating the product during extracting. The retorts were sometimes cylindrical or frustum shape, and sometimes spherical. This type of retort offers a curious example of the recurrence of ideas and of the failure to profit by experience and other men's work. Some eight or nine years ago when this idea of a revolving retort was first brought to the writer's attention it seemed to offer enough advantages over other methods at that time in use to warrant a fair trial, so two retorts were designed and installed in a plant in Mississippi.

Just about the time these were ready to operate, a patent was granted to a man in Georgia who apparently had the same idea, although as far as I could learn he never progressed far

enough to install an apparatus to try out his idea. A few months later the writer was called to inspect a plant in Georgia and found there the same general type of retort in operation. It differed somewhat in shape and construction, but the idea was there as in the others. A few years later the same type of retort was called to his attention at a plant in Louisiana, and within the last three months an inspection of another plant near the Texas border revealed our old friend only recently installed, and ready to operate.

In all these cases the newer plant had apparently been installed without any knowledge of the previous ones, and in each case the previous methods had been practically abandoned after careful study of them before the new ones were built. This is simply one of the many cases that might be cited of a continual repetition of errors resulting in financial loss, and in each case the loss might have been wholly avoided by careful study of what had been done before and what data was available regarding that type of installation.

All these installations mentioned had the retort in the shape of a cylinder or frustum of a cone hung on trunnions. Two installations were also made in Georgia of spherical retorts hung on trunnions. These retorts had a much larger capacity than the cylindrical ones, but the method of operation was the same.

When any of these types of retort was used, the method of extraction removed only the turpentine and pine oil. The chipped refuse when discharged from the retorts was very similar in appearance to the original raw material. A portion of it was used for fuel to run the plant, the remainder being sold either locally or at a distance to railroad shops, tug boats or firms requiring fuel for power. As the wood was very resinous the waste made an excellent fuel when the furnaces were properly equipped to use it.

Steam distillation plants required large quantities of steam for power to run the hogs, shredders and conveyors, and for extraction in the retorts, but their waste materials always furnished them more than sufficient fuel, and from one-half to two-thirds of the total extracted wood was available for sale, provided there was a market.

Many attempts were made to extract further products from this waste in the same retort. In one plant in Florida the retorts were built to stand 80 or 100 lb. pressure, with the idea that steam under this pressure would melt the rosin and allow it to be collected at the bottom of the retort. In another, superheated steam was used on the assumption that the wood would be charred and destructive distillation products obtained, but all of these methods failed in actual practice because of the physical condition of the wood. The very large amount of surface exposed prevented an efficient draining and collection of rosin or tar.

Regardless of the type of retort, all of the steam distillation plants finally came to about the same method of operation. The wood was usually brought in in 4-ft. lengths, as stated, but in one case the hog was large enough to receive full-sized logs, and in some cases stumps were used. The wood cost variously from \$1.75 to \$4.50 per cord, probably averaging about \$2.75.

When the wood was hogged, and sometimes shredded, it was placed in the retorts and subjected to a saturated steam extraction. Experience showed that the best results were obtained by varying the pressure with the amount of volatile products distilling.

At the outset the distillation proceeds normally with little or no pressure. When the volatile products decrease if the pressure is raised to 10 or 15 lb. the steam penetrates further into the wood, then when the pressure is decreased the steam and volatile products come out of the chips together and a normal flow of distillate proceeds. This is repeated several times until the extraction is as complete as conditions at that plant warrant.

The maximum pressure usually does not exceed 25 lb. and the time of extraction varies from three hours to six or seven hours, usually not more than four. The crude distillate varies from 10 to 17 gal. per cord, depending on the quality of the wood. Individual cords often give as high as 20 or 25 gal.

but 14 or 15 gal. per cord of crude distillate is considered a very good average unless the wood is especially well selected. The crude distillate separated from the water is then redistilled in stills, with or without fractioning columns, and separated into two marketable products, turpentine and pine oil.

Steam distillation plants deserve the credit for being the first ones to place on the market large quantities of well-refined turpentine and pine oil of a uniform quality and to develop a large market for those products. Steam distillation will be continued probably as a part of other processes, but not as a sole means of utilizing wood, excepting in the case of sawmill waste, which will be discussed in a later article.

It must be said in favor of the steam distillation plants that most of them were built on the very clear understanding that they were not complete processes. The owners in many cases recognized that there was a comparatively small margin in the recovery of turpentine and pine oil alone, even under



FIG. 3.—FEEDING THE HOG

favorable conditions, but the recovery of further products seemed reasonably certain on further investigation.

This recovery did not progress as fast as most of them hoped. Attempts at destructive distillation of the residue revealed many obstacles. Methods of extraction were a long while in being worked out and the ones of these in successful operation will be discussed in another article.

Methods of heating the chipped residue from the outside failed because it was very difficult to make heat penetrate through the mass of sawdust or chips easily, and because when it did penetrate the resin or tar clung to the chips so closely that it could not be recovered to advantage. Methods of incomplete combustion of the residue by burning a portion of it in order to distil the remainder gave some results in small-scale retorts holding a half cord or more, and this might possibly be worked out. A fair grade of tar and tar oil can be produced in this way.

The steam distillation plants were of varying capacity, from 5 to 10 cords per day up to 80 cords per day, and represented investments of from \$10,000 to \$200,000. Where the proper location was selected and construction was good, the plants will probably be permanent, after making the necessary additions. Where poor judgment was used in selection of location or where the construction was not substantial, the plants will be very nearly a total loss. Many have already been dismantled or transformed into other types of plants or had additions made to bring up the total production per cord of wood to a profitable basis.

The difficulty of the steam distillation plants as such was a simple one of revenue. Given a cost of wood of, say, \$3 per cord and manufacturing cost of \$1 to \$2, and taxes, insurance, depreciation, interest and overhead charges of, say, another \$1, besides the cost of packages and selling, their operation was entirely dependent on the price of turpentine. When turpentine was high so that their products brought from \$7 to \$11 or \$12 per cord, they were happy. If their products bring only \$4 they are out of business, and this is their present condition.

Comments on the Report of the Investigation of the United States Patent Office.

By L. H. Baekeland

Whoever is interested in the patent situation of this country should make it his duty to read the unusually interesting publication¹ just issued by the U. S. Government, which covers 624 closely printed pages.

On August 21, 1912, the following joint resolution was passed:

"That the President of the United States be, and he is hereby, requested to cause the accountants and experts from official and private life now or hereafter employed in the inquiry into methods of transacting the public business of the Government in the several executive departments and other executive Government establishments, known as the Commission on Economy and Efficiency, to investigate fully and carefully the administration of the Patent Office with a view of determining whether or not the present methods, personnel, equipment, and building of said office are adequate for the performance of its functions, taking into consideration the present character and volume of business, and also such increase in complexity or volume as may reasonably be expected in the future, and to ascertain and recommend specifically to Congress not later than December 10, 1912, what changes in law, what increases in appropriations, and what additional building accommodations, may be necessary to enable the Patent Office to discharge its functions in a thoroughly efficient and economical manner, and to what extent any expenditures which may be recommended can be met by increases of Patent Office fees.

"All expense incurred in carrying out the purposes of this resolution shall be paid out of any funds in the Treasury of the United States not otherwise appropriated, and the sum necessary for said purposes is hereby appropriated: PROVIDED, That the total expense authorized by this resolution shall not exceed the sum of \$10,000."

The report as presented is signed by Frederick A. Cleveland, Walter W. Warwick, and Merritt O. Chance, Commissioners on Economy and Efficiency.

This interesting report enters into the many ramifications of our patent system, shows up its defects and indicates in a practical way possible methods of improvement.

An enormous amount of precise information has been collected by the above-mentioned commission, but what is more impressive is that all this has been accomplished with the expenditure of such a comparatively small sum of money as \$10,000, and the report should have been prepared and published three months and a half after the resolution was approved. This in itself is a worthy example of "economy and efficiency."

With these data at hand, in addition with the report of the hearings on the Oldfield Bill (see report No. 1161, to accompany H. R. 2347, by Hon. W. A. Oldfield, House of Representatives, 62nd Congress, 2d Session), any future commission for the revision of our patent system will have its work much simplified.

Certain defects in the handling of the personnel of the patent office are pointed out: for instance, it is mentioned that there are no efficiency records for the examining force and that for many years there have been no examinations for the purpose of determining the efficiency of examiners and regulating their promotions and reductions. Such efficiency records, in conjunction with an examination system, would tend to encourage all employees to do their best work and would make adequate salaries a real and lasting benefit to the service.

Emphatic stress is laid on the unsuitability of the present Patent Office.

"In the preliminary survey preceding the investigation the commission was impressed with the idea that the building accommodations for the Patent Office are so inadequate as to

¹62d Congress, 3d Session, House of Representatives, Document No. 1110. Report of the Investigation of the United States Patent Office, made by the President's Commission on Economy and Efficiency, Washington, December 6, 1912.

render extremely difficult any substantial improvement in the work of the office. This impression has been changed as a result of the investigation until it has become a settled conviction that any permanent improvement in the quantity and quality of work done by the office, if done at a reasonable cost, must wait upon provision being made for adequate office accommodations for the force of more than 900 people employed in the Patent Office."

It is a hopeful sign that the Bulkley Bill (H. R., 28,193, 62d Congress, 3d Session) should already have taken up this matter, and intends to provide a \$4,000,000 fireproof building. It is questionable, however, whether this sum of money will be sufficient for the purpose. When we take in consideration that the Patent Office has now an accumulated surplus of \$7,000,000 over all its expenses, there is no reason why this surplus should not be used for providing a suitable building, and the equipment that goes with it.

Through the courtesy of the authorities of the German Patent Office, I have recently been enabled to inspect that magnificent patent office building in Berlin, and its operation. I have been more than ever impressed with the fact how antiquated and inadequate our Washington Patent Office equipment has become in view of the enormously increased development of our country.

As to the needed changes in our patent laws, the remark is made that the present law is substantially the same as the act of 1836. There is a significant allusion to the conservatism of lawyers and their opposition against any changes in procedure.

"No change, however much it may be needed to keep pace with changed conditions, can be made without vigorous opposition due to a large extent to that inherent prejudice against disturbing the daily routine life and the inconvenience of adapting one's mind to a new course of action. . . . Recommendations made by the present Commissioner of Patents for the amendment of laws to effect what he considers are improvements in the methods of the office and to expedite its work are opposed by those who do not agree with him as to the wisdom of the changes as well as by those whose financial interests would be affected unfavorably. . . . the commission is convinced that any proposed change in the patent law and almost any proposed change in the methods of procedure in the Patent Office will arouse opposition from some persons interested in securing patents."

This statement is very well corroborated by the answers to the list of questions sent out by the Economy Commission to members of the bar, inventors and others interested in matters before the Patent Office. More than 400 replies were received. On most questions there was a considerable divergency of opinion.

But even here it is significant that on question No. 24, relative to the desirability of creating one single Court of Patent Appeals, the answers were practically all affirmative and only a few opposed.

Questions 1, 2, and 3 asking opinions as to the advisability of abolishing one of the two appeals in the office and creating one appeal board of five or six members were answered in the affirmative in about 80 per cent of the replies. The direct negative to these questions was less than 20 per cent. The trend of the opinions expressed lies decidedly toward a single appeal to a board in the office, particular stress being laid on the importance of having the work of the primary examiner performed in a thorough and capable manner. The great majority of answers opposed the sitting of the commissioner, the assistant commissioners, or any administrative officer on this board, the idea being expressed that the commissioner should be left free to devote his time to purely administrative duties.

Answering question No. 14 "to what extent should the decision on such appeal operate as a final adjudication of patentability," a small number answered that the decision should be final, and a few that it should not be final. Some favored it being final only as to adverse decisions. Other suggestions were: It should be final, but not as against newly discovered

prior art, in case of litigation under the patent; decision of the Patent Office in interference cases to be final and only to be opened in court upon new evidence found, as the proceedings are far less expensive in the office than in the courts.

Chapter 4 is devoted entirely to an examination of our cumbersome and time-robbing system of interference. The commissioners conclude that it would be better not to make any important changes in our present interference procedure. They seem to overlook the fact that our interference system, as it stands today, hardly protects the "first" inventor if the latter does not possess the abundant cash required to defend his rights against wealthy opponents. Furthermore, a good patent system should be devised in the interest of the nation, the public at large, and not for the limited individual benefit of some inventor or another.

The essence of good patent law is to induce inventors to disclose their invention as promptly as possible, so that the public may have the early benefit of such publication. Our interference proceedings, in conjunction with other defects in our methods of allowing patents, encourage systematic delay in the publication of patents; in reality, a premium is now put on such delays.

All this would be much simpler if in all interference cases priority were allowed to the independent inventor who has filed first his patent, provided he has not copied nor stolen the invention from others.

The German law allows an almost absolute claim to priority to the earlier applicant. This system, therefore, has the great advantage of simplicity over the system in practice in the United States, but it may become a source of injustice in case the latter applicant is really the earlier inventor. However, the latter applicant has only himself to blame if he does not file his application in due time, instead of doing as is now the case, and purposely postpones filing his claims under the immunity of our interference rules. This finally comes down to the question whether it is better to confer the right of priority to the earlier date of application which can clearly and instantly be established, or to the date of conceiving the invention and reducing it to practice, which brings forth all the uncertainties, vagueness, which frequently make our interference proceedings so absurdly long, complicated and expensive. Our present interference rules are undeniably a real handicap to the inventor of limited means; and our system is detrimental to the best interests of the nation. Indeed, it delays the essential benefit of publication of the invention; furthermore, in some cases it becomes a positive danger to any industrial enterprise when nobody knows whether he is not infringing some unpublished patent rights which have been silently ripening to interference proceedings, protected by all the possible customary delays sanctioned by our rules of practice.

The German and British systems permit to file opposition proceedings within a certain period after a patent is allowed; this is an excellent simplified substitute for our interference proceedings. Members of the German patent office have invariably declared that they attach considerable importance to the maintenance of this system, because it tends to reduce the responsibility of the examiners and to enhance public confidence in the integrity and capability of the board of examiners; this in itself checks criticism to which such sensitive men as inventors are only too prone after they have encountered some disappointments in the office. I have noticed the same uniformly favorable opinion among German inventors, as well as patent attorneys and manufacturers, as to the advantages of the so-called opposition proceedings. Many of them, however, are of the belief that the British system, which allows opposition to a patent during a certain period *after* the latter has been printed and published is, however, more practical and expedient than the German method where opposition has to be filed *before* the patent has been printed, and before copies are readily obtainable except by means of special copies made from the pending application at the patent office.

As another check to worthless patents, the German law provides a safeguard in the method of annulment proceedings.

Such proceedings are quite distinct and independent from any infringement proceedings. Any person, whether sued for infringement or not, may apply to the patent office for the annulment of any patent if he can prove that, at the time of application the subject-matter was publicly known, or was claimed in a prior patent, or was illegally appropriated by the patentee. However, in the latter case, annulment proceedings can be instituted only by the person from whom the invention was appropriated.

In a similar manner, a person who believes he can prove that he is a prior inventor of an invention patented to another, and that the patentee has stolen his invention, can sue the patentee for an assignment of the patent to him.

There is still another safeguard in the German law, called the right of prior use (*Vorbenutzungsrecht*) which prevents the dangerous possibility which now exists in the United States, that a business enterprise, started in good faith, might be held up at any time on account of a long secretly pending patent application.

In Germany, any person who, at the date of application of a patent, has already used the invention in that country, or made the necessary preparations for such use, has the right to use the invention for the requirements of his own business in his own works; but this right can only be transferred to others together with the good-will of the business.

It should be noted here that sometimes the suggestion has been made to cut down entirely our examination system, as well as our interference system, and to come back to a plain registration system, which was abandoned in this country 76 years ago, but is now known as the French system. The report of the commission says emphatically:

"To return to such a system would be a backward step."

"While there are some persons interested in the subject who believe that the United States should return to the system of registering patents and allow all questions of validity and priority of invention to be determined in the courts, these persons are decidedly in the minority."

In France, a patent may exist for years before an examination is made as to priority and patentability, because such a scrutiny only takes place in case of a patent suit; and even then, the examination is not made by technical experts, but by the ordinary courts, who may or may not decide to hear experts. This absurd system accounts for the scant esteem with which patents are treated in the Latin countries where the French system is in use. The French system has certainly not shown the stimulating effect to invention and enterprise which the American system has introduced in such nations like Germany, England and the Scandinavian countries, which have adopted it in a more or less modified form.

It should be mentioned, furthermore, that in all countries where the French system is used, a patent really begins to be looked upon as serious after the corresponding German patent has been issued. This in itself, is the best tribute to the practical value of the preliminary examination system.

The most striking incongruity in our American system is that after a patent has been granted by a technical board, in infringement cases, this same patent is reexamined as to validity and patentability by a non-technical judge, in a non-technical court. This is the main reason why our system has been defined as "best in the world for lawyers and worst for inventors." (H. W. Leonard, *Electrical World*, Vol. 51, p. 1055.)

In this relation, the following words are very significant:

"The commission has been unable to see the benefit of the present system by which every question relating to a patent can be litigated in any part of the country.

"When the Patent Office shall have been furnished with adequate quarters and equipment and the most efficient personnel it will be time to consider to what extent the decision of this office ought to be final in patent matters. When that time comes, it may be found advisable to consider whether a patent should not be made valid by law to the extent of giving the patentee a right to an injunction in a case against alleged infringers.

"This would probably require the adoption of the practice of publishing applications when ready for allowance, with the opportunity for anyone to file opposition within a limited time, and if none were filed or it was decided that the patent should issue, it ought to be held valid for all purposes until declared invalid by court. Such a system would probably require also that any person claiming to be injured by the grant of a patent could file annulment proceedings within a limited period . . .

"There is a general complaint, although the commission is not advised as to the facts of the matter, that a grant of a patent is of no value to an inventor if it is in the interest of any person or corporation of financial strength to infringe it. The results after a patentee wins an infringement suit are often not of real benefit to him, for, if he survives the suit, the actual recovery of adequate damages is exceptional.

"It would seem that the remedy for many of the difficulties of the present situation regarding patents will be found in the development of the Patent Office as an administrative court in which all questions relating to patents, excepting the question of infringement, will be finally decided, subject only to a review by an appeal to a court of patent appeals, with possibly a provision for certain questions going to the Supreme Court. Giving such jurisdiction to the Patent Office, and to one court, will tend to the development of a system of patent law that will be a benefit to the public and a protection to real inventors."

The chapters 5 and 6 are devoted to the classification division and to the scientific library and the search room. They point out the existing defects and contain many excellent suggestions.

The fact that in the files of the patent office, there are still missing copies of 60,000 German patents and of 6000 French patents, is significant. It is true that for a certain period France discontinued publishing its patents.

Several other countries, for instance, Canada and Belgium, do not publish their patents excepting by title. Yet copies of every one of the patents published in those countries ought to be available in our search room; indeed, many of these patents contain important disclosures which have a direct bearing on the validity of the patents issued by the U. S. Patent Office.

It would not involve an exorbitant expense to have type-written copies made of the patents filed by those countries. A more satisfactory method would be to compel all countries which belong to the International Union to print every patent issued by them. This subject should be taken up at the next meeting of the International Convention, and could hardly cause any opposition, in as far as almost all countries belonging to the convention already print these patents.

A very detailed criticism of the totally inadequate conditions of the building and the equipment of the patent office is made, and specific recommendations are made for remedying this intolerable condition.

The commission overlooks the fact that the equipment of a new patent office will never be complete unless provision be made for a chemical, a physical and a mechanical laboratory, where simple direct tests can be made under the supervision of two or three experts—a chemist, a physicist, and an engineer; so that many simple questions which now baffle the examiners in their decisions on patentability could easily and quickly be decided by a direct test. Such a laboratory exists in the German Patent Office building.

The report is accompanied by several appendixes, all of which make interesting reading. These appendixes are as follows:

- A. History of the United States Patent System.
- B. United States laws and rules of practice relating to patents, trademarks and prints and labels.
- C. The German Patent law
- D. The English patent law.
- E. Discussion of the German patent law and patent procedure.
- F. A comparison of the patent laws and procedure in Germany, England and the United States.
- G. Methods of examining applications.

H. Publications of the patent office.

I. Statement of the business of the patent office.

J. Bibliography of the United States patent office.

K. Classification of patents and printed publications.

Specially instructive are appendixes E and F, written expressly by Prof. Dr. Albert Osterrieth and Mr. A. du Bois-Reymond, both of Berlin, and leading authorities on the subject. The latter paper not only compares the advantages of the laws of Germany, England and the United States, but points out their relative defects which experience has proved to exist and furthermore discusses the differences in procedure in these countries.

This discussion of the advantages and defects of the leading patent systems of the world, is a most valuable contribution and will be very useful in the consideration of changes that are suggested in the law and procedure of the patent system of the United States.

It is specially interesting to read the comments on the practical value of the so-called compulsory working clause or compulsory license clause in Germany and England, in view of the fact that an attempt is being made by the Oldfield Bill, and other proposed measures, to introduce this innovation in the United States.

In Germany, the compulsory working of patents has practically been abandoned and is merely kept up as a club to compel some countries like England, France and others into reciprocity treaties waiving the compulsory working of patents.

To the credit of the present commissioner of patents, it should be mentioned that he has already negotiated a treaty to that effect with Germany.

The United States, in accord with Germany, might have a decisive influence in abolishing this compulsory working clause in foreign countries, so irksome to American patentees, by passing a law whereby foreigners are compelled to work their patents in the United States, unless their country waives compulsory working of patents issued to American inventors.

The first Oldfield Bill had a clause of the kind, but it was so erroneously worded that in some paradoxical way, it gave superior advantages to foreigners over American inventors.

As to the compulsory working and compulsory license in Germany, we might best quote Prof. Osterrieth:

"In fact, very few patents have been revoked in Germany for non-working (one patent out of more than 2000). On the other hand, the working clause always threatens the patentee, hanging over his head like the sword of Damocles, as a great expert in patent matters once said. Besides, experience has shown that petitions for revocation are mostly entered by patent infringers, using the working clause as a defense.

"This explains why in the last 15 years, there has been a very strong movement in Germany for abolishing the working clause. This first induced the German Government to abolish the obligation of working by international treaties in the relations with Italy, Switzerland, and with the United States. And finally the old working clause of the first patent law was abolished by the law of June 6, 1911.

"Since this law came into force, i. e., July, 1911, no obligation of working a patent exists in Germany. No patent can be revoked on the ground that it is not being worked. Yet, considering the legislation of some countries which threaten foreign patentees with revocation of these patents, if the invention is exclusively or mainly manufactured abroad, the German Government resolved to provide for a similar clause, applicable at least against such countries where German manufacturers have to suffer from such legislation.

"It has therefore been stated in above-mentioned law of 1911, that a patent can be revoked, as far as international treaties do not provide to the contrary, if the invention is exclusively or mainly manufactured or carried out outside the German Empire, and its possessions. This clause does evidently not apply to citizens of the United States, even if an American has acquired a German patent from a patentee belonging to another country. Yet this might lead to abuses. Therefore, there has been provided a complimentary clause, saying that transferring

a patent to another person is ineffective if this transfer has been made only with the intention of avoiding revocation.

"According to the former German law a patent could be revoked, after expiration of three years from the date of granting, if it appeared in the public interest that permission to use the invention be granted to others, but the patentee refused to give such permission in return for adequate remuneration with adequate security. This clause has proved most ineffective as no single case of revocation of a patent on this ground has been known. Therefore, when the abolition of the working clause was discussed, it was suggested to do away also with that clause on the obligation of granting licenses. It was objected that cases might happen where the use of a patented invention by others than the patentee could be necessary for the public welfare, for instance, manufacturing certain drugs in case of an epidemic.

"This consideration seemed to justify the insertion of the following clause in that new law of 1911:

"If the patentee refuses to grant license to another for using the invention upon the offer of an adequate compensation and security, such grant for using the invention can be allowed (compulsory license), if such granting seems necessary in the public interest. The grant may be limited or subject to special conditions."

This detailed report of the Commission on Economy and Efficiency is supplemented by some specific recommendations as follows:

1. That a new building specially designed, equipped, and furnished be constructed on a suitable site in the city of Washington, for the exclusive use of the United States Patent Office.
2. That the number of officers and employees of the United States Patent Office be increased, and the increases and readjustments of salaries be made as shown in detail in this report involving an increase of 36 in the number of employees and a total increase of \$236,550 in the pay roll.
3. That the Commissioner of Patents be the head of the Patent Office; that his duties be the same as are now prescribed by law, excepting that he be relieved from the consideration of cases on appeal; that he be aided by an assistant commissioner and seven supervising examiners in the administrative work, including control of the methods and procedure of the 43 examining divisions in the allowance and rejection of applications for patents.
4. That one appeal within the United States Patent Office be eliminated; that the number of members of the board of examiners in chief of the Patent Office be increased from three to five; that all appeals within the office be taken to that board; that its decision be the decision of the Patent Office; that the appeal therefrom be to the Court of Appeals of the District of Columbia, as now allowed from the decisions of the Commissioner of Patents.
5. That the fee for filing an application be increased from \$15 to \$20; that appeal fees be adjusted to the conditions arising from the elimination of one appeal; that a fee of 25 cents be charged for each additional patent, etc., included in one instrument presented for record; that all fees be paid directly to the Patent Office; that refundment of fees paid by mistake be made by the financial clerk and not by warrant from the Treasury.
6. That the life of a patent be so limited as to expire 10 years from the date of filing the application therefor, excluding the time (not exceeding two years) during which an application may be involved in interference.
7. That the work of reclassifying patents and digesting of printed publications, and providing facilities for simplifying and making more accurate the search, be recognized by an appropriation for an adequate force to be employed upon such work.
8. That the subscription price of the Official Gazette be increased from \$5.00 to \$10.00 and the method of distribution to libraries be changed to reduce the number of copies so distributed.
9. That all the work of producing the publications of the

Patent Office, including copies of patents, be done at the Government Printing Office.

10. That an appropriation be made for the repair of the rooms occupied by the Patent Office and for the installation of suitable lighting and ventilating facilities and for the purpose of new furniture and equipment.

Yonkers, N. Y.

Essentials in the Making of the Finer Steel Sheets

By John G. Homan

Wood is combustible, fragile, increasing in price, decreasing in quality and availability; accordingly other materials are naturally sought for substitutes. Where automobile bodies, certain furniture, interior trimmings of houses, etc., are concerned, sheet steel, with clean, highly finished surfaces has found welcome. The relation this class of sheets bears to the common and familiar "One Pass" material is about the same that hemlock or elm might bear with rosewood or lignum vitae.

For instance, where the architect calls for the splendid walnut-finished doors, etc., for the modern fireproof hotel, he will be satisfied with no less than a product which requires careful inspection to determine the body material. His art calls for finish and imitation, while his engineering demands durability and fireproofing qualities.

The sheet steels for this work have much in common with the woods they supplant; they are necessarily of fine texture or grain, and the practice which produces the commoner material can no more produce satisfactory auto sheets or metal furniture steel than a cupola could make armor plate.

It is the intention of this article to relate something of the essentials in the manufacture of these highly finished steels, as is carried on by a concern devoting all its energies and fabricating equipment to these products.

The steel making is carried on in a plant equipped with basic open-hearth furnaces of moderate capacity; moderate capacity because of the satisfaction obtained in the cleaning action of such furnaces, due likely to the larger value of the ratio of slag surface to bath volume and other considerations not available in the larger furnace of 60 or 100 tons capacity. The fuel is natural gas and has therefore the advantage of easy regulation, sharp thermal value, and absence of sulphur. The raw materials are as usual, pig iron and scrap, both of which are selected and secured from sources which insure certain regularity of composition.

The policy in steel making of producing clean steel from clean materials, rather than making questionable material and answering the question in the ladle, with all sorts of deoxidizers, etc., is followed, as it is essential that the steel be clean; and the cleanliness is to be determined not by the testing machine, but by the material accepted or rejected by the finisher.

The casting is done by the bottom cast method and has many advantages already familiar to those who read this article; the disadvantage is expense. Here, as in all the processes, care is needed, the runner brick and pouring system must be clean. Compressed air is useful in removing particles of brick, etc., which would otherwise contaminate good steel.

When cast and cool the ingots are stripped from their molds and sprues connecting them broken, separating the ingots, which are then piled in stock. From this stock they are taken to the heating furnaces preparatory to reduction. During the heating a rather heavy covering of scale is formed.

The first mechanical reduction is accomplished by the blows of an 8-ton steam hammer. The action of the hammer on this steel is very much the same as for the fine hammered tool steels, it accomplishes a homogenous refining of the grain; a refinement which no rolling method in the writer's experience has been able to show. At the first blow of the hammer almost

all of the scale envelope falls away from the ingot, carrying with it any surface impurities from unclean molds. The final blow finishes the 1500-lb. ingot into billets of section about one-fourth that of the ingot. From the hammer these billets are conveyed on buggies by men to the bar mills. The billets are not reheated after leaving the hammer.

At the bar mills the billet is roughed down in five passes to a suitable size for the finishing stand of rolls, in which it is finished in five more passes to a bar 8 in. wide and of weight corresponding to the sheets to be made from it. At all times when possible water is used on the hot bars; one of its important effects being the tendency of the steam formed to remove surface scale, if any. The long, finished hot bars are conveyed one at a time to a bar shear where sheet bars are cut to length under the inspection of a man who throws aside any bad looking material. The selected bars are allowed to fall into a vat of circulating cold water, where they are further cleaned by the explosive action of the steam formed on their surfaces.

For certain classes of highly finished product the bars thus made are cleaned by pickling in a solution of sulphuric acid, but in the manufacture I am outlining the steel is made clean and every effort is used to keep it clean, so that ordinarily little is gained by the bar pickling.

The bars are rolled into sheets by very heavy mills; the main advantage being the ability of such rolls to produce sheets from bars relatively cooler than the usual practice would permit. The rolls in question are 38 in. in diameter and 44 in. long and weigh approximately 12,000 lb. each. The stands are correspondingly heavy and designed to allow as little elasticity as possible. Brushes and water are used freely at these mills.

The large roll has the advantage of the larger periphery and the consequent longer life of a turning, and as they are turned up much more frequently than the ordinary sheet mill rolls, they will produce material with much more uniform gauge because of this fact.

While in ordinary practice the roughing or "soft" mills are little regarded as to condition, in this procedure utmost care is bestowed on them. Attempt is made to produce a sheet so uniform in physical and mechanical make-up at the hot mills that the subsequent treatment will have a uniform action because of a uniform base.

The furnace in which the bars are heated, as well as that in which the sheet packs are heated, requires care that appeals to the tonnage producer in the ordinary mill as impractical. Here care is required that can be exercised by a skill that comes only from experience. The design of the furnace, the kind of fuel and the quality of flame are very important considerations.

The finished packs are slowly cooled and no water is used on them as is sometimes required to assist the opening process. When cool the packs are sheared and the sheets separated or "opened." The temperature of rolling, together with the chemical composition of the sheets allow easy "opening."

After shearing, various methods or procedures may be followed, depending upon the requirements of the user. It is in order to say that these requirements are very different even with manufacturers using the materials for the same purposes, so that a finished sheet which appeals to the idiosyncrasies of one purchaser may be too hard, too soft, or lack certain surface characteristics that may be desired by his fellow craftsman.

After shearing, cold rolling is usually in order, if so, the material is conveyed to the cold rolls. In the works where fine goods are made two classes of cold mills are required, one for the beginning or rough work and the other for the finishing passes. Both kinds require great care because the little waves and small marks that are always present in the very best grades of common sheets are absolutely out of question here. When a sheet is fed into the cold rolls the sharp edge of the end of it marks the rolls so that after a little while the rolls become almost covered with parallel lines from this cause, so that the following sheets have these marks transferred to their surfaces and are objectionable. The marks must be removed.

and it is the function of the finishing cold mills, and intermediate processes, to remove them as well as many other errors that emphasize themselves only when one has to deal with the rigid requirements of the usual buyers of highly finished sheets.

After cold rolling one or more passes through the cold mills, the sheets may be cleaned by pickling in acid, washing and drying. This is essentially a hand work and requires all the patience and skill of a good launderer because some sheets respond quicker than others and some require attention that machines of any type familiar to the writer cannot give.

After this important operation, for a certain finish, more cold rolling may be in order, or for another requirement annealing may be required. This latter operation is carried out by placing the sheets in a pile on a cast steel plate, or "bottom," and covering the pile with an air-tight wrought-steel box. This combination is sealed at the junction of the box and bottom with sand and run into a furnace where heat is applied to the extent required. This operation is usually in the hands of an experienced foreman who knows how to apply the heat properly, which means not only time, temperature and uniformity, but the expert knowledge of the phenomenon that what appears from outward signs to be going on inside of the annealing box may or may not be going on. The annealer must know when and how to draw the boxes and he must, of course, be familiar with the causes that produce even the little effects usually overlooked entirely by the ordinary sheet annealer. Lately we have assisted the skill of this workman with that very useful tool of the metallurgist, the pyrometer. Equipment is here, as usual in this manufacture, of utmost importance; fuel, furnace design, type of boxes, etc., require detailed study.

The operations mentioned above in brief detail constitute the more important ones, and for the different kinds of goods they are applied in different combinations of routine. The thing least altered is chemical composition of the steel, probably because the one used is known to give the better surface tendencies and likely also because, for the use intended, sufficient hardness or softness can be had by skillful working.

Besides the above-mentioned processes there is an adjunct one connected with that of annealing and which is known as "deoxidizing." While the theory of this process is apparently simple, the real work is more or less difficult. The process is a matter of breaking up by heat the oxide film which covers the usual sheet of steel and preventing its reformation while cooling by filling the box in which the heating is done with a reducing or neutral atmosphere, usually natural gas. The process should leave a very clean, silvery surface, and it is sometimes spoken of as "white annealing."

Then there are operations for final levelling of the sheets, of these the "patent" or "stretcher" leveller is most effective, though very wasteful. The patent leveller is a machine which grips the ends of a pack of three or four sheets in vice-like jaws and stretches them flat by means of a hydraulic system. This operation hardens the material somewhat. Where the jaws of the machine grip the sheets indentations are obviously made which require another shearing operation as well as the loss of the marred ends of the sheets.

The roller leveller is a small machine consisting of two series of parallel rolls about 5 in. in diameter. The rolls are so placed that when a sheet is passed through the machine one series of rolls is above the sheet and one series below. The purpose of this process is to remove buckles, hellies, etc., from sheets with minimum hardening effect.

The composition of steel which seems to offer the best all around usefulness when subjected to the processes above mentioned seems to be one of the more pure iron content, i. e., carbon about 0.12 per cent and manganese about 0.40 per cent, with sulphur in the neighborhood or perhaps below 0.025 per cent.

For auto sheets of the highest class considerable cold rolling is necessary, the first two or three passes reduce the thickness somewhat and seems to definitely refine the grain. The hardness produced can be removed by annealing, which process is up to the skill of the annealer, as the degree of hardness in this material is subject to the peculiar wants of the user.

To prevent corrosion in transit, etc., these finished materials are often oiled with an acid-free heavy oil.

Follansbee Brothers Company.
Pittsburgh, Pa.

Laboratory Apparatus for the Exact Analysis of Flue Gas

In many manufacturing plants apparatus are now used and are proving very successful for determining a few of the constituents of flue gas. In some automatic CO_2 recorders are employed, in others samples are collected at intervals or during extended periods of time and analyzed by means of various types of gas-analytic apparatus, such as those devised by Orsat, Hempel, Elliot, and Bunte. With any of these apparatus results can be obtained by which the performance of a boiler furnace can be intelligently supervised and the introduction of these apparatus has resulted in important advance in efficiency of operation.

If for technical research the exact composition of the waste gas is desired, however, the analysis is more complicated and has been the subject of an investigation at the Bureau of Mines.

The results of this investigation are described in a paper by **George A. Burrell and Frank M. Seibert** (Technical Paper 31 Bureau of Mines). From this paper the following extracts are taken:

For the determination of carbon dioxide, oxygen, carbon monoxide, hydrogen and methane in flue gas during combustion tests conducted by the Bureau of Mines the authors have used the apparatus shown in Fig. 1.

This apparatus is a modification of the well-known Haldane apparatus. (J. S. Haldane and C. L. Foster, *The Investigation of Mine Air*, 1905, p. 100.) It has been simplified from the original pattern in that the pipettes are all placed in a common train. Three 3-way stopcocks have been replaced by three 2-way stopcocks, and a simpler pipette has been

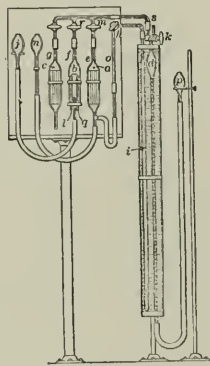


FIG. 1.—LABORATORY FORM OF APPARATUS FOR THE EXACT ANALYSIS OF FLUE GAS

provided for the alkaline pyrogallate solution. The essential features and the method of working remain the same as proposed by Dr. Haldane for analyzing mine air.

The burette *d* contains mercury, and has a total capacity of 21 c.c. The bulb of the burette, which is not graduated, has a capacity of 15 c.c. The stem of the burette has a capacity of 6 c.c. and is graduated at 0.01 c.c.

The three pipettes shown are the potassium-hydroxide, *a*; the slow-combustion, *b* and the alkaline pyrogallate, *c*. The reservoir of the alkaline pyrogallate pipette is placed back of the stand and is provided with a rubber bag to prevent access of air to the alkaline pyrogallate solution.

The surfaces of the liquids in the reservoir bulbs must, in all cases, be at the same level as the surfaces of the liquids in the corresponding pipettes when the liquids are brought to the marks represented at *e*, *f*, and *g*, in the figure. This condition is necessary to prevent a difference in pressure acting against the columns of the liquids in the pipettes when the gas sample is transferred from the burette to the different pipettes.

To perform an analysis, the liquids in the three pipettes are brought exactly to the marks, *e*, *f*, *g*, on the capillary tubes. If an analysis has been recently made, this preliminary step is unnecessary, but if the apparatus has stood unused for some time, the solutions usually have to be adjusted before making an analysis. The three-way stopcock *h* is turned to communicate between the outside air and the compensating tube *i*, so that the pressure in the tube may equal that of the atmosphere, and

is then turned to communicate between the tube *i* and the column of liquid in the capillary tube *o*. The gas sample is transferred to the burette by displacing it with mercury.

The first 5 or 10 c.c. of gas drawn into the burette *d* from the sample container is not retained, but is used to sweep the air out of the connections between the burette and the sample container. For this purpose an extra 3-way stopcock is placed between the burette and the sample container. Gas is then drawn into the burette until the mercury is depressed somewhat below the 21 c.c. mark; then, with the burette stopcock *k* closed, the gas is placed under pressure by slightly raising the level bulb *p*. Next the burette stopcock *k* is opened to the air for a second to bring the gas in the burette to atmospheric pressure. Before reading the volume of gas inclosed, the stopcocks at *k* and *m* are turned so that communication is made between the burette and the potassium-hydroxide solution pipette *a*.

If the directions for manipulating the apparatus have been followed with care there is only a slight movement of the potassium-hydroxide solution at *e*. If, through failure to follow directions exactly, there is a decided movement of the solution, error will ensue. A resulting increase of pressure causes an absorption of some carbon dioxide before the reading is taken, and a decrease of pressure pulls the potassium-hydroxide solution into the horizontal capillary tubing, thereby ruining the value of the combustion data subsequently obtained.

By slightly raising or lowering the level bulb *n* the surface of the potassium-hydroxide solution is brought exactly to the mark *o* in the capillary tube.

The potassium-hydroxide solution is brought to the mark *e* above the pipette by a slight movement of the mercury in the burette, and the volume of gas is read to the third decimal place. The burette can be read to 0.002 c.c. The gas is then passed into the potassium-hydroxide pipette, and the carbon dioxide is entirely removed by passing the gas back and forth between the burette and the potassium-hydroxide pipette about four times. In no case should the mercury in the burette be raised above the stopcock *k*. The reduction in volume of the sample caused by the removal of the carbon dioxide is recorded.

The sample is next passed into the slow-combustion pipette *b*. The platinum coil in this pipette is No. 30 (B. & S.) wire and is in contact with platinum wires in two glass tubes filled with mercury. Each wire is sealed through the bottom of its tube and to the projecting ends are attached the copper wires for carrying the electric current. The platinum coil is brought to a white heat by a current of about 5 amp. About two minutes is required for the complete oxidation of the combustible gases contained in the sample. The gas remaining in the capillary tubing is brought in contact with the platinum spiral by passing the sample back and forth between the burette and the combustion pipette several times, care being taken that the mercury in the combustion pipette is not raised to the platinum spiral in the pipette. Finally, the current is broken and the pipette allowed to cool. Cooling is hastened by playing a stream of compressed air upon the pipette. When cool, the gas is transferred to the burette and the contraction in volume is recorded.

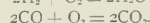
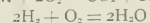
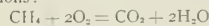
The residual gas is then passed into the potassium-hydroxide pipette *a* to absorb the carbon dioxide produced by the combustion. After the contraction in volume caused by this absorption has been measured, the gas is passed into the pipette *c* containing the alkaline pyrogallate solution to absorb oxygen.

At this stage of the analysis the capillary tubes are manifestly filled with residual gas that contains oxygen. In order to avoid sweeping them out by dilution and to obtain the last trace of oxygen, the mercury in the combustion pipette is brought to the stopcock *r* by raising the bulb *j* and the potassium-hydroxide solution is brought to the stopcock *m* by raising the bulb *n*. The oxygen is completely absorbed by passing the gas back and forth between the burette and the alkaline pyrogallate solution five or six times.

The alkaline pyrogallate solution is finally brought exactly to the mark *g*; the mercury and the potassium-hydroxide solution are brought to their respective marks at *f* and *e*, and the reduction in volume of the gas is measured. The oxygen percentage thus determined represents the oxygen remaining after a certain volume has been consumed in the burning of the combustible constituents. To determine the volume of oxygen consumed, a separate determination is made by drawing a fresh portion of the sample of gas into the burette and passing it in turn into the potassium-hydroxide and alkaline pyrogallate solutions and measuring the reduction in volume produced. The total oxygen content thus determined, minus the residual oxygen already found, represents the oxygen consumed.

Other burette readings are made in a manner similar to that in which the original volume of the sample was read. After combustion, for instance, the gas is withdrawn to the burette and the mercury in the combustion pipette brought exactly to the scratch at *f*. The stopcock at *r* is closed and the one at *m* opened. The solution in the capillary tube is brought to the scratch *o* by raising or lowering the level bulb *n* and the solution in the potassium-hydroxide pipette *a* is brought to the mark *e* by a slight movement of the mercury in the burette. The burette is then read.

From the contraction, the carbon dioxide produced by the combustion, and the oxygen consumed, the methane, carbon monoxide, and hydrogen may be calculated by the following well-known equations:



An analysis such as is described above can be performed in 35 minutes.

Minor Details.—Heavy-wall pure-gum tubing is used on all connections, and the stopcocks are kept well lubricated. The stopcock grease used is made by adding one part paraffin and three parts vaseline to one part pure gum rubber. The rubber is melted, the other ingredients are added, and the mixture is then heated to melting for several minutes. The inner walls of the burette and the compensating tube are kept slightly moistened with water. The instrument should be protected from drafts and the water in the water jacket, each time before burette readings are made, is agitated by blowing air through it, thus insuring uniformity of temperature throughout the water jacket.

It is desirable, if compressed air can be obtained, to keep air bubbling through the water jacket during the entire analysis.

TABULATED RESULTS OF DUPLICATE ANALYSES OF A SAMPLE OF FLUE GAS

Observation or Calculation	FIRST ANALYSIS				SECOND ANALYSIS			
	Volume 1	c.c. 2	Volume 3	c.c. 4	Volume 5	c.c. 6	Volume 7	c.c. 8
Volume of sample taken	20.178		20.300		20.379		20.300	
Carbon dioxide (after caustic polish)	18.181	0.90	18.291	0.90	18.361	0.91	18.288	0.91
Oxygen (after pyrogallate solution)			16.373	9.45			16.373	9.43
Total volume of residual gas	18.181				18.361			
Volume after burning	18.006				18.181			
Contraction	0.175				0.175			
Volume after carbon-dioxide absorption	17.800				17.982			
Carbon dioxide produced	0.206				0.204			
Volume after oxygen absorption	16.022				17.101			
Residual oxygen	1.778				1.791			
Oxygen consumed	0.128				0.127			
Methane	0.01				0.02			
Carbon monoxide	1.02				0.98			
Hydrogen	0.24				0.23			

Bubbles of water mercury must be kept out of the horizontal capillary tubing. It is scarcely necessary to add that a trace of potassium-hydroxide solution left in the tubing will ruin the

value of the combustion analysis. The tubing from *o* over to the compensating tube *i* must also be kept clear of detached bubbles of the potassium-hydroxide solution.

Recording of Results.—The following results of two duplicate analyses show the details observed and the calculations made. As will be observed, the analyses are of flue gas from a boiler furnace operating under very poor conditions:

In column 1 are shown data representing the changes in volume of the gas sample at different stages of the analysis.

In column 2 are shown the proportions of carbon dioxide, carbon, monoxide, hydrogen, and methane calculated to a percentage basis.

In column 3 are shown the observed data involved in the separate oxygen determination.

The final figures representing the percentages of carbon dioxide and of oxygen are shown in column 4.

In columns 5, 6, 7, and 8 are shown the readings and results of the duplicate analysis.

From the combustion data calculations were made to show the errors that might result if the oxygen consumed were not determined, and the results were calculated to carbon monoxide and methane. The possible errors are indicated below:

POSSIBLE ERRORS FROM FAILURE TO DETERMINE OXYGEN CONSUMED

	First Analysis, Per Cent	Second Analysis Per Cent
Carbon monoxide	0.78	0.76
Methane	0.24	0.24

A New House Organ

We have received No. 1 of Vol. 1 of *The Abrasive Age*, "devoted to the better use of abrasives," and published monthly by the Carborundum Company, of Niagara Falls, N. Y. It is interestingly written, well printed, and nicely illustrated. There are articles by F. B. Jacobs on disc grinding, by A. B. Maine on garnet paper and cloth in the chair factory, by Edward Edwards on glass grinding, and numerous smaller notes. We wish this new journal a happy and useful career in furthering the already great popularity of artificial electric-furnace abrasives.

The selection of proper screens in any industry where products are sized is one of the most important branches of that industry. This subject has received close attention at the hands of W. S. Tyler Company, Cleveland, Ohio, and is well treated in that company's artistic catalog No. 40. The company is scientifically equipped to render screen users all possible assistance in solving their screen problems. A testing laboratory contains 2000 variations of screen sections, different in size of opening, mesh, metal, and diameter of wire. From the catalog it should be possible to select a screen suitable for almost any purpose, as over 200 styles are illustrated and fully described. The catalog contains also some valuable information and data on screen analysis, with methods of plotting such an analysis in two different forms. Special attention is given to standard testing sieves, with openings increasing in the ratio of the square root of 2, or 1.414. For closer sizing another series has openings increasing in the ratio of the fourth root of 2, or 1.189.

The production of copper in Russia has increased greatly in the last few years, according to a report from the Consul-General at Moscow. Seven years ago the output was about 10,000 tons, and did not meet half the requirements of the country. In 1907 the production was 14,750 tons; in 1910, 23,000 tons; in 1911, 26,500 tons, and the estimated output for 1912, 30,000 tons. The production of electrolytic copper in Russia started in 1907, and in 1911 another plant was started. At present all domestic requirements of electrolytic copper are met with metal of Russian origin.

The Determination of Sulphur in Ferro-Vanadium

By Wm. W. Clark

Sulphur cannot be determined in ferro-vanadium by the evolution method, as varying amounts of the sulphur remain in the evolution flask as insoluble sulphides. Every possible method of evolution has been tried and it has been found, that in each case, some of the sulphur remains in the flask. The modification which gave the best results was the addition of 1 gram metallic aluminum and to cc. hydrofluoric acid to the alloy in the flask, using dilute hydrochloric acid as the dissolving agent. About 80 per cent of the total sulphur was the maximum that could be evolved, while on some grades of alloy the results would run as low as 10 per cent of the total sulphur. C. M. Johnston mentions this fact in his book on the Chemical Analysis of Special Steels, Steelmaking Alloys and Graphites, page 16.

The barium chromate method, which is used to determine sulphur in coal and coke, was tried, but it requires, first, the removal of the vanadium, and finally the iron. The method necessitates several filtrations and unless the conditions are kept absolutely constant, the results vary beyond checking limits.

The old gravimetric method is the only one by which the sulphur can be accurately determined in ferro-vanadium. The details of the method in use are as follows:

A factor weight (1.373 grams) of the finely powdered alloy is dissolved in a mixture of 40 cc. concentrated hydrochloric and 10 cc. concentrated nitric acid, and 1 gram sulphur free sodium chloride. The solution is evaporated to dryness, moistened with strong hydrochloric acid and again desiccated. If a red granular precipitate settles out during evaporation, causing the solution to spit, more hydrochloric acid must be added. If the alloy is refractory to these acids, it is fused in a nickel crucible with a mixture of 3 grams magnesium oxide and 7 grams sodium peroxide. The magnesium oxide is used to deaden the reaction, as powdered ferro-vanadium, when heated with sodium peroxide, combines so rapidly with the oxygen from the peroxide, that it melts the crucible and is almost an explosion. The fused mass is dissolved in the least possible amount of water, acidified with strong hydrochloric acid, adding about 40 cc. excess, and evaporated to dryness.

The residue in each method is dissolved in 60 cc. strong hydrochloric acid, diluted to 400 cc. with boiling water, and the silica removed by filtration, washing the precipitate free from salts with hot 3 per cent hydrochloric acid. The filtrate is heated to boiling, 10 cc. of a saturated solution of barium chloride added, the solution boiled for 10 minutes, and set aside over night in a warm place. It is then filtered on a OO Munktell filter, washed free from salts with a hot 1 per cent hydrochloric acid, 1 per cent barium chloride solution, and finally with a hot 3 per cent hydrochloric acid solution until free from all soluble barium salts. When washing the barium sulphate precipitate from the soda fusion, it must be remembered that soda is hard to remove from barium sulphate, and the washing must be continued with the barium chloride solution considerably longer than when washing the precipitate from the acid solution of the alloy.

The precipitate and paper are ignited in a weighed platinum crucible cooled and weighed. The difference in weight, multiplied by 10, gives the percentage of sulphur in the alloy.

The above method, while slow, does not require much of the operator's time, not taking more than 3 hours of actual work on 6 determinations. It is very accurate, and results should check within 0.005 per cent sulphur.

American Vanadium Co., Bridgeville, Pa.

Congress has authorized the expenditure of \$2,596,000 for a building to accommodate the Geological Survey, the Bureau of Mines, the Reclamation Service and the Indian Office, all bureaus of the Interior Department. Plans for construction will go forward at once.

The Heat Balance of the Open Hearth

An Account of an Elaborate Heat-Efficiency Test Made on Two 60-ton Open Hearth Furnaces. By Sidney Cornell.

There is a deplorable lack of any printed data in English in relation to the open-hearth furnace from a metallurgical standpoint. Hence, the following is published with the hope that the data given may be criticized, and lead to further information on this most important subject.

The modern blast furnace, as operated today, is run with due consideration for known chemical and metallurgical laws, but this is not so much the case with the open-hearth furnace. The advantage of a change in this direction can be appreciated, and the time has come for betterment of the conditions as existing in open-hearth practice from a metallurgical standpoint.

The data from which this test was figured was obtained at the Duquesne Works of the Carnegie Steel Company. The open-hearth furnaces tested were two 60-ton furnaces, together with their four Hughes mechanically operated gas producers. It was necessary to test two furnaces due to the fact that the gas flues from the four producers are not separated, hence the gases from four producers feed two open-hearth furnaces. The hearths of the furnaces tested were 32 ft. 0 in. x 14 ft. 9 in., and the Hughes producers were of the standard design for that type, and capable of gasifying 25 lb. of coal per square foot of grate area per hour.

The test covered an entire week, starting when the steam was turned on the producers, the Saturday of one week, and terminating when the steam was turned off the producers the Saturday of the following week, thus covering a period of 161 hours and 32 minutes, and during which time twelve heats were made in one furnace and 11 heats were made in the other, with a total tonnage for the two furnaces of 1,615 tons for the week.

The conditions under which the data for this test was obtained, were strictly operating conditions, and the heat balance is figured from the average of all collected data.

The log of the test follows:

FURNACE CHARGES

Furnaces	No. 89 Pounds	No. 90 Pounds	Total Pounds
Charges:			
Hot metal	938,800	871,200	1,810,000
Cold pig	117,000	38,400	155,400
Blooms	474,500	479,200	953,700
Plate		31,500	31,500
Ingot huts	55,000	40,600	95,600
Rail scrap	25,500	38,000	63,500
Tin scrap	43,000		43,000
Turnings	5,500		5,500
Steel scrap	86,900	81,000	167,900
Stirring rods	2,400	2,200	4,600
Additions:			
Hot metal	122,300	143,900	266,200
Ferro manganese	8,600	5,300	13,900
Ferro silicon	650	400	1,050
Roll sulphur	245	120	365
Ore charged	185,500	139,100	324,600
Ore added	61,000	30,000	100,000
Cinder charged	5,000	4,100	9,100
Scale charged	8,900		8,900
Limestone	126,100	122,400	248,500
Fluor spar	2,250	2,200	4,450
Dolomite	57,000	53,000	110,000
Coal (anthracite slack)	1,300	2,150	3,450
Pounds product	1,905,450	1,715,040	3,618,490
Injoints made	309	271	580
Heats made	12	11	23
Average analysis:			
C	0.320	0.462	0.388
Mn	0.506	0.484	0.495
S	0.0426	0.0430	0.0428
P	0.0174	0.0188	0.0238
Si		0.034	0.0200

The starting point in figuring the distribution of heat in an open-hearth furnace is naturally the fuel, hence the following log was necessary in order to get the time of operation of the Gas Producers. The logs of the operation of furnaces 89 and 90, as here given, put the gas producer operation on a test basis, and which summated gives the steam blown in the operation. Summating the data in relation to the amount of gas used during the operation of the furnaces, and taking into consideration the time the producers were off for cleaning, we get the table showing the steam blown.

OPEN-HEARTH FURNACE NO 89

Date	Hr.	Min.	Remarks
Saturday			
20th	11:25 A.M.		Commenced charging for a soaker.
20th	7:26 P.M.		Gas producers started. Steam on.
20th	8:00 P.M.		Ended charging scrap for soaker. Gas on.
21st	6:45 A.M.		Hot metal added. Gas lowered.
21st	2:30 P.M.		Commenced tapping heat, No. 89,476. No. 1
21st	2:43 P.M.		Finished tapping heat, No. 89,476. Start patch.
21st	2:50 P.M.		Finished patching bottom. Gas off.
21st	3:02 P.M.		Gas on again.
21st	3:15 P.M.		Commenced charging for heat, No. 89,477.
21st	6:15 P.M.		Finished charging.
22d	4:57 A.M.		Commenced tapping. Heat No. 89,477. No. 2
22d	5:40 A.M.		Finished tapping. Gas lowered.
22d	5:25 A.M.		Gas taken off furnace
22d	5:51 A.M.		Gas put on again
22d	8:07 A.M.		Commenced charging for heat No. 89,478. No. 3
22d	8:40 A.M.		Finished first charge.
22d	11:25 A.M.		Hot met-1 to furnace. Gas lowered.
22d	11:40 A.M.		Gas increased.
22d	5:50 P.M.		Tapping commenced on heat No. 89,478. No. 3.
22d	7:05 P.M.		Tapping finished, making bottom. Gas off.
22d	7:30 P.M.		Bottom made. Gas put on again.
22d	7:30 P.M.		Commenced charging for heat No. 89,479. No. 4
22d	8:30 P.M.		Finished charging.
23d	6:15 A.M.		Hot metal added. Gas lowered.
23d	4:49 A.M.		Commenced tapping heat No. 89,479. No. 4
23d	5:15 A.M.		Gas taken off, tapping finished. Make bottom.
23d	6:00 A.M.		Gas put on again. Bottom made.
23d	6:25 A.M.		Commenced charging for heat No. 89,480. No. 5
23d	9:12 A.M.		Hot metal added. Gas lowered.
23d	9:30 A.M.		Finished charging. Gas increased
23d	4:22 P.M.		Commenced tapping heat No. 89,480. No. 5
23d	4:32 P.M.		Finished tapping. Gas taken off
23d	5:20 P.M.		Gas put on again. Bottom made.
23d	5:40 P.M.		Commenced charging for heat No. 89,481. No. 6
23d	8:10 P.M.		Finished charging.
24th	3:32 A.M.		Commenced tapping heat No. 89,481. No. 6
24th	3:40 A.M.		Tapping finished. Bottom making. Gas off
24th	4:06 A.M.		Gas on again. Bottom made.
24th	4:15 A.M.		Commenced charging for heat No. 89,482. No. 7
24th	6:10 A.M.		Hot metal added. Gas lowered
24th	6:15 A.M.		Charging completed. Gas increased
24th	4:15 P.M.		Commenced tapping heat No. 89,482. No. 7
24th	4:25 P.M.		Tapping complete. Bottom making. Gas off
24th	5:10 P.M.		Bottom made. Gas put on again.
24th	5:20 P.M.		Commenced charging for heat No. 89,483. No. 8
24th	9:05 P.M.		Charging finished
25th	4:40 A.M.		Commenced tapping heat No. 89,483. No. 8
25th	4:50 A.M.		Tapping complete. Making bottom. Gas off
25th	5:30 A.M.		Bottom made. Gas put on again.
25th	8:35 A.M.		Commenced charging for heat No. 89,484. No. 9
25th	11:00 A.M.		Hot metal added. Gas lowered
25th	11:15 A.M.		Gas increased. Hot metal in. Charge all in
25th	4:55 P.M.		Tapping commenced on heat No. 89,484. No. 9
25th	5:10 P.M.		Tapping completed. Gas lowered
25th	5:45 P.M.		Bottom made. Gas put on again
25th	6:55 P.M.		Commenced charging for heat No. 89,485. No. 10
26th	3:41 A.M.		Finished charging
26th	3:53 A.M.		Commenced tapping heat No. 89,485. No. 10
26th	3:53 A.M.		Tapping finished. Gas off. Bottom making
26th	4:40 A.M.		Gas on again. Bottom made
26th	6:15 A.M.		Commenced charging for heat No. 89,486. No. 11
26th	8:38 A.M.		Hot metal added. Gas lowered
26th	8:50 A.M.		Charging completed. Gas increased
26th	4:40 P.M.		Commenced tapping heat No. 89,486. No. 11
26th	4:50 P.M.		Tapping completed. Gas off. Making bottom
26th	5:13 P.M.		Bottom made. Gas on again
26th	6:07 P.M.		Commenced charging for heat No. 89,487. No. 12
26th	8:20 P.M.		Charging completed.
27th	3:35 A.M.		Hot metal added. Gas lowered and increased
27th	4:15 A.M.		Commenced tapping heat No. 89,487. No. 12
27th	4:58 A.M.		Tapping completed. Gas off. Making bottom
27th	6:00 A.M.		Bottom made. Gas on again low increased
27th	9:35 A.M.		Cold pig charged for soaker
27th	12:58 P.M.		End of test. Steam off producers

OPEN-HEARTH FURNACE. No. 90

Date	Hr.	Min.	Remarks
Saturday			
20th	7:26 P.M.		Gas producers started. Steam on.
20th	8:30 P.M.		Furnace still empty. Gas on, low.
21st	1:45 A.M.		More gas used.
21st	3:15 A.M.		Gas again increased.
21st	8:55 A.M.		Gas taken off.
21st	8:55 A.M.		Gas put on again.
21st	10:43 A.M.		Charging cold pig. Charging commenced.
21st	11:25 A.M.		Gas put on full.
21st	3:25 P.M.		Charging complete for heat No. 90,500. No. 1.
21st	10:25 P.M.		Commenced tapping heat No. 90,500. No. 1.
21st	10:40 P.M.		Finished tapping. Making bottom. Gas off.
21st	11:20 P.M.		Bottom made. Gas on again.
22d	12:32 A.M.		Commenced charging for heat No. 90,501. No. 2.
22d	12:57 A.M.		Gas increased.
22d	3:15 A.M.		Hot metal added. Gas lowered.
22d	3:30 A.M.		Charging completed. Gas increased.
22d	9:30 A.M.		Hot metal added. Gas lowered.
22d	9:41 A.M.		Commenced tapping heat No. 90,501. No. 2.
22d	9:54 A.M.		Tapping complete. Making bottom. Gas off.
22d	10:12 A.M.		Making bottom complete. Gas on again.
22d	11:25 A.M.		Commenced charging for heat No. 90,502. No. 3.
22d	2:24 P.M.		Gas increased more.
22d	3:45 P.M.		Hot metal added. Gas lowered.
22d	4:00 P.M.		Gas increased.
22d	10:36 P.M.		Commenced tapping. Heat No. 90,502. No. 3.
22d	10:50 P.M.		Tapping complete. Gas still on.
22d	12:15 A.M.		Gas off. Making bottom.
22d	12:57 A.M.		Bottom made. Gas on again.
23d	1:00 A.M.		Commenced charging. Heat No. 90,503. No. 4.
23d	2:30 A.M.		Gas increased.
23d	2:40 A.M.		Scrap charged.
23d	4:55 A.M.		Hot metal added. Gas lowered.
23d	4:00 A.M.		Charge completed.
23d	2:30 P.M.		Hot metal added.
23d	2:34 P.M.		Tapping commenced. Gas still high.
23d	2:50 P.M.		Tapping complete. Gas off. Bottom made.
23d	3:13 P.M.		Bottom made. Gas put on.
23d	3:38 P.M.		Commenced charging for heat No. 90,504. No. 5.
23d	5:55 P.M.		Charging complete.
24th	4:35 A.M.		Commenced tapping heat No. 90,504. No. 5.
24th	4:50 A.M.		Tapping complete. Gas still on.
24th	6:00 A.M.		Making bottom. Gas taken off.
24th	7:05 A.M.		Gas on again. Bottom made.
24th	9:30 A.M.		Gas increased.
24th	10:52 A.M.		Gas taken off again.
24th	10:56 A.M.		Gas put on again.
24th	11:00 A.M.		Charging commenced for heat No. 90,505. No. 6.
24th	1:37 P.M.		Hot metal added. Gas lowered.
24th	1:45 P.M.		Charging complete. Gas increased.
24th	10:20 P.M.		Tapping commenced. Gas off. Bottom.
24th	10:30 P.M.		Tapping complete. Gas off. Bottom.
24th	10:38 P.M.		Bottom made. Gas on again.
24th	11:45 P.M.		Commenced charging for heat No. 90,506. No. 7.
25th	2:10 A.M.		Charging complete.
25th	2:45 A.M.		Gas increased.
25th	4:15 A.M.		Gas increased.
25th	4:55 A.M.		Gas decreased.
25th	9:27 A.M.		Commenced tapping heat No. 90,506. No. 7.
25th	9:35 A.M.		Tapping finished. Gas off. Making bottom.
25th	9:50 A.M.		Tapping and bottom made. Gas on again.
25th	10:05 A.M.		Charging commenced for heat No. 90,507. No. 8.
25th	11:20 A.M.		First charge complete.
25th	11:15 P.M.		Charging complete.
25th	11:40 P.M.		Hot metal added. Gas still high.
25th	11:52 P.M.		Finished tapping. Gas still on. No. 8.
25th	12:00 A.M.		Making bottom. Gas off.
25th	12:53 A.M.		Bottom made. Gas put on again.
26th	2:23 A.M.		Commenced charging heat No. 90,508. No. 9.
26th	4:20 A.M.		Charging completed. Gas increased.
26th	1:54 P.M.		Commenced tapping heat No. 90,508. No. 9.
26th	2:06 P.M.		Tapping finished. Gas off. Bottom.
26th	2:36 P.M.		Bottom made. Gas put on again.
26th	2:39 P.M.		Commenced charging heat No. 90,509. No. 10.
26th	4:15 P.M.		Finished charging.
27th	1:07 A.M.		Commenced tapping No. 90,509. No. 10.
27th	1:15 A.M.		Finished tapping.
27th	1:32 A.M.		Making bottom. Gas off.
27th	2:02 A.M.		Bottom made. Gas on again.
27th	2:09 A.M.		Commenced charging No. 90,510. No. 11.
27th	4:48 A.M.		Hot metal added. Gas lowered.
27th	4:52 A.M.		Gas increased.
27th	12:35 P.M.		Commenced tapping No. 90,510. No. 11.
27th	12:52 P.M.		Tapping, etc., completed.
27th	12:58 P.M.		Producers down end of test.

TIME PRODUCERS WERE OFF FOR CLEANING

DATE	PRODUCERS	No. 1	2	3	4
21	Minutes	25	30	40	25
				70	
22		30	35		35
23		30	30	50	30
24		55	35		30
25		55	35	55	30
26		60	10	55	
27					30

It will be noted in the table for steam consumed that a total figure for steam blown is obtained, this being the figure that is often used in gas producer testing. When the producers are operating at one-half, one-fourth, or no load, less steam

STEAM BLOWN

TEST OF 161 HOURS, 32 MINUTES, ON FOUR PRODUCERS AND TWO FURNACES

Furnace	Charging	Working	Hot Metal	Tapping	Patching	Waiting on Material
Gas On	On $\frac{1}{2}$	On Full	On $\frac{1}{2}$	On Full	Off	On $\frac{1}{2}$
No. 89	23° 33'	106° 26'	1° 19'	2° 33'	8° 29'	19° 12'
No. 90	33° 22'	92° 12'	1° 17'	2° 15'	6° 11'	26° 36'

Steam nozzle, 0.4375" diameter, hence 0.1500 square inches area.
Average gauge pressure when working = 45 pounds. = 60 lbs. absolute
Steam per minute per producer = (Area $\times$ Abs. P. $\div$ 70) $\times$ 60 = 7.713 pounds
Total steam, no delays, for four producers for 161° 32' = 299,018 pounds

Delays:
Equivalent of one producer off cleaning for 15° 10' = 7,019 pounds
Equivalent of two producers off when patching. 14° 40' = 13,575 pounds
Gas on $\frac{1}{2}$ when adding hot metal. 2° 36' = 1,805 pounds
Two producers at $\frac{1}{2}$ steam for 45° 48' = 21,195 pounds
Two producers at $\frac{1}{2}$ steam for 56° 53' = 13,168 pounds

Total. 56,762 pounds
Steam used during the test = 299,018 - 56,762 = 242,256 pounds

is used and in direct proportion to the gas used, hence the above deductions, giving the net quantity of steam consumed.

TABLE NO. 1

	Per Cent.	Weight, Pounds	Per Ton of Coal	Per Ton of Ingots, 1615
Coal charged	100.00	856,000	2,240	530 pounds
Ash loss	5.65	48,360	126.5	29.93
Coal lost in ash	1.07	9,160	23.9	5.65
Coal lost in soot	.69	5,906	15.4	3.64
Total coal loss	1.76	15,066	39.3	9.29
Moisture	1.11	9,500	24.8	5.86
Coal gasified	98.24	840,934	2,200.7	520.0
Carbon in coal gasified	76.23	641,000	1,677.5	396.0
Carbon in gas, per cubic foot		(0.010802)		
Cubic feet gas made		59,340,000	155,290	36,740 cu. ft.
Pounds of gas made		4,219,000	11,040	2,612 pounds
Steam blown		242,256	654	150.3
Air blown		3,353,180	8,775	2,075

Table I shows the distribution of carbon from the coal in the gas made and in the refuse produced in the process. This table shows the distribution per ton of coal and also per ton of ingots made, and is taken from test conditions, and during this test some 530 lb. of coal were required per ton of ingots made. Since it is desired to have this heat balance figured for strictly operating conditions, the figure 510 lb. of coal per ton of ingots is taken, this being an average figure, and the results figured accordingly. Table II is thus figured from the results obtained from a test, and from which the conversion of one ton of coal into gas is obtained, and these figures used in obtaining the distribution for the rate of 510 lb. of coal per ton of ingots made.

TABLE NO. 2

	Per Cent.	Weight	Per Ton Coal	Per Ton Ingots
Coal charged	100.00	829,000	2,240	510 lbs.
Ash loss	5.65	46,840	126.5	28.8
Coal loss in ash	1.07	8,870	23.9	5.5
Coal loss in soot	.69	5,720	15.4	3.5
Total coal loss	1.76	14,590	39.3	9.0
Moisture	1.11	9,200	24.8	5.66
Coal gasified	(98.24)	814,410	2,200.7	501.
Carbon in coal gasified	(76.23)	620,800	1,682.4	381.7
Carbon per cubic foot of gas		0.010802		
Cubic feet of gas made		57,100,000	155,100	35,310
Pounds of gas made (dry)		4,081,000	11,028	2,510
Steam blown		242,256	654	149.
Air blown		3,289,000	8,888	2,024

Following is the analysis of the coal. The proximate analysis is representative of that obtained in the laboratory. Some difficulty was experienced in obtaining the ultimate analysis, especially considering the hydrogen, but the ultimate analysis here given is obtained by Dr. Parr's method and the hydrogen content assured of.

ANALYSIS OF COAL

Proximate Analysis		Ultimate Analysis of Volatile	
Volatile matter.....	34.94%	Carbon.....	17.93%
Fixed carbon.....	58.30%	Hydrogen.....	5.02%
Total carbon.....	76.23%	Nitrogen.....	1.38%
Ash.....	5.65%	Oxygen.....	10.61%
Moisture.....	1.11%	Total.....	34.94%
Combustible in ash.....	15.13%		
Pure coal.....	93.24%		
Coal lost in the ash. Pure coal = 93.24%		True ash.....	= 84.87%
Ash.....	= 5.65%	Combustible.....	= 15.13%
Moisture.....	= 1.11%		
Refuse produced in burning coal. 5.65 + 84.87% = 6.65%			
6.65 - 5.65 = 1.00 = Combustible. 1.00 + 93.24% = 1.07%			
1.07% = Coal lost in ash.			

Approximate Analysis of Ash, Estimated

Fe.....	= .85
S.....	= 1.16
SiO ₂	= 1.54
CaO.....	= .80
Al ₂ O ₃	= 1.30
Total.....	= 5.65%

In the following analysis of gases, the gases are figured at 32 deg. F., and not at 62 deg. F., as is common practice, for the reason that figuring at 32 deg. F simplifies the transfer to the metric system. It would be a great advantage in many calculations if 32 deg. F., and not 62 deg. F., were taken as the standard temperature for measuring gases.

GAS PRODUCERS ANALYSIS OF GAS
VOLUME CONVERTED INTO WEIGHT. GAS AT 32°F.

Volume	Stand- ard Weights	By Weight at 32°F.	Stand- ard BTU, 32°	BTU, in Gas 32°	BTU, in Gas 62°	Weight of Gas at 32°
CO ₂	5.905	0.12424	10.317			0.007336
CO.....	22.625	0.07858	25.003	3.49	78.961	0.017778
O ₂	0.020	0.08953	0.025			0.000018
CH ₄	2.700	0.04505	1.711	10.65	28.755	0.001216
C ₂ H ₆	0.415	0.07870	0.458	16.73	6.943	0.000326
H ₂	12.580	0.00565	1.000	3.48	43.778	0.000711
N ₂	55.755	0.07859	61.486			0.043718
Total.....	100.000		100.000		158.437	0.071103

ANALYSIS OF GAS

Ultimate Analysis		Carbon in the Gas, 32°F.		Analysis of Air	
				Wet	Dry
C.....	15.206	Carbon in CO ₂ = 0.002001 N ₂		75.92	77.00
O ₂	21.815	CO = 0.007610 O ₂		22.67	23.00
N ₂	61.486	CH ₄ = 0.000912 H ₂ O.....		1.41	
H ₂	1.493	C ₂ H ₆ = 0.000279			
Total.....	100.000	Total..... = 0.010802			
Weight of gas per cubic foot, at 32°F.....				= 0.071103 pound	
Weight of carbon per cubic foot at 32°F.....				= 0.010802 pound	

ELEMENTARY DISTRIBUTION AT RATES OF 510 AND 530 POUNDS
COAL-TON INGOTS

510. Rate	Weight	%	C.	O ₂	N ₂	H ₂
Coal.....	814.410	76.23	620,800	10,61	86,410	1,38
Air.....	3,289,000		22,677,85	800,75	92,249,260	
Steam.....	180,360		88,881,60	320		
Gas.....	4,081,000	15.21	620,800	21,81	890,200	61,48
			2,508,500	1,49	60,930	

530. Rate	Weight	C	O ₂	N ₂	H ₂
Coal.....	840,934	641,000	89,220	11,600	42,210
Air.....	3,353,180		770,780	2,582,400	
Steam.....	187,090		166,300		20,790
Gas.....	4,219,000	641,000	920,400	2,594,000	63,000

MOISTURE DISTRIBUTION

Rate	530 lb.	510 lb.
Moisture in coal.....	1.11	9.500
Moisture in air.....	1.41	17,280
Moisture in steam.....	100.	242,256
Total moisture.....		299,036
Decomposed.....		187,090
Free moisture.....		111,946
Grains.....		783,600,000
Cubic feet gas made.....		59,340,000
Grains moisture per cubic foot.....		13.2
B.T.U., per cubic foot, gas 32° F.....		158.4
B.T.U. in gas (latent heat).....	9,399,456,000	9,070,000,000

Having figured the fuel down to a working basis, thereby obtaining the heat available and the losses in the making of the gas, it is necessary to similarly treat the materials charged into the furnace. On account of the numerous chemical reactions involved it is necessary to compile an "elementary distribution," with the object in view of their final distribution and arrangement in the metal, slags, and gases. Accordingly, the following analyses have been taken as representative of the materials charged, and for the object in view have been adjusted to read 100 per cent, since no laboratory analyses are reported as totaling 100 per cent.

OPEN HEARTH. ANALYSIS OF RAW MATERIALS
ANALYSES ADJUSTED TO READ 100°

Element	Cold Pig Hot Metal	Ferro Manga- nese	Ferro Silicon	Blooms	Plate	Ingot Butts	Rail Scrap
Si.....	1.05	0.70	52.00	.025		.025	.15
Mn.....	1.48	78.00		.500	.350	.500	.96
C.....	3.41	5.75	0.20	.400	.250	.400	.47
P.....	.294	.325	.080	.030	.040	.030	.080
S.....	.038	.155	.005	.040	.040	.040	.050
Fe.....	93.728	15.070	47.745	96.005	95.820	95.005	97.330
FeSiO ₄				3.000	3.500	4.000	1.000

Element	Tin Scrap	Turn- ings	Steel Scrap	Stirring Rods	Anthracite Slack
Si.....		.020	.020	.020	VM = 7.45
Mn.....		.480	.500	.450	FC = 75.56
C.....	.18	.180	.250	.150	Ash = 16.99
P.....	.075	.045	.035	.035	TC = 75.56 + 3.72
S.....	.040	.045	.035	.035	O = 2.38
Fe.....	97.305	99.210	96.160	96.310	N = .40
FeSiO ₄	2.000	.000	3.000	3.000	H = 1.05

Element	Roasted Magne- site	Calci- ned Dolomite	Standard Lime stone	Fluor Spar	Soft Ore Charg- ed	Lump Ore	Roll Scale
SiO ₂	2.66	.22	.403	0.39	0.50	6.200	3.71
Fe ₂ O ₃	7.43	1.43	2.637	1.43	0.30	70.148	93.456
Al ₂ O ₃	1.46	.26	.476	0.26	0.40	2.350	1.37
MnO.....	.40				0.00	0.600	1.17
CaO.....	4.18	31.34	57.400	55.15		0.200	.490
FeO.....	.066	.004	.007	.005		.152	.037
CaF ₂				95.00			.158
MgO.....	83.680	21.30	38.897	.33			.200
Moisture.....	0.124	.10	.180	.50	3.80	20.260	.420
FeSiO ₄							.94
CO ₂	45.346		42.035				

The following temperatures are the results of averaging many observations, and can be taken as representative of the temperatures met with in the parts of the furnace quoted. This table also shows the efficiency of the producers and the regenerators, which figures are obtained in the heat balance which follows.

SUMMARY	
MATERIALS USED PER TON OF INGOTS PRODUCED	
Producer Gas:	
Coal.....	510.00
Steam.....	149.00
Air.....	1,995.50
H ₂ O (air).....	28.50
H ₂ O (coal).....	5.60
Gas made (dry).....	2,510.00
H ₂ O (gas).....	72.00
Hot gas efficiency of the producers.....	94.99%
Efficiency at delivery to the regenerator en- trance.....	91.77%
Temperature of gas regenerator.....	2,120°F.
Temperature of gas coming from regenerators.....	1,940°F.
Temperature of gas going to regenerators.....	1,100°F.
Pounds of air required for combustion of pro- ducer gas.....	7,378.00
Moisture in this air.....	100.00
Temperature of air regenerator.....	2,260°F.
Temperature of air going to regenerator.....	905°F.
Temperature of air coming from regenerator.....	2,012°F.
Efficiency of the regenerators.....	79.22%
Reversals on No. 90 Furnace, during the week's test:	
Total number during week.....	= 509
Average time.....	= 19 0 minutes
Longest time.....	= 70 0 minutes
Shortest time.....	= 2 0 minutes
Temperature of steel running from furnace.....	= 3,012°F. temperature of steel
Temperature of burned gases at end of furnace.....	= 2,832°F.
Temperature of gases just outside regenerators.....	= 1,252°F.
Temperature of stack.....	= 1,200°F.

TABLE 3-A

TABLE OF DISTRIBUTION OF THE VARIOUS CONSTITUENTS OF THE CHARGE AND THEIR SUBSEQUENT CHEMICAL ARRANGEMENT IN THE FURNACE—NAMELY METAL, SLAG AND GASES

Material	Pounds	Si		SiO ₂		Mn		MnO		P		As. P ₂ O ₅		Fe		Fe ₂ O ₃	
		%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.
Hot metal.....	1,810,000	1.050	19,000	40.720	1,480	26,790				294	5,322			93,728	1,696,550		
Cold pig.....	155,600	1.050	1,634	3,502	1,480	2,303				294	457			93,728	145,800		
Bloom.....	953,700	.025	238	510	500	4,769				030	286			96,005	915,650		
Plate.....	31,500			51	500	110				040	13			95,820	30,185		
Ingot butts.....	95,600	.025	24	51	500	478				030	28			95,005	90,820		
Rail scrap.....	63,500	.150	95	204	900	571				080	51			97,350	61,810		
Tin plate.....	43,000			400	172					075	32			97,305	41,845		
Turnings.....	5,500	.020	1	2	480	26				045	3			99,210	5,457		
Steel scrap.....	167,900	.020	34	72	500	839				035	58			96,160	161,450		
Stirring rods.....	4,600	.020	1	2	450	21				035	2			96,310	4,430		
Total.....	3,330,900	.631	21,027	45,063	1,084	36,079		46,576	187	6,251		14,313	94,695	3,153,997			
Fine ore.....	324,600			6,290	20,420			600	1,947			152	493			70,148	227,700
Cinder.....	9,100			5,100	464			400	37			160	15				
Scale.....	8,900			1,440	128			400	35			158	14				
Dolomite.....	110,000			403	443							007	8			2,637	2,900
Limestone.....	248,500			490	1,217							005	12			1,430	3,554
Lump ore.....	100,000			3,710	3,710			117	117			037	37			93,456	93,456
Fluor spar.....	4,450			300	22											300	14
Total, added.....	805,550			3,277	26,404			265	2,136			0717	578			40,670	327,623
Run off slag.....	109,084			10,020	10,930			9,230	10,067			1,998	2,180	27,825	30,353		
Normal slag run.....	260,379			15,500	40,358			7,915	20,600			2,845	7,408	14,980	39,000		
Hot metal.....	266,200	1.050	2,795	5,990	1,480	3,940				294	782.5			93,728	249,500		
Fe manganese.....	13,900	.700	97	208	75,000	10,840				325	45			15,070	2,095		
Fe silicon.....	1,050	52,000	546	470						050	5			47,745	501		
Sulphur.....	365																
Coal.....	3,450			7,000	241									2,000	69		
Aluminum.....	411																
Total, No. 3.....	285,376	1.205	3,438	2,826	7,609	5,180	14,780			290	828			88,340	252,165		
Furnace steel.....	3,345,087				179	6,000		7,745	0209	700		1,600	99,640	3,334,097			
Product steel.....	3,618,490	.620	724	1,551	495	17,910			0238	861				3,586,000			
Difference.....	276,823			1,551		11,910				161							
In recarburizer.....	285,376			2,826	7,609	5,180	14,780			290	828						
To ladle slag.....	12,705			47,680	6,058	22,590	2,870	29,060	3,692	5,250	667	11,150	1,468				
One-third normal slag.....	130,189			15,500	20,179			7,915	10,300				3,703				
Ladle slag.....	142,894			18,360	26,237			8,792	13,992			3,620	5,171				
Ladle gases.....																	
Producer gas.....	4,198,476																
Air used.....	12,090,000																
Flue gases.....	17,104,426																

TABLE 3-B

Material	Fe ₂ O ₃		As. FeO		Fe From Oxides		Oxygen		Carbon		CO ₂		Sulphur		Aluminum	
	%	Lbs.	%	Lbs.			Absorbed	Freely	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.
Hot metal.....									3.410	61,720			038	687		
Cold pig.....									3.410	5,306			038	59		
Bloom.....	3,000	28,600							400	3,815			040	381		
Plate.....	3,000	1,102							250	79			040	13		
Ingot butts.....	4,000	3,824							400	382			040	38		
Rail scrap.....	1,000	635							470	298			050	32		
Tin plate.....	2,000	860							180	78			040	17		
Turnings.....	3,000	5,036							180	10			045	3		
Steel scrap.....	3,000	138							250	419			035	59		
Stirring rods.....	1,000	138							150	69			035	2		
Total.....	1,207	40,195			29,106 (Fe ₂ O ₃)	42,595	11,089	3,164	72,076		12,965	104,450	0387	1,290		
Fine ore.....																
Cinder.....	94,340	8,584														
Scale.....	94,902	8,448														
Dolomite.....																
Limestone.....									75,965		28,485	42,035	104,450			
Lump ore.....																
Fluor spar.....					229,331 (Fe ₂ O ₃)											
Total, added.....	2,110	17,032			10,516 (Fe ₂ O ₃)											
Run off slag.....			35	750	39,000		8,647									
Normal slag run.....			19	250	50,117		16,675									
Hot metal.....									3.410	9,077			038	101		
Fe manganese.....									5,750	799			155	21		
Fe silicon.....									200	2			005			
Sulphur.....													101.0	365		
Coal.....									59	79,280	2,735					
Aluminum.....									1,420	12,613				487		
Total, No. 3.....																
Furnace steel.....					3,334,097		2,669			3,000			0385	1,290		
Product steel.....					3,586,000					3,880			0428	1,538		
Difference.....					251,903									258		
In recarburizer.....					252,165		4,396							487		
To ladle slag.....			2,630	335	262			1,695								
One-third normal slag.....					25,058											
Ladle slag.....			17	770	25,393											
Ladle gases.....									59	1,573				229		
Producer gas.....							21	200%	890,200	14,780	620,800					
Air used.....							22	670%	2,741,000							
Flue gases.....							21	967%	3,757,790	4,200	718,361					

TABLE 3-C

Material	Al ₂ O ₃		CaO		CaF ₂		MgO		H ₂		N ₂		Moisture	
	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.	%	Lbs.
Hot metal														
Cold pig														
Bloom														
Plate														
Ingot butts														
Rail scrap														
Tin plate														
Turnings														
Steel scrap														
Stirring rods														
Total														
Fine ore	2.350	7,628	200	649									20.260	65,760
Cinder														
Scale														
Dolomite	.476	524	57.400	63,140			38.897	42,790					3.100	276
Limestone	.260	646	55.150	137,050			.330	820					.180	198
Lump ore	1.570	1,570	.490	490			.200	200					.300	745
Fluor spar	.400	18			95.000	4,227							.420	420
Total, added	1.289	10,386	24.990	201,329	.524	4,227	5.438	43,810					3.800	169
Run off slag	3.006	3,279	33.985	37,070			6.012	6,558					8.386	67,568
Normal slag run	1.819	4,738	42.050	109,506	1.083	2,818	9.540	24,935						
Hot metal														
Fe manganese														
Fe silicon														
Sulphur	6.000	207	1.990	68										
Coal		877												
Aluminum		1,084	.535	68					1.00	34	1.00	34		
Total, No. 3										34		34		
Furnace steel														
Product steel														
Difference														
In recarburizer														
To ladle slag	8.530	1,084	.535	68										
One-third normal slag		2,369			1.409	12,417								
Ladle slag	2.417	3,453	38.370	54,753			8.690	12,417						
Ladle gases					.986	1,409								
Producer gas														
Air used									1.451	60,930	59.75	2,508,500	2.798	117,476
Flue gases									75.92	9,176,861	9.176,861	1.410	170,500	
									68.33	11,685,361	5.514	942,914		

Taking each of the materials charged, their weight and analysis, the tables 3-A, 3-B and 3-C are obtained, which materials split up into the various oxides formed, and also those reduced, together with other reactions, give the elementary

ically, from the elementary distribution in tables 3-A, -B, -C, and also gives a check analysis, obtained from actual analysis in practice. This table also gives the analysis of the flue gases, both wet and dry, by volume and by weight.

TABLE IV
PRODUCER GAS DISTRIBUTION. (WET ANALYSIS BY WEIGHT)

Gas	Pounds	OXYGEN		CARBON		HYDROGEN		NITROGEN		MOISTURE	
		%	Weight	%	Weight	%	Weight	%	Weight	%	Weight
Producer gas											
downtake	4,198,476	21.20	890,200	14.78	620,800	1.451	60,930	59.750	2,508,500	2.791	117,476
At reversing valve	4,237,476	21.01	890,200	14.65	620,800	1.438	60,930	59.200	2,508,500	3.693	156,476
From furnace gases											
add			126,590		97,561						67,568
Combustion of H ₂											
add											
Air used	12,090,000	22.67	2,741,000								548,370
Flue gases made	17,104,426	21.967	3,757,790	4.200	718,361			75.920	9,176,861	1.410	170,500
								68.330	11,685,361	5.514	942,914

final arrangement in the metal, slags, and gases. It will be noted that a run-off slag is first produced, then a normal slag, a portion of which normal slag goes into the ladle to help make up the ladle slag.

The above Table IV on the producer gas is here given to show the distribution of moisture. It will be noted that there is an increase of 1 per cent in the amount of moisture carried in the producer gas after it has passed the reversing valve. Reversing valves, in nearly all cases are water cooled, and the hot producer gas coming in contact with a water seal naturally picks up a quantity of moisture, which throws a large portion of sensible heat that was of service into latent heat that cannot be used. Does not this increase in moisture, and the accompanying decrease in the effective heat delivered to the furnace, permit of the suggestion that reversing valve design might be improved, with the use of other material, such as tar, as a sealing fluid?

On account of the importance of the flue gases, Table V is here printed. This table shows the analysis obtained theoret-

TABLE V FLUE GASES				
Analysis		By Weight		
Gas	Test	Check	Wet	Dry
CO ₂	16.082	16.730	Oxygen	21.967
CO	0.135	0.140	Carbon	4.200
O ₂	11.476	7.930	Nitrogen	68.330
N ₂	72.307	75.200	Moisture	5.514
		Pounds dry gas	10,004	
		Pounds H ₂ O	584	

In all the calculations it has been necessary to consider the moisture present in the materials used and the gases formed. The heat taken out by the moisture, superheated to the high temperatures involved amounts to a large figure, and is an important part of all the calculations. It will be noted in the analysis of the raw materials charged that soft ore runs as high as 20.26 per cent moisture. Economy would be obtained by drying such material before charging.

TABLE VI
 OPEN HEARTH. SUMMARY SHEET MATERIALS

Material	Pounds	Si	SiO ₂	Mn	MnO	P	P ₂ O ₅	Fe	Fe ₂ O ₃	Fe ₃ O ₄	C	Free O ₂	Absorbed O ₂	S
Charged.....	3,330,900	21,027		36,079		6,251		3,153,997		40,195	72,076	11,089	42,595	1,290
Addition No. 1.....	805,530	26,404			2,136		578		327,623	17,032	28,485	75,965	25,322	
Run off slag.....	109,084	10,930			10,067		2,180					104,808		
Normal slag run.....	260,379	40,358			20,600		7,408							
Normal slag to the ladle.....	130,189	20,179		10,300			3,704							
Furnace steel.....	3,345,087		6,000		700		3,334,097				3,000	2,645		1,290
Recarburizer.....	285,376	3,438	241	14,780		828	252,165				12,613		4,396	
Product.....	3,618,490		17,910		861		3,586,000				14,040			487
Ladle slag.....	142,894	26,237		13,992								1,695		1,848
To ladle gases.....	1,929										1,573	89		229
To furnace gases.....	291,719													

Material	FeO	Al ₂ O ₃	CaO	CaF ₂	MgO	Moisture
Charged.....						NOTE
Addition No. 1.....	10,386	201,329	4,227	43,810	67,568	To Gases
Run off slag.....	39,000	3,279	37,070	6,558		C..... = 97,561
Normal slag run.....	50,117	4,738	109,506	2,818	24,835	O..... = 126,590
Normal slag to the ladle.....	25,058	2,369	54,753	1,409	12,417	H ₂ O..... = 67,568
Furnace steel.....						
Recarburizer.....	1,084	68				
Product.....	25,393	3,453	54,821	1,409	12,417	
Ladle slag.....						
To ladle gases.....						
To furnace gases.....						

TABLE VII

MATERIALS PER TON OF INGOTS. TEST OF FURNACE NOS. 89 AND 90.

Material	Pounds	In the Furnace	Pounds
Hot metal.....	1,120.00	Carbon oxidized.....	42.750
Cold pig.....	96.30	Silicon oxidized.....	13.015
Total.....	1,216.30	Manganese oxidized.....	18.620
Blooms.....	590.30	Phosphorus oxidized.....	3.436
Plate.....	19.50	Iron oxidized to FeO.....	70.660
Ingot butts.....	59.17	Fe ₂ O ₃ reduced.....	202.800
Rails.....	39.30	Fe ₃ O ₄ reduced.....	35.420
Tin scrap.....	26.61	H ₂ O liberated.....	41.820
Turnings.....	3.44	CaCO ₃ decomposed.....	64.660
Steel scrap.....	103.90		
Stirring rods.....	2.84		
Total steel scrap.....	844.06		
Fine ore.....	200.90	In the Ladle	Pounds
Cinder.....	5.63		
Scale.....	5.50		
Dolomite.....	68.70	Silicon oxidized.....	1.679
Limestone.....	153.80	Carbon oxidized.....	.975
Lump ore.....	61.90	Manganese oxidized.....	1.776
Fluor spar.....	2.75	Phosphorus oxidized.....	.412
Total.....	499.18	Aluminum oxidized.....	.254
Run off slag.....	67.50	Sulphur oxidized.....	.141
Two-thirds normal slag.....	161.15		
ADDITIONS.....			
Hot metal.....	164.75		
Ferro manganese.....	8.60		
Ferro silicon.....	.65		
Sulphur.....	.226		
Coal.....	2.135		
Aluminum.....	.254		
Total additions.....	176.615		
Furnace metal.....	2,070.00		
Product.....	2,240.00		
Difference.....	170.00		
In recarburizer.....	176.615		
To ladle slag.....	7.865		
One-third normal slag.....	80.575		
Total ladle slag.....	88.440		
Dry producer gas used.....	2,510.00		
Water (downtake).....	72.00		
Water (valve).....	96.80		
Dry air used.....	7,378.00		
Water in air.....	106.00		
Dry flue gas.....	10,004.00		
Water in flue gas.....	584.00		

Table VII gives the pounds of materials used in the production of one ton of ingots, together with the quantities of slags and gases produced.

In the figuring of the heat balance, which follows, the figuring is based on heat per ton of ingots.

The above table would also be of value in open-hearth furnace design, when considering the gases used and formed.

Table VI gives in a condensed form the arrangement of the

various materials in pounds, giving the amounts of material charged, slags formed, and steel produced, and from which table an adjustment of the oxygen was obtained. That is the amount of oxygen freed, and the amount of oxygen absorbed in the various reactions, together with the oxygen coming into the furnace from the gas mixture, gives the amount of free oxygen that enters into the burned gases passing out of the furnace.

All data taken for the figuring of the heat balances are operating figures such as are found in practice, since test conditions are often in error when taken as representing average practice.

The following heat balance naturally divides itself up into many parts, and for this reason on the left-hand side of each sheet appears the data and results in obtaining the heat involved at each stage of the process, while on the right-hand side of the sheet is printed the calculations involved in producing the result.

In the calculations effecting the heat balance the methods employed are those of Prof. J. W. Richards, as given in his most valuable work, "Metallurgical Calculations." Figures used in obtaining the mean specific heat of the materials involved, and symbolized SM, are those constants given by Dr. Richards. These constants for the different materials and elements are, in nearly all cases, figures for the specific heat at 0.0 deg. C., or 32 deg. F.; hence for obtaining the specific heat at a higher temperature the figure at 32 deg. F. is increased by the addition to this figure of the product of a known constant and the temperature range. The results are given in detail in the calculations.

In figuring the latent heat in gases due to the moisture content, the metric system, as given by Dr. Richards, is used, and the result obtained in kg. cal. converted into B.T.U.

In collecting the data it was very difficult to obtain any reliable information in relation to the moisture contents of air and gas, but the figures as here given were checked a number of times, and accordingly can be taken as representative of the conditions met with in the regenerator cellars in open-hearth plants.

It will be noted that the hot gas efficiency of the producers' figures out to be 94.99 per cent, and that the loss of heat after passing the reversing valves amounts to 3.22 per cent, so that the net efficiency of the producers delivering gas to the furnace regenerators is 91.77 per cent. The gas flues on the furnaces tested were underground, and the question naturally comes up: Would there be as great a loss of heat if the flues were carried overhead and not buried as in this case?

Producer Gas

Calculation of the Total Heat in the Producer Gas Before Being Burned

 HEAT CHART AND DISTRIBUTION. GAS UP TO FURNACE PORTS
 (A) HEAT TO PRODUCERS. PER TON OF INGOTS

Material	Lbs.	B.T.U.- Lb.	B.T.U.	T°	Range	SM (O-t) x	B.T.U. Sensible
Coal	510.0	13,427.2	6,847,872	70°	38°	0.33522	6,497.4
Steam	149.0	1,170.7	174,434	90°	58°	0.23421	27,107.2
Air (dry)	1,995.5	1,104.49	31,478	90°	16°	0.42597	193.8
H ₂ O in air	28.5			Sp. 74°	70°		
H ₂ O in coal	5.6			70°	38°	1.00000	212.8
Total latent B.T.U. = 7,053,784							Total sensible B.T.U. = 34,011.2
Total B.T.U. in materials = 7,087,795							

(B) PRODUCER GAS MADE AND IN DOWNTAKE FROM PRODUCER

Material	Lbs.	B.T.U.- Lb.	B.T.U.	T°	Range	SM (O-t)	B.T.U. Sensible
Dry gas	2,510.0	2,218.0	5,567,180	1400°	1368°	0.30130	1,034,371
H ₂ O in gas	72.0	1,109.07	79,853	89°SP	1279°	0.36090	51,646
Total latent B.T.U. = 5,647,033							Total sensible B.T.U. = 1,086,017
Total B.T.U. in producer gas = 6,733,050							
Hot gas efficiency of producer = 94.99%							

(C) PRODUCER GAS AT ENTRANCE TO REGENERATOR AND AFTER PASSING WATER COOLED VALVE

Material	Lbs.	B.T.U.- Lb.	B.T.U.	T°	Range	SM (O-t)	B.T.U. Sensible
Dry gas	2,510.0	2,218.0	5,567,180	1100°	1,068°	0.29488	779,640
H ₂ O in gas	96.8	1,112.13	107,654	99°SP	979°	0.53000	50,239
Total latent B.T.U. = 5,674,834							Total sensible B.T.U. = 829,879
Total B.T.U. in producer gas delivered to entrance of regenerator = 6,504,713							
Efficiency of producer making gas delivered to furnace = 91.77%							
B.T.U. lost in cooling up to regenerator entrance = 3,222°							

(D) GAS AFTER PASSING THROUGH GAS REGENERATOR TEMPERATURE OF REGENERATOR BEING 2,120°F.

Material	Lbs.	B.T.U.- Lb.	B.T.U.	T°	Range	SM (O-t)	B.T.U. Sensible
Dry gas	2,510.0	2,218.0	5,567,180	1,940°	1,908°	0.312774	1,497,780
H ₂ O in gas	96.8	1,112.13	107,654	99°SP	1,908°	0.616520	113,837
Total latent heat B.T.U. = 5,674,834							Total sensible B.T.U. = 1,611,617
Total B.T.U. in the gas through regenerators = 7,286,451							
Heat in B.T.U., gained in regenerator = 781,738							

CALCULATIONS

(A)	Coal.	B.T.U. = 14,544	C × 62,000 (H ₂ - O ₂ + 8) + 3,996 S.	
			= 11,086.9 × 2,294.0 + 46.3 = 13,427.2 B.T.U.	
	Steam.	Forty-five pound gauge.	Sixty-pound absolute.	
	Coal.	SM. (32° - 70°)	temperature difference = 38°	
C		76.23%	0.1567 + 38 × 0.00036	= 0.17038 0.12998
H		5.02%	3.3700 + 38 × 0.00017	= 3.37646 0.16949
N		1.38%	0.2405 + 38 × 0.000119	= 0.24053 0.00332
O		10.61%	0.2104 + 38 × 0.000010	= 0.21079 0.02236
Ash		5.65%	0.1800	0.18000 0.01017
	SM. Coal.			
Air.	SM. (32° - 90°)	temperature difference = 58°		0.33522
N		76.9%	0.2405 + 58 × 0.0000119	= 0.24119 0.19547
O		23.1%	0.2104 + 58 × 0.0000104	= 0.21100 0.04874
	SM. Air.			0.23421
	Moisture in air = 9 grains per cubic foot.	Saturation point = 74°F.		
	74°F. = 23.33°C.	606.5 + 0.305 (23.33) = 613.61 kg. cal.		
	Superheat = 90° - 71° - 16°F.		= 1,104.49 B.T.U. latent.	
		SM. = 0.42 + 0.000103 t°F.		
	Sensible heat = 0.42597 × 28.5 × 16° = 193.8 B.T.U.			

(B)	Producer gas.	SM. (0° - t°).	Temperature range = 1,368°.	T. = 1,400.
CO ₂		10.317%	0.1900 + 1,368 × 0.00006	= 0.27208 0.028070
CO		25.003%	0.2405 + 1,368 × 0.0000119	= 0.25678 0.064192
O ₂		0.25%	0.2104 + 1,368 × 0.0000104	= 0.22462 0.000055
CH ₄		1.711%	0.5300 + 1,368 × 0.000150	= 0.73520 0.012579
C ₂ H ₆		0.458%	0.3670 + 1,368 × 0.000130	= 0.54484 0.002495
H ₂		1.000%	0.3700 + 1,368 × 0.000170	= 0.60256 0.036025
N ₂		61.486%	0.2405 + 1,368 × 0.0000119	= 0.25678 0.157883

SM. Producer gas			0.301300
Gas weight = 0.071103 pounds per cubic foot.			14.06 cubic feet per pound.
Moisture = 14.3 grains per cubic foot.			14.3 grains H ₂ O = S. P. of 89°F.
89°F. = 31.65°C.			606.5 + 0.305 (31.65) = 616.15 Kg. Cal. = 1,109.07 B.T.U.
SM. (0° - t°) = 0.42 + 0.000103 t°.			t° = 1,368°.
SM. = 0.56090.			

(C)		Temperature range = 1,068°F.	
CO ₂		0.25408	0.026220
CO		0.25321	0.063302
O ₂		0.22150	0.000055
CH ₄		0.69020	0.011806
C ₂ H ₆		0.50584	0.002326
H ₂		3.51500	0.035515
N ₂		0.23321	0.155664
SM. Gas			0.294888

Moisture per cubic foot gas = 19.1 grains.			Saturated at 99°F.
99°F. = 37.2°C.			606.5 + 0.305 (37.22) = 617.83 Kg. Cal. = 1,112.13 B.T.U.
SM. (0° - t°) = 0.42 + 0.000103 t° = 0.53000.			
Superheat = 1,068 - 0° = 979°F.			

(D)		Temperature range = 1,908°F.	
CO ₂		0.30448	0.031361
CO		0.26320	0.065800
O ₂		0.23024	0.000057
CH ₄		0.81620	0.013965
C ₂ H ₆		0.61504	0.002816
H ₂		3.69436	0.036943
N ₂		0.26320	0.161832
SM. Gas			0.312774

Moisture in gas = 19.1 grains.			SP. = 99°F.
SM. = 0.42 + 0.000103 t° = 0.61652.			

Air

Calculation of the Total Heat in the Air for the Combustion of the Producer Gas

 AIR. HEAT CHART AND DISTRIBUTION
 HEAT IN AIR BEFORE PASSING THROUGH REGENERATOR

Material	Lbs.	T ₁	T ₂	Range	SM (0° - t)	Latent B.T.U.	Sensible B.T.U.
Dry air	7,378	90°	32°	58°	0.23421		100,340
H ₂ O in air	106	SP. = 74°	Superheat = 16°			117,797	
Total B.T.U. in air = 218,137							

Material	Lbs.	T ₁	T ₂	Range	SM (0-t)	B.T.U. Sensible	Latent B.T.U.
Dry air	7,378	90°	2,012°	1,922°	0.256422	3,636,573	
H ₂ O in air	106				0.627236	130,906	117,076
Total B.T.U. after regenerator = 3,884,555 B.T.U.							
Heat absorbed by the air in regenerator = 3,884,555							
Heat originally in the air = 218,137							
Absorbed = 3,666,418							

(C)	Heat in unburned gas at ports	7,286,451 B.T.U.
	Heat in regenerated air at ports	3,884,555 B.T.U.
	Heat to furnace	11,171,006 B.T.U.

 CALCULATIONS
 AIR BEFORE REGENERATOR

(A)	AIR BEFORE REGENERATOR			
N. SM (32° - 90°).				
O	76.9%	0.2405 + 58 ×	0.0000119 =	0.24119 0.1954°
	23.1%	0.2104 + 58 ×	0.0000104 =	0.21100 0.04874
SM. air				0.23421
Water in air = 9 grains.				Latent heat = 1104.49 × 10° = 117,076 B.T.U.
Superheat = 16°F.				SM. = 0.42597
Sensible heat = 0.42597 × 106 × 16°				= 21 B.T.U.

S.M. AIR

(B)	TEMPERATURE OF AIR (0. T. OF REGENERATOR = 2,012°F.				
	SM. (32° - 2,012°)				
N		76.9%	0.2405 + 1,980 × 0.0000119	= 0.264062	0.2081°
O		23.1%	0.2104 + 1,980 × 0.0000104	= 0.250992	0.05459°
	SM. Air				0.25422
	Moisture in air, 9 grains per cubic foot. Total weight = 10 pounds				
	SP. = 74°F.				Latent B.T.U. = 1,104.49 B.T.U.
	Superheat = 2,012 - 74				1,938°
	SM. = 0.42 + 0.000103 t°F				0.627236
	Latent heat in water vapor = 1,104.49 × 10°				117,076
	Sensible heat = 106 × 0.627236 × 1,938°				130,906

Materials

OPEN HEARTH. SUMMARY

a (A) HEAT DISTRIBUTION. HEAT IN MATERIALS CHARGED

Material	Pounds	T°	SM (0-t)°	SM × (T-32°) × Lbs.	B.T.U. -Lb.	Gross B.T.U.
Hot metal....	1,120.00	2050			441	493,920
Cold pig ..	96.30	70	0.12095	442.98		443
Blooms	590.30	70				
Plate	19.50	70				
Butts	59.17	70				
Rails	39.30	70				
Tin	26.61	70				
Turnings....	3.44	70				
Steel	103.90	70				
Rods	2.84	70				
Total steel ..	844.06	70	0.11650	3,714		3,714
Fine ore	200.90	70	0.14959			1,145
Cinder	5.63	70	0.14869			63
Scale	5.50	70	0.14869			
Solomite	68.70	100	0.21790			1,016
Limestone ..	153.80	100	0.20860			2,184
Lump ore	61.90	100	0.14959			619
Fluor spar ...	2.75	100	0.21540			40
Cold material. Total B.T.U.						503,144

OPEN HEARTH. SUMMARY

CALCULATIONS

a. (A)
Cold pig. SM (0°-t°) = SM (32°-70°) = 0.1200 + 0.00025 t.
= 0.12095
96.30 × 38 × 0.12095 = 442.98 B.T.U.
Steel = 844.06 × 38 × 0.11650 = 3,714 B.T.U.
Lump ore = 0.1456 + 0.000103 t = 0.14959 = SM.
Cinder scale = 0.1447 + 0.000103 t = 0.14869 = SM.

(B) HEAT GENERATED BY OXIDATION OF MATERIALS

Material	Pounds	Product	Base B.T.U.-Lb.	Gross B.T.U. Generated
Carbon oxidized....	42.750	CO ₂	14,580	623,295
Silicon oxidized....	13.015	SiO ₂	14,094	183,433
Manganese oxidized.	18.620	MnO	3,114	57,982
Phosphorus oxidized	3.436	P ₂ O ₅	10,620	36,490
Iron oxidized.....	70.660	FeO	2,430	171,704

B.T.U. generated by oxidation 1,072,904

TOTAL HEAT TO FURNACE

In gases 7,286,451 B.T.U.
In air for combustion 3,884,535 B.T.U.
In materials charged 503,144 B.T.U.
By oxidation in furnace 1,072,904 B.T.U.
Total B.T.U. to furnace 12,747,054 B.T.U.

Heat Absorbed

HEAT ABSORBED IN THE FURNACE
HEAT ABSORBED IN REDUCTION, ETC. (A) POUNDS PER TON OF INGOTS

Material	Pounds	B.T.U.- Pound	Gross B.T.U.
Fe ₂ O ₃ reduced	202.80	2,270.0	460,356
Fe ₃ O ₄ reduced	35.42	2,137.0	75,692
H ₂ O liberated	41.82	2,678.8	112,027
CO ₂ (lime) liberated ..	153.80	666.0	102,431

Total B.T.U. absorbed 750,506 B.T.U.

HEAT ABSORBED IN SLAG FORMATION. (B)

Slag	Pounds	B.T.U.- Pound	Gross B.T.U.
Run off slag	67.50	1,134	76,545
Two-thirds total normal slag ..	161.15	1,226	197,570
One-third total normal slag to ladle slag	80.57	1,226	98,779
Ladle slag	88.433	(*)	

Total B.T.U. in slags run from furnace 372,894 B.T.U.

*Not in furnace, hence not calculated.

HEAT IN THE MELTED STEEL (C).

Material	Pounds	T°F.	Total B.T.U.-Lb.	Gross B.T.U.
Dead melted steel ..	2,070.0	3,012°	622.8	1,289,196

HEAT UTILIZED IN FURNACE

Heat of reduction, etc	750,506 B.T.U.
Heat in run off slag	76,545 B.T.U.
Heat in normal slag	296,349 B.T.U.
Heat in melted steel	1,289,196 B.T.U.
Total B.T.U. utilized	2,412,596 B.T.U.

In the foregoing calculations it will be noted that the gas producers were taken as a separate unit in the amount of heat produced, and which likewise follows for the air required for the combustion of this gas. The heat in the cold materials as charged into the furnace is also considered since the normal temperature of this material is considerably above 32 deg. Fahrenheit. The heat due to oxidation of the carbon, silicon, and other ingredients in the metal by the iron ore charged is also treated separately, as it may be noted in the calculation.

The summary of the above calculations is as per the following table, which gives the total heat which is delivered to, or produced within the furnace, and the sum total of which gives the first part of the heat balance. The distribution and utilization of this heat is as per the calculations which follow.

CALCULATIONS

Water evaporation. 70°-212°. 606.5 + 0.305 (67.5°) = 627.08 Kgr.-C.
= 1,128.74 B.T.U.
SM. (O-t) = 0.42 + 0.000103 t. t = 2,500° S M = 0.67750
B.T.U., superheat = 1 × 0.67750 × 2,500° - 212° = 1,550.1
Latent 1,128.7

Total = 2,678.8 B.T.U.-Lb.

(B) SLAGS

Run off slag. MP = 1,350°F.
t° = 1,350.
MS = 0.16359 (1 + 0.00039 t).
SM = 0.16359 (1.5265)
SM = 0.24972 at MP.

RUN OFF SLAG

Analysis	%	Sp. H. 0°	SM
SiO ₂	10.02	0.1833	0.01836
Al ₂ O ₃	3.00	0.2081	0.00624
FeO	35.75	0.1460	0.05219
MnO	9.23	0.1511	0.01394
CaO	33.98	0.1715	0.05827
MgO	6.03	0.2420	0.01459

16,359

Latent heat (fusion) = (1,350 + 273) (0.24972) = 405.3 Kgr.-C. × 225 = 91.2 Kgr.-Cal.

A balance of the heats for formation = 200 Kgr.-Cal.

Total heat in slag up to MP. = 337 + 91 + 200 = 628 Kgr.-Cal.

= 1,134 B.T.U.-Lb.

t° = 1,450.

SM = 0.16970 (1 + 0.00039 t°).

= 0.16970 (1.5655)

= 0.26566

Latent heat of fusion.

(1,450 + 273) (0.26566) (0.225) = 103 Kgr.-Cal.

Lf = 103 Kgr.-Cal.

Hf = 385 Kgr.-Cal.

Hf = 150 Kgr.-Cal.

Total = 638 Kgr.-Cal.

NORMAL SLAG

Analysis	%	Sp. H. 0°C	SM
SiO ₂	15.50	0.1833	0.02841
MnO	7.91	0.1511	0.01195
FeO	19.25	0.1460	0.02810
Al ₂ O ₃	1.82	0.2081	0.00378
CaO	42.05	0.1715	0.07211
MgO	9.54	0.2420	0.02308
CaF ₂	1.08	0.2100	0.00227

0.16970

NORMAL SLAG. MP = 1,450°

Superheated to 1,600°C = 1,60° × 0.27559 =

Total B.T.U. per pound of slag 1,220

HEAT IN STEEL (C). SM = 0.19887 - (23.44 + t°) = 0.18427

Heat in melted steel at 1,150°C. = 300 Kgr.-Cal. 250° × (0.18427) = 46 Kgr.-Cal.

346 Kgr.-Cal = 622.8 B.T.U.

Table VIII gives the summary of the heat distribution and is the heat balance.

Conclusion

HEAT BALANCE.

HEAT DISTRIBUTION PER TON OF INGOTS

	B.T.U.	%
Heat in materials charged to producers.....	7,087,795	
Heat in producer gas in downtake.....	6,733,050	
Efficiency of producer (hot gas).....		94.99
Producer gas at entrance to regenerators.....	6,504,713	
Efficiency of producer.....		91.77
Heat in producer gas after passing through the regenerators.....	7,286,451	
Heat gained in the regenerators.....	781,738	
Heat in air before passing to regenerators.....	218,137	
Heat in air before passing through regenerators.....	3,884,555	
Heat gained by air in regenerators.....	3,666,418	
Heat in gas and air going to furnace.....	11,171,006	
Heat in materials charged into furnace.....	503,144	
Heat generated by oxidation in furnace.....	1,072,904	
Total heat delivered to furnace.....	12,747,054	100.00
Heat absorbed in reduction.....	750,506	5.88
Heat absorbed in slag formation.....	372,894	2.92
Heat absorbed in melting steel.....	1,289,196	10.11
Total heat used in useful work.....	2,412,596	18.91
Heat in gases passing to regenerators.....	9,715,015	76.21
Heat loss in CO gas unburned.....	53,933	.42
Heat loss by radiation, etc., furnace proper.....	565,510	4.46
Heat in gases passing out of regenerators.....	4,152,723	32.57
Heat deposited in regenerators.....	5,616,225	44.06
Heat taken out by air and gas in regeneration.....	4,448,156	34.89
Efficiency of regenerators.....		79.22
Heat loss in flue to stack.....	287,373	2.25
Heat going up the stack.....	3,865,350	30.32
Total heat delivered or generated in furnace.....	12,747,054	100.00
Useful work.....	2,412,596	18.91
Radiation at furnace.....	565,510	4.46
Heating of regenerators.....	5,616,225	44.06
Radiation in flue.....	287,373	2.25
Loss up stack.....	3,865,350	30.32
Total.....	12,747,054	100.00

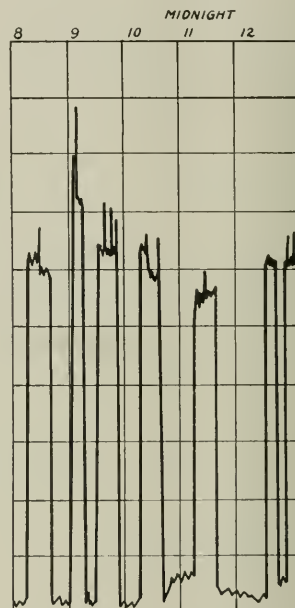
From the above heat balance it will be noted that the amount of heat actually used in the melting of the steel amounts to but 10.11 per cent of the heat in the fuel, and the total amount of heat used in performing useful work is 18.91 per cent, that is, the efficiency of an open-hearth furnace, giving the furnace credit for reduction, slag formation, and melting metal, is less than 20 per cent. There are a number of ways of utilizing the waste heat from the furnace, one of which is the utilization of waste heat boilers connected to the stack. Aside from this, and considering the furnace as a heating unit, there is great room for improvement in the furnace itself.

In modern boiler houses and engine rooms there are gauges, etc., giving information as to the working of the apparatus, and in many plants there are steam engineers that make regular use of the indicator as a guide to engine performance, but the use of gauges, etc., in the open hearth, has unfortunately not met with a favorable reception. Some plants have meters on their gas lines to each furnace, but this is the exception and not the rule, and where the meters are applied they are on natural-gas furnaces.

The value of having a recording mechanism on a furnace may be noted from the following. On one of the furnaces from which this heat balance was figured, there were 509 reversals for the week. The average time for each reversal was 10 minutes, the longest period on one set of regenerators was 70 minutes, and the shortest period was 2 minutes. In open-hearth practice it is the custom for each turn to keep up one end of a furnace. That is the night shift might have the left end of a furnace and the day shift might have the right end of a furnace to keep up. One turn naturally favors its end of

the furnace, and in the operation of the furnace will do all that it can to favor its side and keep it from cutting away, thereby placing all the wear and tear on the other end, as the case may be.

This practice leads to irregularity in the operation of the regenerators, and soon there is a good and a bad end to the furnace. The adjoining illustration shows the difference in time involved in the reversals. A long period and then a short period on a regenerator naturally does not give a proper distribution of heat to the regenerators, and one set of regenerators will be at a lower temperature than the other. At certain periods in an open-hearth heat it is necessary to have quick reversals, but there is no reason why these reversals



should not and could not be made of equal duration for each set of regenerators, thereby maintaining them at a balanced temperature.

If it were the practice to have all furnaces equipped with a recording mechanism which would show the reversals, the superintendent would be aware of what took place at all hours, and, by paying more attention to the utilization of heat in a furnace, would have a very good opportunity to effect economy.

The above heat balance was for a set of two furnaces of 60 tons capacity, with a hearth 32' 0" x 14' 9", such a 60-ton furnace showing an efficiency of 18.91 per cent.

There is a wide discrepancy in open-hearth design, as may be noted by the following.

Plant.....	Capacity.....	Hearth.....
Duquesne.....	60 Ton.	32' 0" x 14' 0"
Ohio Works.....	60 Ton.	29' 0" x 14' 11"
Gary.....	60 Ton.	40' 0" x 15' 3"
Republic.....	60 Ton.	32' 0" x 15' 0"
Lackawana.....	60 Ton.	43' 0" x 16' 0"

There is an even greater variation in the dimensions of the regenerators, flues, etc., in the above furnaces than there is in the hearth, hence it would follow that the utilization of heat would be different, and it would be profitable to all steel manufacturers to know just what design would be the most effective for the result desired. The logical way to determine this is from a metallurgical standpoint, which in this case would be in heating efficiency.

Pittsburgh, Pa.

Rational Absorption of Hydrochloric Acid*

By Dr. Theodor Meyer

Whoever tries to acquaint himself with the methods and apparatus for the production of muriatic acid from hydrochloric acid gas and reads the chapter of more than 100 pages in Lunge's great treatise may easily have the sensation of wheels turning in his head, as said by the scholar in Goethe's "Faust." Surely, I do not want to depreciate Lunge's excellent work as the completeness and critical exactness of his exposition of the subject can hardly be surpassed. But the processes and apparatus discussed by him are of such a variety, the propositions and views even partly contradictory and often irrational, that much acumen and care is necessary to discover what is suitable and rational in the chaos.

As Lunge shows, even Hurter's writings and experiments based on sound science and on practical experience contain many errors and lead in part to wrong conclusions such as might not be expected from an expert like Hurter. A point in question is, for instance, the representation of the coke tower as the most excellent condensation apparatus. The acme of absurdity is the "invention" of Schloesing, who finds pre-cooling of the gases to be detrimental and concludes that to

sorption of hydrochloric acid is the content of HCl in the gas, apparatus for the production of muriatic acid from hydrochloric acid gas and reads the chapter of more than 100 pages in the gas mixture.

These two laws are the essential basis of the rational absorption of hydrochloric acid. They are illustrated by the curves in Fig. 1 which have been plotted according to the formula of Hurter:

$$C = (0.3040 - 0.0016 t) P^{0.18}$$

in which C is the grams of HCl absorbed by 1 gram of water at the temperature t and P is the partial pressure of the HCl gas in millimeters of mercury.

The most important rule which may be deduced from these curves for the requirements of practice is to keep the concentration of HCl in the gas mixture at not less than 20 or 30% if possible.

Besides the above considerations a number of other points must also be considered, especially the influence of the diluting air on the heat phenomena and the content of water in the gas mixture; the effect of cooling on the water precipitation by which latent heat is again set free; the speed of the gas; the impurities in the gas; the relation of hydrochloric acid of different concentration on the excess of air, and finally the particular properties of the materials used for the apparatus.

Above all it is clear that a good and quick absorption of the HCl can only be accomplished by most intimate contact of the gas with water. This contact may be provided in general by three methods which are different in principle.

(1) The gas is subdivided into very small bubbles and is passed through the water by pressure or suction.

(2) The water is subdivided into exceedingly small droplets and made to act on the gas.

(3) As large a liquid surface as possible is provided to be acted on by the gas.

The method (1) is useful in the laboratory, but is not suitable for manufacturing on a large scale, since if the depth of the liquid is low the gas bubbles pass through it too quickly, so that there is not sufficient time for an intimate contact. On the other hand, if the depth of the liquid is great a rather considerable amount of work would be required for the gas to overcome the pressure of the liquid.

The method (2) was discussed unfavorably by Hurter in 1893 (*Journal Society Chemical Industry*, page 226). His criticisms were justly refuted by Lunge, though they could be understood since the patented process of Newall and Bowman of 1874 had not proven successful. But two points must be taken into consideration in explaining this lack of success. Firstly, at that time no direct means had yet been evolved to subdivide liquids into such a very fine spray as is possible to-day. Secondly, Newall and Bowman tried to apply the method in a wrong manner. It is well known that muriatic acid with more than 25% HCl loses HCl when in contact with air at a temperature of 15° C. (59° Fahr.) and this loss of HCl is, of course, the quicker the more intimate the contact with the air. As the temperature increases that strength of acid which will remain constant in the presence of air is still further reduced. For instance, at 60° C. (140° Fahr.) it is 23% HCl. This shows that muriatic acid of at least 32% (20° Baume) could be obtained by means of a water spray only if the HCl concentration in the gas is correspondingly high.

But, even in this case, trouble is caused by the heat of absorption which manifests itself the more, the less diluent gases are present which could take it up. These two points were not sufficiently taken into consideration by Newall and Bowman. As they carried out the process the spray of water acted directly on the gases coming from the furnace and this reaction took place in sandstone vessels which were connected by means of stoneware pipes, so that there was little opportunity for the escape of the heat evolved. We will show later that the use of spray water can give very good results if it is applied in the correct way and at the correct place.

The method (3) is the one which is being generally and almost exclusively used at present, although details of operation

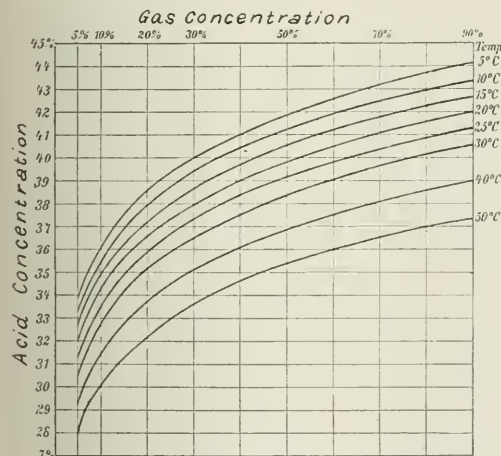


FIG. 1.—ABSORPTION CURVES FOR DIFFERENT GAS CONCENTRATIONS AND TEMPERATURES

aid the absorption of the gases their temperature and vapor pressure should be as high as possible.

It is well known that the absorption of hydrochloric acid is an exothermic reaction. According to Berthelot when x grams of HCl are dissolved in a gram of water $477.5x - 157x^2$ calories are produced; for instance, the heat evolved in the formation of hydrochloric acid of 20° Baume corresponds to a rise of temperature of 118° C. (242° Fahr.). It is therefore evident that the absorption will be easier and quicker the better means are provided to carry off this newly produced heat.

But the maximum concentration of the hydrochloric acid produced also depends on the temperature. For instance, when the hydrochloric acid is absorbed from the gas of 5% concentration at 20° C. (68° Fahr.) it is still possible to produce acid of standard concentration (32% HCl), while at a temperature of 50° C. (122° Fahr.) a higher concentration than 28% cannot be obtained. On the other hand, at 5° C. (42° Fahr.) a concentration of almost 34% can be obtained. According to Hurter's calculations, at a temperature of t ° C. one gram of water is able to absorb $0.3040 - 0.0016 t$ gram HCl.

The second most important factor which determines the ab-

*Translated from *Zeitschrift für Angewandte Chemie*, February 21, 1913, p. 97.

vary greatly and apparatus of greatly different forms are employed, such as towers with coke filling or stoneware filling, Lunge plate towers, stoneware vessels, and stoneware tourills, and Cellarius cooling and absorption vessels.

In England, the mother country of the manufacture of hydrochloric acid, the big coke towers in connection with some stoneware vessels have remained chiefly in use. On the other hand, in continental Europe stoneware tourills with a final washing tower of stoneware of moderate dimensions are pre-eminent. The Schaffner (Aussig) system with towers of 8 to 12 meters (26 to 40 feet) height filled with stoneware dishes are in particular favor.

The large coke towers, which have a height up to 36 meters (120 feet), are generally built of sandstone, so that there is little chance for the air to produce a cooling action. If the uncooled gases enter such a tower at the bottom a moderate cooling effect is obtained by the evaporation of the muriatic acid trickling down from above so that the heat becomes latent and as such is returned to the higher portions of the tower where condensation takes place and the latent heat becomes free again, while the condensed acid trickles down again, evaporates partly anew and so on.

The net effect of what happens is, therefore, that the heat is transferred to a somewhat higher point, while the real condensation process is restricted to the highest and coolest part.

The hydrochloric acid formed in the highest and coolest part of the tower must, therefore, pass through the lower hot portion, and as the constant concentration of muriatic acid at the boiling point is not more than 20% or 13° Baume, it is evident that it is very difficult to get in this way a fairly high concentration of the acid.

Moreover, in many cases the tower is not even able to reduce the concentration of HCl in the exit gases to the desired degree, so that it becomes necessary to pass the gas finally through a washing tower and the low acid obtained from it is a source of trouble.

Formerly the hydrochloric acid was used principally for the Weldon process, especially in England, and for this purpose a concentration of 16 to 18° Baume was sufficient. At present the conditions have been changed, especially in continental Europe, where the Weldon process has been replaced almost completely by the electrolytic process. To have a commercial value, hydrochloric acid should have at present a concentration of not less than 20° Baume, and for some purposes, like the manufacture of aniline salts, a considerably higher concentration, up to 24° Baume, is required.

The system of a number of tourills with a final washing tower, which is used largely in Germany, is undoubtedly superior to the large coke tower, since part of the sensible heat and of the heat of absorption is carried off in advance by the tourills with their relatively large outside surfaces and since the rather thin stoneware walls of the tower carry off far more heat than the sandstone walls in the British system, which are six times thicker.

Nevertheless, even in the German system about one-half of the absorption still takes place in the towers, especially if the inaccessible surface of the liquid in the tourills becomes covered with fatty impurities. This trouble occurs often when some oil or grease is put into the saltcake pan to prevent ebullition. In such cases a failure of the tourills often occurs and it may happen that in order to get the acid up to 20° Baume it must flow off at 15 to 17° Baume from the tower and that the tower becomes so hot as to crack the walls. In a system designed in a rational manner with pre-cooling and cleaning of the gases the acid flowing off from the tower has a concentration of only 1 to 2° Baume.

Effective pre-cooling cannot be obtained with air, especially with chemical stoneware as the only material that is really to be taken into consideration a wall-thickness of 10 to 15 millimeters (0.4 to 0.6 inch) must be used. The specific heat of water is four times that of atmospheric air if figured per weight, and more than 3,000 times that of atmospheric air if figured per volume.

After Fryer and others had made unsuccessful experiments, water-cooling was successfully introduced in connection with the invention of the absorption vessel of Lehmann and Cellarius in 1899, although their absorption vessel was originally not intended for water-cooling and could not be used with water-cooling (*Angew. Chem.*, Vol. 21, page 1069, 1908). When in 1901 I had to solve the problem to get hydrochloric acid of 21° Baume from the gas of the Oehler muffle furnace, I tried to employ the new Cellarius absorption vessel by modifying its construction for water-cooling. The success was excellent.

The gas coming from the furnace, after having been cooled to air temperature by passing it through 11 empty water-cooled vessels, was passed through a system of 35 absorption vessels placed in water circulating in the opposite direction. In spite of pre-cooling and absorption cooling, a rise of temperature up to 80° C. was still obtained. This occurred approximately in the twelfth vessel and proves the intensity of the absorption. The acid running off from the front vessel could easily be obtained with a concentration 23.5 to 24 or even 24.5° Baume. This concentration changed, of course with the fluctuations in the concentration of the gas, which amounted to about 40%.

If the maximum temperature is displaced toward the exit, this should be taken as a warning to restrict the water supply somewhat. It is thus possible to control the operation of the absorption process with this type of apparatus by means of a few thermometers, which is much simpler than the measurement of the specific gravity, although the latter may also be observed in a convenient manner.

After having been passed through the water-cooled vessels the acid was passed through a few air-cooled vessels and then through a washing tower, which yielded an acid of 1.5° Baume when supplied with pure water.

In 1902 C. Uebel devised a vessel similar to the Cellarius tourill which is, however, not intended for circulation of liquid through it, but especially for cooling of gases and vapors, for instance, in nitric acid manufacture. In order to distinguish it from the Cellarius absorption tourill it might be properly called a condensation tourill. It is being used now generally for pre-cooling the hydrochloric acid gases which it robs of their sensible heat and latent heat, while the Cellarius absorption tourill takes away the heat of absorption while it is being formed.

In the course of time several more or less different forms of vessels have been designed, which, however, have no advantages over the original patented construction. Yet it cannot be claimed that the Cellarius tourill already satisfies all requirements.

In an ideal absorption vessel the gas and the liquid should act on each other in counter-current. Since the absorption makes the liquid heavier the liquid should enter at the top and leave the vessel at the bottom. In the Cellarius vessel, however, the liquid enters and leaves at the bottom. Further, the liquid should enter where the gas goes out and the liquid should flow off where the gas enters. In the Cellarius vessel, however, the liquid enters and leaves alternately at the entrance and exit of the gas. The liquid, therefore, circulates in one-half of its vertical as well as horizontal passageways only in counter-current with the gas, and in the other half flows in the same direction as the gas.

Moreover, an effective counter-current arrangement is rendered difficult by the shortness of the passageway of the gases, which, measured between the centers of the gas inlet and the gas outlet, is only 50 centimeters (19 inches) for size I and 70 centimeters (28 inches) for size II.

This short length of the Cellarius vessels has another disadvantage. The front side of the vessels should be accessible by means of a passageway, say 1 meter (3.3 feet) wide, while the width of two rows of tourills with their ends placed against each other is 225 centimeters (7½ feet), including cooling boxes for size II, so that almost one-third of the floor space of the building is required for the passageways.

Finally, it cannot be denied that the form of the Cellarius

tourill is a little complicated, or at least that a simpler form could surely be made at less cost.

The design shown in Fig. 2 is intended to realize the advantages of the Cellarius tourill while avoiding its disadvantages. The gas passes here in the ordinary manner from the back to the front of the absorption vessel, then from the front to the back in the next absorption vessel, and so on. The passage way of the gas in each vessel is 140 centimeters (55 inches). The gas and the liquid are always in counter-current, since the liquid also enters the suc-

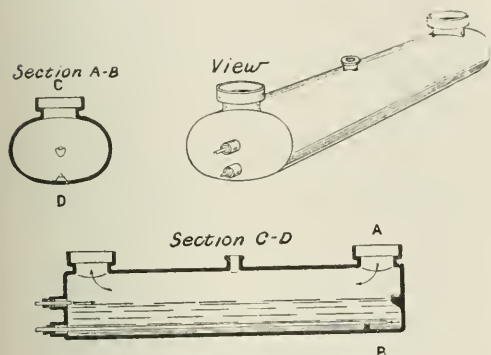


FIG. 2.—ABSORPTION VESSEL FOR HYDROCHLORIC ACID

cessive absorption vessels alternately at the front and at the back. The liquid always enters near the surface and leaves at the bottom. As these absorption vessels are 160 centimeters (63 inches) long, hardly one-fourth of the floor space of the building is required for the passageways. On account of their simple form the manufacture of these absorption vessels involves no difficulties.

An advantage of the Cellarius vessel is the quiet surface, which is due to the fact that the liquid enters and leaves below the surface. The minimum height of the level of the liquid is the height of the saddle. Since my new absorption vessel has no saddle, it might seem that the level of the liquid could not be maintained properly. However, it is easy to see that the level depends entirely upon the drop from the exit from one vessel to the influx into the next vessel, or in case of the last vessel, upon the inclination of the exit pipe. If the vessels are properly arranged with a descent of 10 to 15 millimeters (25/64 to 19/32 inch) from one vessel to the next vessel, it is easy to adjust the level of the liquid so that it is a few millimeters above the influx.

In one respect the quiet surface of the liquid is liable to become a source of trouble. If the gas is not free from particles insoluble in water and if it contains especially fatty particles, these are liable to form a film on the surface of the liquid so as to prevent absorption. When this happens it is easily observed by the fact that the vessel is cool. Even when using pure materials this was found to happen, and the cause of the trouble was discovered to be a film of a white substance like paraffine, which seemed to have had its origin in the rock salt. By means of a gas filter such troubles can be avoided with certainty.

Instead of two gas inlet and outlet pipes three or four may be easily attached to each absorption vessel, if this is desirable for the distribution or the mixing of gases. If the vessel is intended to serve for condensation only, the only change necessary is to omit or to close the upper pipe for the entrance of the liquid. The same vessel may, therefore, be used for absorption as well as condensation.

While an extended quiet surface of the liquid is most suitable for the production of a strong acid it is by no means suitable for extracting the acid from dilute gases, which becomes necessary at the finish of every condensation process.

At that point it is proper to use a finely sub-divided spray of water or dilute acid whereby a most intimate mutual contact can be obtained without danger of reducing the efficiency by the excess of air or by the heat of absorption, which is hardly noticeable in this case.

In the following I give comparative data for both types of absorption vessels, the figures of the Cellarius tourill referring to size II.

Cellarius Tourill II.	New Absorption Vessel.
Length 103 cm., or 40 in.	160 cm., or 63 in.
Width 66 cm., or 26 in.	55 cm., or 22 in.
Total Volume	
110 liters, or 29 gallons.	173 liters, or 45.7 gallons.
Liquid Volume	
40 liters, or 10.6 gallons.	88 liters, or 23 gallons.
Gas Volume	
70 liters, or 18.5 gallons.	85 liters, or 22.5 gallons.
Absorption Surface	
0.55 sq. m., or 6 sq. ft.	0.88 sq. m., or 9.5 sq. ft.
Cooling Surface	
1.4 sq. m., or 15 sq. ft.	2.3 sq. m., or 25 sq. ft.

Since only above 24 to 25% muriatic acid loses in concentration by contact with air it might appear practical at the first glance to carry out the absorption process by means of a spray of water up to that limit and to absorb only the balance by means of a quiet surface of water. But this would be a mistake. Firstly, because the absorption apparatus for spray liquid cannot be easily constructed for water-cooling and, secondly, because the spraying requires power, which means an extra expense.

So far the absorption of the last traces of acid from the gas has been always carried out in washing towers as far as possible. Undoubtedly, in a tower there is a much more intimate contact between the gas and liquid than in vats or tourills, but not nearly as intimate a contact as in spray apparatus.

Experiments made on a large scale have given, for instance, the following figures for the concentration per cubic meter: before the spray cylinder 56 grams HCl, 1.9 grams SO_2 , 3.5 grams SO_3 , and behind the spray cylinder 2.2 HCl and 28 grams ($\text{SO}_2 + \text{SO}_3$) figured as SO_3 .

The final washing tower is in a sort of an intermediate position and has, therefore, no right of existence whatever for the absorption of hydrochloric acid. Where the object is to get the last traces of acid from the gases and to enrich the dilute acid thereby obtained up to a

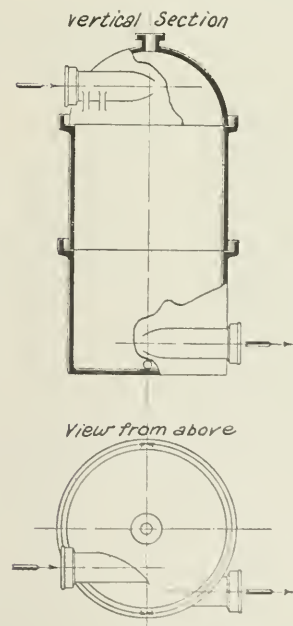


FIG. 3.—SPRAY APPARATUS

concentration of 4 to 6° Baume the most preferable procedure is to use water spray process and to circulate the obtained dilute acid directly through the absorption vessels in order to bring it up to the concentration which can be obtained with the acid concentration prevailing in the gas.

In conjunction with the Deutsche Ton- und Steinzeugwerke

in Charlottenberg* I have designed an apparatus for the treatment of dilute hydrochloric acid gas with spray water.

The principles of this apparatus for which a patent has been applied is based on the following:

(a) The gas and the spray of water are to be brought into as intimate contact as possible so that the gas is "hydratized," if we may use this expression.

(b) There should be no "dead corners" where there is no reaction, nor should the gas have any opportunity to avoid in part the action of the spray of water.

(c) The gas should be brought in a very slow and uniform current in contact with the spray of water.

(d) The spray of water should be in excess.

For this purpose the pipes through which the gas enters and leaves the apparatus are arranged in a tangential direction, the one at the conical or hemispherical top of the apparatus, the other at the bottom, while the spray nozzle yielding or emitting the water spray is best placed in the center of the top. (Fig. 3.)

The capacity of the nozzle is chosen in such a way that it gives almost the whole, but a little less than the whole quantity of the water which is necessary for recovering the HCl from the furnace in form of muriatic acid of the desired concentration. The hydrometer or thermometer readings determine the exact adjustment of the operation, which is carried out by supplying the small additional necessary amount of water directly into the last absorption vessel. The regulation of the acid concentration is, therefore, very much simpler than with the present concentration system with the final tower where it is necessary to ascend the tower in each case.

Especially with high coke towers this is very inconvenient. It should be pointed out that it is not sufficient as Lunge writes (*Handbuch*, Vol. II, third edition, page 375) to control the concentration of the finished acid several times during the day, for when this is found too low it is then too late to repair matters since the whole apparatus is filled with acid of too low a concentration. The daily measurement and adjustment of the supply of water to the tower is therefore absolutely necessary. This is, of course, also necessary whenever there is any change in the amount or number of the charges.

In the final tower (excepting, perhaps, Lunge's plate tower) about one-half of the volume is occupied by the filling material. For this reason and on account of its low efficiency it is necessary to make the tower high, several times higher than the spray apparatus. The lifting of the water to this height entails an expense which is practically as much as the expense for generating the pressure in the spray apparatus.

For towers of chemical stoneware the width is limited for constructive reasons, the maximum being about 1.4 meters (4.6 feet). Since about one-half of the cross-section in the same is taken up by the filling material, we have, with the spray apparatus of the same width, double the effective cross-section, while the velocity of the gas is reduced to one-half. However, the greatest advantage of the spray apparatus is the simplicity and cheapness of construction as compared with the final tower with its heavy foundation and high frame work and stairways.

According to the size of the hydrochloric acid plant and other circumstances one or more spray apparatus may be used and these may be connected in series or in parallel.

*The European Associates of the German American Stoneware Works.

It is advantageous to place behind the spray apparatus one or more gas filters, filled with granulated acid-resisting material, in order to catch the acid particles contained in the gases as mist (*Angew. Chem.*, Vol. 19, page 1313, 1906).

By using this new spray apparatus it is possible to recover the acid in a much more complete manner and to clean the exit gases in a much better way than heretofore. The acid is completely removed from the gas so that it is possible to let the gas pass directly into the atmosphere. It is not necessary to pass the gas into the smokestack, thereby greatly impairing the draft, so that it is necessary to build it higher.

The other rules which are to be taken into consideration for the absorption of hydrochloric acid, as well as constructive details, have been described by me elsewhere (*Th. Meyer, Sulfat und Salzsäure, Halle, 1907*).

I will, therefore, discuss only a few points bearing on the present subject.

The principle is essentially as follows: The more or less wet and hot gases, having been brought to a concentration as high as practicable, pass from the furnace with the least possible artificial suction into the pre-cooling apparatus. This consists of a preliminary empty vessel and a series of condensation tourills.

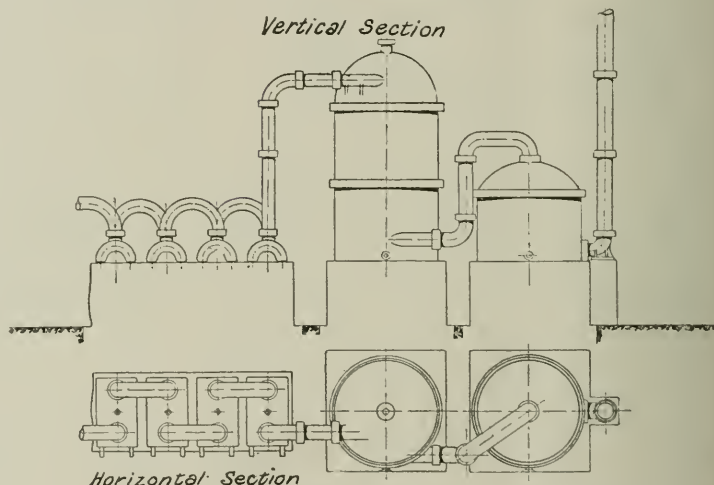


FIG. 4.—VERTICAL AND HORIZONTAL SECTIONS OF FINAL PART OF APPARATUS

At the beginning of this system a little water is added if necessary. The gases then pass through a stoneware exhauster and immediately afterwards through a gas filter; after having thus been purified and cooled, the gases enter into the series of water-cooled Cellarius tourills or absorption vessels of my approved design described above. (Fig. 3.)

They finally pass through one or more water spray apparatus and if necessary through a final gas filter. Fig. 4 shows the horizontal and vertical section of the final part of the apparatus.

With respect to the concentration of the hydrochloric acid gas, the apparatus for the decomposition of the sodium chloride by sulphuric acid is of chief importance. The reverberatory furnace which yields the most dilute gas is rarely found now, at least in Germany, but in the same class there are the gas from the Hargreaves process and the gas from the decomposition of magnesium chloride which is carried out on a commercial scale in the Stassfurt district, where the condensation of the HCl vapors is indeed very difficult.

The gas from the muffle is also very low in concentration. According to Lunge (*Handbuch*, Vol. II, 3 Aufl., pages 276 and 277) the average concentration was 1.3% HCl in one case and 2.6% in another case in the Chemische Fabrik Griesheim. According to the curves in Fig. 1 it is impossible

to get a 32% acid from such a gas, even if a very low temperature is employed such as is impractical on a commercial scale.

If the much richer gas from the pan which contains from 10 to 30% HCl would be mixed with the gas from the muffle a gas would be obtained from which acid of normal concentration could be obtained (although the concentration of the mixture would not be the arithmetical mean, since the volume of gas from the muffle is eight times that from the pan).

Usually, however, this mixing is undesirable, because pan acid and muffle acid are of different purity.

There is, however, another very simple way which is apparently little known; to bring the muffle acid up to 20 or 20° Baume. It is simply necessary to place the first one to the three tourills carrying the muffle acid in front of the absorption battery on the pan side. The pan acid is, of course, not connected into these tourills. I do not know if this easy way is used in factories.

It is certainly preferable to operate in advance with concentrated gases. I have worked on this problem for many years. The result has been on one side the Oehler furnace for bisulphate and on the other side the mechanical muriatic acid retort (German patents 186,398 and 238,570).

The mechanical retort requires sulphuric acid of approximately 90%, but the product of absorption is in proportion less impure.

With the Oehler furnace, consisting of several connected muffles, it is possible to get a gas containing 40 to 50% HCl in the average. With the retort it is possible to get an acid of 60 to nearly 100%.

If the retort is placed behind the Oehler furnace so that the firing of the Oehler furnace also serves for the retort and if the bisulphate coming from the Oehler furnace is further treated in the retort this combination represents a complete substitute for the ordinary saltcake furnace and besides other advantages it is able to yield a HCl gas of high concentration which can be easily condensed.

The pipes which conduct the gas from the furnace are liable to break pretty often, whether they are made of fire-clay, stoneware, glass or quartz. In order to avoid this it is advisable to insulate them well up to the point where they enter the pre-cooling apparatus, if it is not preferred to make them of sandstone.

Directly next to the preliminary vessel there is the battery of pre-coolers. In this combined system, as well as in the exhauster and the gas filter, so much impure acid is condensed as corresponds to the humidity and the temperature of the gas.

With the saltcake pan, the Oehler furnace, and also with the retort, the condensed acid amounts to 3 to 6% of the total acid. From the much more dilute and drier muffle gas not enough acid would be condensed; moreover, the muffle gas would enter the pre-cooling system at too high a temperature. For this reason a little water is added through the nozzle into the preliminary vessel, which must have for this purpose an inside height of 1.5 to 2.5 meters (5 to 8 feet). This water spray has a cooling action. Part of the heat is rendered latent and the water vapor thus formed is taken along to the center of the cooling system where it is condensed together with sulphuric acid and other impurities.

If the cooling system is arranged with cooling water running uniformly in counter-current so that it leaves the first vessel at a temperature of 80 to 100° C. (176 to 212° Fahr.) the gas may enter at a temperature of 200° C. without any danger of breaking.

The gas filter was first used in hydrochloric acid manufacture by Boulouvard (French patent 284,130, 1899), but at the wrong place, since it was placed directly after the furnace. But the gas filter can be efficient only when acting on the cooled gas. The particular object of the gas filter is to precipitate particles carried along in a misty form. Together with these acid particles the greatest part of the impurities is precipitated, especially sulphuric acid, iron chloride, etc., as well as some of

the arsenic. But the filter also serves to catch any fatty particles which would prevent the absorption.

One of the chief requirements of proper operation of the apparatus is the dimensioning of the pipes and of the draft. Stoneware pipes cannot be built like lead or iron pipes for any diameter. Hence the rule not to make the units of a hydrochloric acid plant too large.

Further, when designing a new plant, care must be taken to calculate as exactly as possible the quantities of gas which will be evolved per unit of time and to calculate from this the speed of the gas per second. The cooling action and condensation, as well as the absorption, depend, of course, under otherwise identical conditions, on the time required for working off the charge.

There are many sins committed against this principle especially with ordinary saltcake plants, in so far as tourills and towers of the same type and size are used for the absorption system of both pan and muffle and only the number of tourills connected in series (instead of in parallel) is sometimes somewhat larger for the muffle. In the illustrations of Lunge, for instance, figures 138 and 152, both the pipes for the pan gas and the muffle gas are exactly the same kind and, therefore, of the same cross-section.

As an example, let us take a large saltcake furnace in which 5,300 kilograms of sodium chloride are treated in 24 hours producing about 1,930 cubic meters HCl at 6° C., distributed as follows:

From the pan 1,160 cubic meters HCl = 5,800 cubic meters gas at 20% concentration at 0° C. = 8,560 cubic meters gas at 130° C.

From the muffle 770 cubic meters HCl = 38,500 cubic meters gas at 2% concentration at 0° C. = 62,580 cubic meters gas at 170° C.

Therefore the rate per second is as follows:

Pipe line from pan: 99 liters gas after the furnace = 66 liters after the pre-cooling system.

Muffle pipe line: 724 liters gas after the furnace = 447 liters after the pre-cooling system. If an internal diameter of 25 centimeters (9 $\frac{7}{8}$ inches) is assumed for the pipes carrying the cooled gas, we find a speed of flow of the gas of 1.37 meters per second in the pipes for the pan gas and a speed of 14.77 meters per second in the pipes for the muffle gas.

Even if, for the muffle gas, the pipes of 25 centimeters diameter should be replaced by pipes of 45 centimeters (18 inches) diameter (the largest which are available) the speed of the muffle gas would still be 4.55 meters per second, that is, three times the speed of the gas from the pan.

Pipes with a diameter of 45 centimeters (18 inches), however, can hardly be used between the Cellarius tourills, even if the largest size is chosen, since the gas inlets and outlets have a diameter of only 30 centimeters (11 $\frac{3}{4}$ inches).

There is, therefore, nothing left but to split up the absorption system for the muffle gas into two or more lines in parallel. For instance, for the construction of the gas filter, as made by the Deutsche Ton- und Steinzeugwerke according to my design, I always specify one filter for the pan and three in parallel for the muffle gas.

Even the best condensation apparatus cannot be maintained absolutely tight. Always a little air will enter and the more air will enter in the greater the suction. For this reason it is very important to maintain the draft not higher than is necessary.

The new spray apparatus can also be used in combination with the exhauster and gas filter advantageously for the auxiliary condensing system, the object of which is to avoid the escape of acid vapor during the operation when the furnace doors are open.

The dilute hydrochloric acid obtained in this way (for instance, in an Oehler furnace 3.7° Baume, 48% HCl, 0.24% H₂SO₄, and 0.007% Fe₂O₃), may be used even though not especially pure, as an addition to the main absorbing system so as to increase the total output of muriatic acid.

The great problem of the muriatic acid industry—the

mechanical furnace—is still a problem of the future and for the present there is little hope that it will soon be solved. But, even so, I trust I have shown that there are still many promising problems to be solved with respect to increasing the efficiency of operation in mechanical as well as hygienic respects.

Offenbach on Main, Germany.

An Electrolytic Theory of the Corrosion of Iron.*

By Bertram Lambert

The application of electrolytic ideas to the question of the corrosion of iron has been discussed by Whitney,¹ Cushman,² Tilden,³ Walker,⁴ Armstrong⁵, and many others, but much confusion and misunderstanding has arisen owing to the different interpretations used by the various writers.

The American writers have discussed the question from the standpoint of Nernst's conception of the source of electromotive force between a metal and a solution. It seems to follow from their reasoning that all iron (even if it be chemically pure and physically homogeneous) ought (a) to rust in contact with air (or oxygen), (b) to cause the deposition of copper from solutions of copper salts.

The present writer's experiments⁶ have proved beyond any doubt that iron can be prepared which will not rust on long exposure to air and water (either distilled or ordinary tap-water), and will not cause the deposition when placed in even strong solutions of copper sulphate or copper nitrate under ordinary conditions.

It also seems to be thought by many supporters and opponents of electrolytic theories that any electrolytic theory of the corrosion of iron must "stand or fall" on the question of the solubility of iron in pure air-free water.

The object of this paper is to show that a simple and natural development of the ideas of Faraday on electrolysis will give us the beginnings of a satisfactory theory of the corrosion of iron—a theory incomplete as yet owing to the lack of experimental facts, but one which is quite in accordance with well-established facts and which is not affected by the question whether iron is soluble to any appreciable extent in pure air-free water.

It is well known that while ordinary commercial impure zinc readily dissolves when placed in dilute sulphuric acid, the pure metal is scarcely affected unless metallic connection is made between it and another more electronegative metal, either within or without the liquid. Under these conditions the pure zinc dissolves, but the action ceases immediately the metallic contact between the pure zinc and the more electronegative element is broken.

In impure zinc the more electronegative element is already present within the mass of the metal, in the form of minute impurities, such as copper or lead, or perhaps, as another form of the metal which is relatively electronegative to the main mass of the metal.

Whenever we have two metals (or two modifications of the same metal) which are electrically different, that is, have different solution pressures, and they are placed in metallic contact in an electrolyte, then the relatively electropositive part will dissolve with the production of an electric current flowing through the electrolyte from the electropositive to the electronegative pole, and in the positive direction through the metal.

The rate of such a reaction depends: (a) on the magnitude of the difference of electric potential—that is, the difference of solution-pressure between the two metals or the two modi-

fications of the same metal, and (b) on the resistance offered by the electrolyte to the passage of the electric current.

No part of the metal can dissolve unless an electric current actually passes through the electrolyte. The rate of the reaction may, however, be so infinitesimal that the amount of metal passing into solution will not be sufficient to respond to chemical tests even after long periods.

Let us consider, in this light, the case of a piece of ordinary commercial iron in contact with the electrolyte water.

Such a piece of iron is impure and not homogeneous—there are some parts of it which have a different solution-pressure from other parts, and so when it is placed in contact with the electrolyte water we have all the conditions for the production of an electric current.

If the conditions are such that an appreciable electric current can pass between the two electrically different parts of the iron the metal will dissolve at the relatively electropositive parts. The fact that iron is practically insoluble in pure water (in the absence of oxygen) shows that the current which actually does pass is so infinitesimal that the amount of iron dissolved cannot be detected, even after long periods, by chemical means.

This may be due to two causes, namely, (a) the small magnitude of the electromotive force owing to the small differences of potential between the electrically different parts of the metal, and (b) the great resistance offered to the passage of an electric current by the electrolyte.

The writer's experiments⁷ seem to be generally accepted as proving beyond any doubt that commercial forms of iron will undergo corrosion quite readily in contact with pure water and pure oxygen in the complete absence of carbonic acid or any other acid—that the only essentials for the corrosion of ordinary iron are water and oxygen. It is generally believed that iron must pass through a process of solution before rust is produced, and so, while the metal is practically insoluble in pure water alone, it must be soluble in the presence of oxygen. It follows, then, that oxygen must bring about some alteration in the conditions of the "voltaic circle"—commercial iron and water—in such a way that a greatly increased electric current passes between the electrically different parts of the iron.

That oxygen can bring about such a change of conditions is beautifully illustrated by an experiment described by Dunstan and Hill.⁸ Cushman⁹ has shown that if a piece of commercial iron is left in contact with a solution of sodium chloride to which a little phenolphthalein has been added, a pink color develops in the solution which is in contact with some parts of the metal. These color zones, which are always confined to a portion (or portions) only of the metal, indicate the presence of hydroxyl ions (or free alkali) at these points. After being kept some time, the metal beneath the pink zones remains quite bright, but corrosion takes place gradually at the other parts of the metallic surface.

This experiment shows that the solution is being electrolyzed, and that the iron is passing into solution at those parts not marked out by the pink color of the phenolphthalein. It shows also that some parts of the iron are electrically different from other parts, and that the electromotive force produced by these differences of solution-pressure is enough, under these conditions, to cause the passage of an electric current of sufficient magnitude to produce perceptible electrolysis of the solution of sodium chloride.

It was shown by Dunstan and Hill, however, that the extent of this electrolysis was conditioned by the presence of oxygen, because, in the absence of air (or oxygen), the pink zones were not developed, and therefore an electric current sufficient to produce perceptible electrolysis could not pass.

Oxygen, therefore, must do one of two things—it must in some way or other increase the electrical differences between the parts of the metal and so increase the electromotive force,

*A paper read before the Faraday Society at the Manchester Meeting, Friday, April 4, 1913.

¹Trans. Amer. Chem. Soc., 1903, 25, 394.

²Trans. Amer. Electrochem. Soc., 1907, 12, 403.

³Trans. Chem. Soc., 1908, 93, 1356.

⁴Trans. Iron and Steel Inst., 1909, 1, 70.

⁵Science Progress, 1911, 22, 311.

⁶Trans. Chem. Soc., 1912, 101, 2068 et seq.

⁷*Loc. cit.*, pp. 2056 et seq.

⁸Trans. Chem. Soc., 1911, 99, 1853.

⁹Amer. Soc. Testing Materials, 1907.

or it must reduce the resistance of the circuit. When a piece of commercial iron is put into water in a vacuum, iron strives to pass into solution at the relatively electropositive part of the metallic surface, but hydrogen, produced by the electrolysis of the water, is deposited at the relatively electronegative part. This film of hydrogen, forming almost simultaneously and covering up the negative pole, introduces an enormous resistance into the circuit, and reduces the electric current to an almost negligible strength, so that the rate at which the iron passes into solution is infinitely small.

Oxygen dissolved in the water probably acts by deoxidizing the hydrogen deposited at the negative pole—destroying the polarization—and so allowing a greater current to pass between the electrically different parts of the metal.

If this argument is true, then commercial iron ought to pass into solution, in the absence of oxygen, if it is placed in an electrolyte, such as copper sulphate solution, where, instead of the non-conducting hydrogen film, there would be a conducting film of metallic copper produced at the relatively electronegative pole. Experiment shows this to be true. Commercial iron, when brought into contact with a solution of pure copper sulphate in a vacuum, causes the immediate deposition of copper on the iron just as readily as when the experiment is conducted in the presence of air; in a short time all the copper is removed from the solution and iron takes its place.

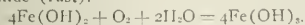
It seems to be clear then that when commercial iron rusts in contact with air and water, oxygen plays an important part in the actual dissolving of the iron, quite separate and distinct from its later function of converting the dissolved iron into rust.

The formation of rust from the dissolved iron is not, as it is sometimes supposed, a simple oxidation of the ferrous ions. It is not possible for the ferrous ion to be oxidized to the ferric ion by means of ordinary molecular oxygen. This can readily be proved by a simple experiment. If a solution of ferrous sulphate is made strongly acid with sulphuric acid and oxygen is passed through the solution, comparatively little ferric salt will be found in the solution even after the oxygen has been passed through it for several days. The greater the proportion of free sulphuric acid present, the less the oxidation which takes place. The oxygen only oxidizes the undissociated ferrous hydroxide formed by the hydrolysis of the ferrous salt, and since ferrous hydroxide is a very weak base, hydrolysis is very considerable unless a large excess of acid is present.

In the rusting of iron, then, the metal passes into solution as ferrous ions owing to electrolytic action between the electrically different parts of the iron, oxygen playing an essential part in the process. The ferrous ions then combine with the hydroxyl ions present in the water to form undissociated ferrous hydroxide:



At this stage oxygen plays a second part, converting the ferrous hydroxide into ferric hydroxide or some hydrated form of ferric oxide (rust).



The fact that iron rust produced under ordinary atmospheric conditions is shown, on analysis, to contain carbonic acid—that rust formed under such conditions is a basic carbonate of iron—does not necessarily prove, as has been supposed, that carbonic acid plays any essential part in its formation. Wet ferric hydroxide absorbs carbon dioxide, so that the carbonic acid found in rust formed under these circumstances may very probably have been absorbed from the air by the rust after its formation.

It follows from the theory that if we could produce metallic iron chemically and physically homogeneous it ought not to rust, because there would not be any differences of solution-pressure on its surface, and therefore no cause for the production of electric currents when the iron was placed in an electrolyte.

It would follow also from the theory that the production of a coherent and absolutely homogeneous coating, of whatever

nature, on the surface of commercial iron would prevent corrosion. The bearing of this on the question of passivity is discussed later.

The Properties of Chemically Pure Iron Considered in Connection with the Electrolytic Theory of Corrosion

The present writer¹⁰ has prepared chemically pure iron, and it has been found that such iron can be exposed to the action of air (or oxygen) and water (either distilled or tap-water) for an apparently indefinite time without showing any signs of corrosion. It must be strongly emphasized, however, that *chemical* purity is not the only essential.

In the preparation of pure iron by the writer's method the same sample of ferric nitrate, treated in exactly the same manner throughout its conversion into iron will not always give like specimens of the metal.

One batch of iron will rust quite readily, while another batch can be exposed for many months to the action of air and water without showing any signs of corrosion. All the pieces of the same batch behave, as a rule, in a precisely similar manner. Now any difference between the batches cannot be due to differences in *chemical* composition. The only possible variable factors are temperature of reduction and rate of cooling, and so differences in the product must be of a *physical* and not a *chemical* character.

It is a very striking fact that the pieces of iron which will not rust can also be put in solutions of copper sulphate or copper nitrate, of any strength, without causing copper to be deposited on the iron, while pieces from a batch of iron which rusts always cause the deposition of copper from the same solutions of copper salts. Sometimes the copper is deposited quickly, while at other times several hours may elapse before the deposition of copper takes place. The metal is always attacked at one or more points, and the deposition of copper spreads from these points over the whole surface in a very short time.

It is clear from these experiments that physical differences in iron of the same high state of chemical purity can cause most profound differences in its behavior.

If the theory is true, we should expect such results. In the case of the iron which does not rust and is unaffected by solutions of copper sulphate or copper nitrate, the metal is probably physically homogeneous, at any rate on the surface. There would, therefore, be no differences of solution-pressure (no electrically different parts) on the surface of the metal, and so no tendency for the metal to pass into solution by electrolytic action when the metal was put into an electrolyte. Since iron could not, therefore, pass into solution, we should not expect rusting to take place, nor should we expect the deposition of copper on the iron from solutions of copper salts.

The processes preceding corrosion and the deposition of copper from copper salt solutions upon metallic iron are undoubtedly similar in character.

In the case of the samples of pure iron which rust and which cause the deposition of copper from solutions of copper salts, the metal, although chemically pure, is not physically homogeneous, owing probably to some inequality in the temperature of reduction or the rate of cooling.

If such be the case, differences of solution-pressure would exist on the surface of the metal, and when it was put into an electrolyte there would be a tendency for electric currents to be set up between the electrically different points. In water, in the presence of air, this would lead to corrosion, in copper salt solutions to the deposition of copper. An interesting confirmation of such an argument is furnished by the following experiments.

Some pieces of the pure metal were taken which had been exposed to the action of water and air for several months without showing any signs of corrosion. They were carefully dried between pure Swedish filter paper, and then some of them were subjected to pressure in a polished agate mortar

¹⁰Trans. Chem. Soc., 1910, 97, 2430; 1912, 101, 2068.

by an agate pestle, while others were left untouched. Some of the pressed pieces were then put in distilled water and exposed to the air; they rusted in the course of a few hours, rust forming first at the edges which had not been pressed by actual contact with the agate, while the pressed portions remained quite bright. Others of the pressed pieces were put into a solution of copper sulphate; copper was deposited immediately on the metal, deposition first commencing at the unpressed edges of the pieces.

The pieces of iron which had been dried, but not pressed, neither rusted when placed in contact with water and air again, nor did they cause copper to be deposited from copper sulphate solution, so that it was the application of pressure alone which caused the profound alteration in the properties of the iron.

This again is readily explained by the theory. If a piece of iron be originally homogeneous and part of it be subjected to a strain, then the piece will consist of two parts, a strained and an unstrained part. It will contain two different modifications of iron, each having a different solution-pressure. If such a piece of iron be put in an electrolyte, there would now be a tendency for the passage of an electric current between the two electrically different parts, and so we should expect corrosion to take place when the iron was placed in contact with water and air. The fact that corrosion and the deposition begin at the unpressed part shows that this part is relatively electropositive to the pressed part.

These same points are illustrated in a striking and beautiful manner by the use of the ferroxyl reagent described by Cushman and Gardner.¹¹ The pure iron which does not rust is unaffected by the ferroxyl reagent, while the pure iron which is not physically homogeneous, or has been subjected to pressure, causes the development of the beautiful colors which indicate electrical differences, or differences of solution-pressure on the surface of the metal.

There are, however, some properties of pure iron which could not be predicted from the theory, but which cannot be considered as arguments against it.

The behavior of pure iron towards solutions of copper chloride is altogether different from its behavior towards solutions of the sulphate and the nitrate. Pure iron which has remained uncorroded after many months in contact with water and air, or which has remained unaltered in solutions of any strength of the sulphate or nitrate of copper, causes the immediate deposition of copper when placed in a solution of copper chloride, even when the solution is very dilute. The same effect is produced if sodium chloride is added to copper sulphate or nitrate solutions in which the iron has remained unaffected for long periods. The experiment is very striking if carried out in the absence of air. The iron quickly disappears, leaving behind finely divided copper, which in the course of a few hours is itself converted into white insoluble cuprous chloride.

Further, if a tube be taken containing pure iron which has remained unaltered in copper sulphate solution for a long period, and the temperature of the tube be raised to 100 deg. C., copper is deposited on the iron, and in a short time all the iron passes into solution, leaving behind small crystals of copper.

These facts led the writer to suspect that the pure iron might conceivably be covered with a thin coherent film of some protective substance, and most careful experiments were carried out to test this possibility. It was finally decided, as the result of these experiments (*loc. cit.*), that this curious behavior could not be accounted for by the presence of a protective film of either oxide or hydride on the iron.

The curious behavior of the pure iron towards copper chloride solutions is interesting if considered in connection with the behavior of the metal in sodium chloride solution. Even very dilute solutions of the chlorides of the alkali metals have a remarkable action in starting the corrosion of pure

iron which has withstood the action of water and air for long periods without showing any signs of corrosion. The action of salt solutions on pure iron is at present under investigation, and the results are not ready for publication.

It is well known that ordinary commercial iron is rendered passive by certain substances, *e.g.*, nitric acid, solutions of caustic alkalis or chromic acid, and that such passive iron, so long as the passivity persists, will not rust in contact with pure water and pure oxygen, and will not cause the deposition of copper from very dilute solutions of copper sulphate.

In view of what has been said above, it seems probable that these substances have the power, in some way or other, of altering the surface of commercial iron so as to destroy, temporarily, the electrical differences on its surface—to produce what might be described as an "electrically equable" surface. The result of this would be that the iron would possess no tendency to go into solution in an electrolyte and so would not rust when placed in contact with air and water.

Much the same sort of "electrically equable" surface is produced in a plate of impure zinc when it is amalgamated.

Since passivity can be produced by such widely different substances, it is not improbable that the "electrically equable" surface produced is not always of the same nature.

The writer is inclined to the view, which is as yet unsupported by experimental evidence, that substances which protect iron from corrosion have the power, in some way or other, of destroying or reducing the electrical differences which always exist in commercial forms of iron—that they have a passivating effect—and that the substances, like the chlorides of the alkali metals, which stimulate corrosion, do so by increasing these electrical differences. This would explain the remarkable behavior of pure iron towards copper chloride and alkali chloride solutions. For the present, however, the subject must be left open.

A New Sodium and Sodium Peroxide Factory

A new electrochemical company has been formed in Norway under the title of Frederikstad Elektrochemiske Fabriker, A. S. The capital is \$100,000. Mr. I. J. Moltke-Hansen, who was formerly in this country, and later works manager of the Norwegian Ashcroft sodium factory, is to be managing director and general manager. The works will be located at Frederikstad, Norway. For the present the manufacture of metallic sodium and sodium peroxide will be taken up while the intention is to add soon the manufacture of other compounds. The enterprise is stated to be backed by an important Norwegian financial group.

A New Testing Laboratory

The Abbe Engineering Company, of 220 Broadway, New York City, manufacturers of pulverizing and grinding machinery, vacuum pumps and pressure blowers, have opened an additional testing laboratory for the grinding, mixing, and crushing of samples submitted by prospective customers. The new laboratory occupies the entire ground floor of the building at Huron and West Streets, Brooklyn, N. Y. The laboratory is equipped with pebble mills of various sizes, "Max" mills, cutters, crushers, excelsior mills, vacuum pumps, and pressure blowers.

It is also intended to install an up-to-date bolting reel, on which the different meshes of Swiss silk bolting cloth sold by this company may be used for bolting samples.

The primary object of this laboratory is, of course, to obtain a more thorough and practical knowledge of the particular material to be tested for customers, so that recommendations for machines are based on experimental facts.

The Elmore vacuum process produced 1137 tons of copper concentrate at the plant of the Sulitjelva company, Norway, during the month of February, 1913.

¹¹Corrosion and Preservation of Iron and Steel, p. 48.

Some Effects of Temperature Upon the Resistance of Graphite and Carbon.

By Edwin F. Northrup

In connection with a test upon a graphite electric furnace, which the writer designed for use in his researches upon the resistivities of metals at high temperatures, observations were made upon the resistance of graphite beyond the temperature of melting nickel. The graphite was Acheson graphite of best quality and it was found that this graphite is remarkably constant in resistance up to a temperature of at least 1500 deg. C. The following table is self-explanatory:

Time P.M.	W	A _s	V _r	V _r A _s	Remarks
4.15	2.5	99.4	20.4	0.205	W = kilowatts supplied to primary of transformer.
4.20	2.5	104.8	19.64	0.189	A _s = amperes through furnace.
4.25	2.52	104.8	19.6	0.187	V _r = volts at terminals of graphite heater.
4.30	2.56	105.0	19.6	0.187	
4.40	2.6	105.0	20.0	0.190	
4.45	2.58	107.0	20.5	0.191	Opened switch for 4 minutes.
4.50	2.57	105.8	20.46	0.193	
5.01	2.68	106.4	21.2	0.199	
5.08	2.88	109.8	22.1	0.201	
5.10	2.92	110.2	22.3	0.202	
5.20	2.88	108.2	22.5	0.208	
5.30	2.67	103.2	21.8	0.211	
5.36	2.9	107.2	22.8	0.212	
5.38	2.89	107.0	22.76	0.212	Nick starts to melt.
5.40	2.89	107.0	22.76	0.212	Nickel all melted, 1452° C.
5.50	2.8	105.0	22.38	0.213	
6.00	3.0	110.0	22.96	0.209	"Lavite" not yet melted ("Lavite" melts at 1523° C.).
6.01	3.0	109.8	22.8	0.208	"Lavite" not melted. Opened switch.

The writer, being aware that ungraphitized carbon (as an electric light carbon) has a marked negative temperature coefficient, devised the following experiment which exhibits the difference between this material and graphite in a striking manner: A rod of graphite and a rod of carbon were selected, each $1\frac{1}{4}$ in. long and $\frac{1}{4}$ in. in diameter. These were used to connect together two plates of graphite. Two holes were drilled in each plate of graphite into which were inserted, for a distance of $\frac{1}{4}$ in., the ends of the carbon and the graphite rods. The graphite plates were connected to the low-tension side of a 2-kw. transformer which yielded, with 110 volts on its primary, 22 volts on open circuit on its low-tension side. A rheostat was put in the primary side of the transformer to limit to an approximately constant value the current delivered by the secondary. On closing the circuit the graphite rod came quickly to incandescence while the carbon rod, in parallel connection with it, remained for a time dark, because of its higher resistance at a moderate temperature. After a fraction of a minute the carbon rod began to heat, and in doing so its resistance rapidly lowered so that it took more current than the graphite rod, and it in turn became brilliantly incandescent, while the graphite rod greatly declined in brilliancy.

The writer concludes that, for small laboratory furnaces at least, graphite makes a better resistor material than ungraphitized carbon. When a constant potential was applied to the terminals of an electric furnace made of Acheson graphite the watt input registered on a wattmeter remained remarkably constant, without any adjustment of rheostats, over a temperature range as great as 1400 deg. C. The regulation of such a furnace is extremely easy.

Palmer Physical Laboratory,
Princeton, N. J.

In sampling ore automatically, time samplers which receive a falling stream of ore for a definite period of time, are regarded as the best devices for the purpose. They operate on the principle of taking the whole stream for a portion of the time, in distinction to that class which attempts to divide the stream of ore and cut part of it all the time. The lack of uniformity in the falling stream renders the latter type of machine inaccurate.

The Uranium and Radium Situation.*

By Dr. Charles L. Parsons.

Some months since rumors reached the U. S. Bureau of Mines of an increased demand for carnotite ores from Colorado and that these ores were being shipped abroad in some quantity. Further, it was reported that the methods of production involved large losses of material and that methods for concentrating low-grade material now being thrown on the dump were greatly needed. Accordingly, Messrs. R. B. Moore and K. L. Kithil were assigned to the task of investigating the situation with headquarters at Denver, where the Bureau established a laboratory for the purpose of investigating the rarer metals occurring in the western part of the United States and problems bearing upon the prevention of waste and increased efficiency in the mining industry. The surprising conclusion has been reached that while all the radium placed upon the market in the last few years has been produced in Europe, a large portion of this output has come from American ores.

Radium institutes have been established in Austria, France, Germany and England, a European science and industry have been developed from American radium ores and even the uranium present with the radium has been manufactured into marketable condition only in foreign countries and returned in finished condition to our own. American hospitals and physicians have been forced to procure from abroad such radium as they could afford for experimental purposes, and investigations in our governmental and university laboratories of the wonderful properties of radium and their possible application to the eradication of disease and the development of industry have been hampered by the almost prohibitive prices at which the finished material is held.

While the Austrian government, realizing the untold possibilities of the radium ores of St. Joachimsthal, has purchased the mines, put their output under direct governmental supervision, and has entered into an arrangement whereby this ore is worked up in cooperation with the Vienna Academy of Sciences for experimental purposes in a carefully administered radium institute, America has allowed her large and much greater resources to be exploited on a basis which wastes perhaps irretrievably a large portion of the material mined, and has exported carefully selected ores at a price by no means commensurate with its radium value if worked up at home.

Even before carnotite was exported, pitchblende of the highest grade was sent out of the country at the time when the world's radium output was supposed to be coming from Austrian ores. At least 20 to 25 tons of high-grade pitchblende has been sent out of the country. Within the last two years, however, foreigners have realized the value of our carnotite resources and most of the radium that has been exported has gone abroad in this ore.

During the last year, carnotite carrying 28.8 tons of U₃O₈, from which 8.8 grams of radium chloride or 11.43 grams of radium bromide could be obtained, were produced. Practically all of this ore was shipped abroad for the extraction of radium. The value of the radium salts extracted would be at the minimum market price \$528,000. The total supply of radium salts from all other sources including the Austrian mines was probably not more than 3.65 grams of radium chloride, basing the production of the Austrian mines for 1912 upon that of 1911 which is known.

Pitchblende, the richest of all uranium minerals, is composed mainly of uranium oxide, but also carries lesser quantities of a large number of other substances. It has been found in small quantities in Connecticut and in the feldspar quarries of North Carolina. Practically the total American output has come from the mines in Quartz Hill, Gilpin County, Colorado. The mineral is a heavy black substance which can be readily identified by any one who will suspend a sample of the pitchblende above a photographic plate wrapped in black paper and kept in the dark for a few days with a key or other metal opaque to radium radiations placed between the sample of ore and the

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plate so that when the plate is developed a shadowgraph of the object may identify the ore. Pitchblende may carry as high as 80 per cent. uranium oxide, although the average ore is not nearly as rich.

Carnotite is a yellow mineral consisting mainly of potassium uranyl vanadate but containing also small amounts of barium and calcium compounds. Being a uranium mineral as is pitchblende it of necessity carries radium, although it has not yet been definitely established that the uranium and radium are in equilibrium as they are in pitchblende. However, it is known that in our western carnotite the amount of radium is not far from the equilibrium ratio and in calculations given above an allowance of 10 per cent has been made to cover this possible deficiency. While carnotite is known to occur in smaller quantities in other states, the more important deposits are scattered over a considerable area in Colorado and Utah, embracing Meeker and Skull Creek, Colo., Green River, Thompson's Moab, Richardson, Table Mountain, Pahreah, and other places in Utah. The largest proportion of the ore, however, has been produced in or around Paradox Valley in southwest Colorado from which it has to stand long hauls by pack animal or wagon to the railroad. Carnotite always carries vanadium as well as uranium and radium, but is purchased almost wholly on its radium content, comparatively little being allowed for the vanadium present.

The ore consisting of a fine grained sandstone containing yellow finely pulverulent carnotite occurs in pockets and is easily mined. As ore below 2 per cent uranium oxide cannot at the present time find a market, a considerable portion of the ore has been thrown on the dump and is now being wasted as material of lower grade has to be discarded on account of the long haul and the fact that European buyers have set this standard as to quality. Ores of higher grade are sometimes obtained, but they occur only in small pockets and it is generally advisable to mix these high-grade ores with ores of somewhat lower grade in order to increase the marketable output. Ore of 2 per cent uranium oxide is now worth approximately \$75 per ton f.o.b. New York.

In the mining of these carnotite ores it is probable that 5 tons of material capable of concentration are thrown upon the dump for every ton that finds its way to market. To develop methods for concentration of these ores and save the valuable material now wasted is one of the problems before the Bureau of Mines with fair prospect of a successful conclusion.

It is difficult to estimate the total amount of radium that has been produced up to the present time, but it is quite certain that if the ores which have been mined in this country and abroad and sold for radium production have been actually worked up into this material there is now in existence something like 40 grams ($1\frac{1}{4}$ ounces) of radium. The price of radium salts varies somewhat. In large quantities it has been \$60,000 per gram for both radium chloride and radium bromide, although the latter contains less metallic radium in proportion to its weight than the former. It should be remembered, therefore, that it is more advantageous to purchase radium chloride than radium bromide. In small quantities the average price has been \$80,000 per gram which represents about \$2,500,000 an ounce.

The figures given show very plainly that the United States has taken the palm from Austria as the radium-producing country of the world. Very few people have been cognizant of the fact that the United States has such deposits within her borders. Up to the present time very little interest has been taken in the matter and only one firm has engaged in the extraction and refining of radium in this country—a condition which is deplorable. This firm has not yet entered the radium market.

Practically every ton of ore mined in 1912 went abroad, and as the American deposits are far from being inexhaustible we are rapidly depleting our own reserve and are shipping from the country material of great value and of unknown possibilities which cannot be replaced.

The applications of radium are still too little understood to admit of definite statement. Its discovery and marvelous

properties have already changed our ideas regarding the constitution of matter and scientific investigation will undoubtedly lead to valuable results which we cannot now even foresee. Altogether too many incorrect statements and vague speculations have been placed before the public as to its use in medicine. A recent report of the London Radium Institute and the many articles emanating from minor laboratories experimenting in the application of radium to therapeutics all tend to show, however, that it has a real value, the certain application of which must await further experimentation. In the meantime no credence should be given to the many stories that are sure to be printed unless they are backed up by the highest medical authority, which will always give publicity with caution.

The best medical authorities appear to agree that up to the present time radium has not been proved to be specific for any disease, although it has been shown to be helpful in many cases and the outlook for its future application to certain diseases not easily treated otherwise are decidedly encouraging.

Apparently no uranium is worked up in the United States, but according to statistics gathered by the division of mineral resources of the United States Geological Survey about \$14,000 worth of its oxides and salts were imported into the United States in 1911. It is one of the few materials shipped abroad as ore and returned in manufactured form.

A preliminary report on Uranium, Radium and Vanadium, by R. B. Moore and K. L. Kithil, will soon be issued by the Bureau of Mines. This bulletin describes the carnotite deposits of Colorado and Utah and the pitchblende deposits of the former state. It also contains detail of which the foregoing is simply a general summary which cannot fail to be of value to all those interested in our mineral resources and their development.

*Division of Mineral Technology,
U. S. Bureau of Mines.*

The mineral production of Canada in 1912, according to preliminary estimates by the Canadian Department of Mines, shows an increase of nearly 29 per cent as compared with that of 1911. Porcupine was the only new contributing district of any importance, with a gold production of \$1,750,000. There was an increased output in all the provinces during the year. Copper and asbestos contributed to the increase in Quebec; nickel, copper and gold in Ontario; coal in Alberta; gold, silver, copper, lead, zinc and coal in British Columbia. The production of cement is interesting: the Canadian output increased over 26 per cent; the imports, 116 per cent, and total consumption, 34 per cent.

Paraffin Paper for Spot-Plates.—The Mine & Smelter Supply Company, Denver, has recently devised a substitute for the ordinary porcelain spot-plate for laboratory use. It consists of a pad of paraffined sheets of white paper, from which a sheet may be torn when it can no longer be used. The paraffined surface causes the spots to assume a spherical form, instead of spreading and running together as is likely to be the case with porcelain surfaces.

Potash Lands Withdrawn.—Three tracts of land in the desert basins of California and Nevada have been withdrawn from entry, as they are believed to contain valuable deposits of potassium salts and brines. The national government is at the same time making provisions to relieve the present chaotic condition regarding such lands, and will endeavor to provide a safe legal basis for the development of the deposits now known and any that may be developed in the future.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in January, 1913, was sixty-one. The quantity of ore milled during that period was 2,347,029 tons. There were 10,000 stamps in operation with an average duty of 8.66 tons per twenty-four hours. Tube mills in commission numbered 286. The yield for the month was 789,310 fine ounces gold. The yield per ton of ore milled was 6.56 dwts.

Atlantic City Meeting of the American Electrochemical Society

The twenty-third general meeting of the American Electrochemical Society was held in Atlantic City from April 3 to 5 and was a most enjoyable affair. There was a good attendance, the registry list containing over 150 names (all coming from other cities). There was ideal crisp spring weather, just right to enjoy Atlantic City at its best. There was a series of very interesting papers on various electrochemical subjects, and a most instructive symposium on electrodeposition of metals. There were lively discussions in open meeting and quietly between man and man. And above all there reigned supreme the old and ever young happy congeniality which is typical of every Electrochemical Society meeting.

Thursday, April 3, was devoted to the annual business meeting and the reading and discussion of papers on various electrochemical subjects. On Friday an excursion was made to Phila-

No current is supplied to the bath through the pole plates, which have been disconnected in order to improve the cooling action by removal of the winding which fed the pole plates when originally installed.

The party then went to the University of Pennsylvania, where the Society was entertained at luncheon. The provost of the University, Dr. Edgar F. Smith, heartily welcomed the Society to the University in a felicitous speech. For the Society the president-elect, Dr. E. F. Roeber, replied, emphasizing the debt of gratitude which electroanalysis and electrochemistry in America owes to Dr. Smith and reminding the audience of the fact that it had been in Dr. Smith's chemical laboratory that the original meeting of the Society had been held eleven years ago.

In the afternoon two different parties were formed. One



PHOTOGRAPH TAKEN AT THE SATURDAY SESSION OF THE ATLANTIC CITY MEETING OF THE AMERICAN ELECTROCHEMICAL SOCIETY

delphia, while a most interesting experimental lecture was delivered in the evening by Professor Kenrick. The two sessions of Saturday were devoted to the presidential address of Prof. W. Lash Miller and to the symposium of papers on electrodeposition.

Excursion to Philadelphia

Early on Friday morning the members of the Society were taken by a special car to Philadelphia and from there to Lansdowne, Pa., where the works of the Crucible Steel Casting Company were visited. Besides the oil fired crucible furnaces the feature of principal attraction was the two-ton Roechling-Rodenhauser induction steel furnace (the only one of this type in the United States). This was shown in operation and explained by Mr. Cunningham, Mr. von Baur and Herr Schoenawa. It is, however, not run now as a Roechling-Rodenhauser combination furnace, but as a pure induction furnace.

visited the works of Harrison Bros. Company, on Gray's Ferry Road, where lead paints, lithophone, and contact sulphuric acid are manufactured, while the other party was taken by automobiles to the United Gas Improvement Company's works at Point Breeze. Afterward all met again at Broad Street Station and returned to Atlantic City.

The hotel headquarters were at the Hotel Traymore, where all professional sessions were held in the solarium. No official dinner or smoker had been arranged for the meeting. But section Q held several informal sessions in the "icebox" and elsewhere.

Business Meeting

The annual business meeting of the Society was called to order in the morning of Thursday, April 4, by the President, Dr. W. Lash Miller, of the University of Toronto. From the report of the Secretary it was learned that the Society is in

a flourishing condition, the total of the investment funds being more than \$6,000 and the membership on December 31, 1912, being 1350.

The result of the latter ballot for election of officers for the next year was then announced.

Dr. E. F. Roerber is the new president.

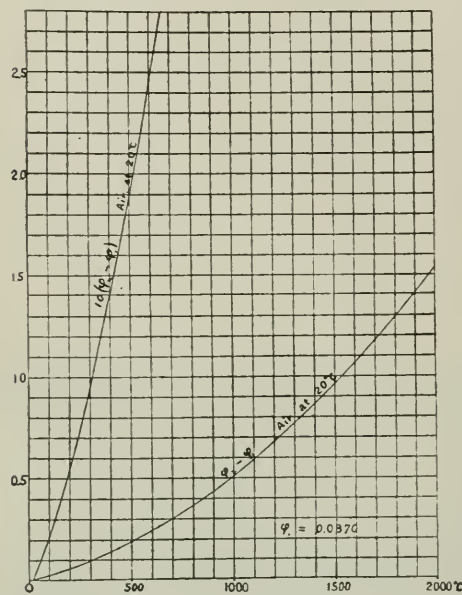


FIG. 1.—CURVE OF ϕ AS FUNCTION OF TEMPERATURE

Dr. Joseph W. Richards and Mr. Pedro G. Salom were re-elected secretary and treasurer respectively.

Prof. C. F. Burgess, Prof. S. A. Tucker, and Mr. C. P. Townsend were elected vice-presidents, and Mr. F. A. J. Fitzgerald, Dr. J. W. Brown, and Dr. C. G. Schlunderberg managers.

Convection and Radiation of Heat

Dr. Irving Langmuir, of the Research Laboratory of the General Electric Company, of Schenectady, N. Y., presented two papers on the convection and radiation of heat. The chief results of the first paper are as follows:

The total energy loss from a heated surface in a gas consists of two parts, convection and radiation.

If the emissivity of the surface is known the radiation can be calculated from the Stefan-Boltzman law.

$$W_r = 5.9 \times 10^{-12} e (T_s^4 - T_f^4) \text{ watts per sq. cm.} \quad (3)$$

e is the emissivity of the surface, and varies for different surfaces from 0 to 1. For any one surface it is accurate enough for most purposes to consider it independent of the temperature.

Free convection from a surface consists essentially in conduction through a film of gas of definite thickness, and can be calculated by the ordinary laws of heat conduction when the film thickness is known.

The loss by convection is equal to the product of two factors, thus:

$$W_c = S (\phi_2 - \phi_1)$$

S is the shape factor, and depends only on the shape of the body and on a quantity B , a constant for any given gas under given conditions of temperature and pressure. B is equal to the thickness of the film of relatively stationary gas through which conduction takes place in the case of convection from a plane surface $\phi_2 - \phi_1$ is equal to $\int_{T_1}^{T_2} k dT$ where k is the heat conductivity of the gas. The heat conductivity can be readily cal-

culated from the specific heat and viscosity of the gas. Tables of ϕ for air, nitrogen, hydrogen, carbon dioxide and mercury vapor are given. A curve of ϕ as a function of the temperature is given for air in Fig. 1.

The shape factor S has been calculated by the author of the case of convection from planes, cylinders and spheres.

In this way the following formulas for convection are derived:

For Plane Surfaces:

$$W_c = \frac{\phi_2 - \phi_1}{B} \text{ watts per sq. cm.}$$

For Horizontal Cylinders of Diameter a :

$$W_c = S (\phi_2 - \phi_1) \text{ watts per cm.}$$

where S is found from a certain equation given in the paper. But it may be more easily determined as follows:

In Table I S as a function of a/B is given. If a curve is prepared from this data the value of S may be found in any particular case with great ease.

TABLE I.—SHAPE FACTOR FOR WIRES

S	a/B	S	a/B	S	a/B	S	a/B
0.0	0.0	5.0	0.453	10	1.696	30	7.738
0.5	0.56×10^{-6}	5.5	0.558	12	2.263	32	8.370
1.0	0.594×10^{-6}	6.0	0.671	14	2.844	34	8.995
1.5	0.725×10^{-6}	6.5	0.783	16	3.438	36	9.622
2.0	0.752×10^{-6}	7.0	0.908	18	4.040	38	10.25
2.5	0.0644	7.5	1.032	20	4.645	40	10.87
3.0	0.1176	8.0	1.160	22	5.263	42	11.50
3.5	0.185	8.5	1.291	24	5.877	44	12.14
4.0	0.265	9.0	1.424	26	6.505	46	12.77
4.5	0.354	9.5	1.561	28	7.122	48	13.40
5.0	0.453	10.0	1.696	30	7.738	50	14.03

For Spheres of Diameter a :

$$W = S (\phi_2 - \phi_1) \text{ watts}$$

where S is found from the equation:

$$\frac{1}{a} - \sqrt{\frac{\pi}{SB}} = \frac{2}{S} \pi$$

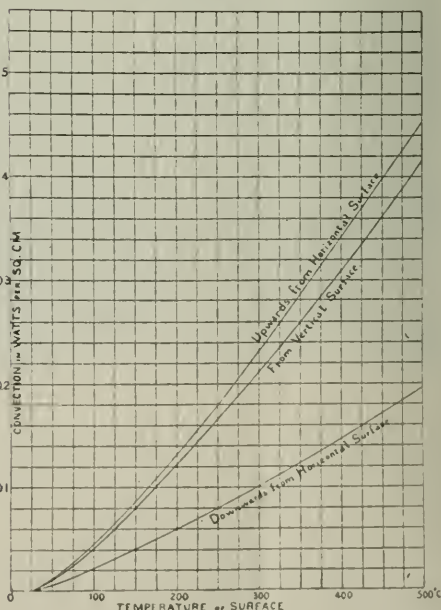


FIG. 2.—RADIATION FROM HORIZONTAL AND VERTICAL SURFACES

In each of these cases the convection can be calculated if we know $\phi_2 - \phi_1$ and B the film thickness for a plane surface.

For air at atmospheric pressure and room temperature, experiments on wires of various sizes and on plan surfaces

give B independent of the temperature of the heated body. The experiments on wires gave $B = 0.43$ cm. and the experiments on vertical plane disks gave

$$B = 0.45 \text{ cm.}$$

From horizontal plane surfaces the convection from the upper surface is about 10 per cent. greater than from a vertical surface, but from the lower surface about 50 per cent. less than from the vertical surface (Fig. 2).

The emissivity of various metallic surfaces was found to be:

Black Body	100
Copper, oxidized	72
Copper, calorized	26
Silver	3
Cast iron, bright	22
Cast iron, oxidized	62
Aluminum paint	50
Gold enamel	37
Monel metal, bright	43
Monel metal, oxidized	43

The total surface losses from these metals at various temperatures up to 600° C. are given in a table in the paper. In

The second paper by Dr. Langmuir deals with the calculation of the shape factor of solid bodies and gave especially a practical formula for rectangular furnaces.

In the discussion Dr. W. L. Miller referred to the analogy between heat diffusion and diffusion in liquids.

Dr. Ilering, in discussing Dr. Langmuir's film on the surface of a body giving off heat, referred to the reverse phenomenon, namely, the importance of the film on the surface of a body which receives heat, like a boiler tube. A single postage stamp pasted on the outside of a boiler tube is not charred. But if enough stamps are pasted one above the other, the last one will be charred. As to the enormous difference between the emissivities of different substances, it might look ridiculous at first glance to silverplate the outside of an electric furnace in order to reduce its radiation loss, but it was certainly worth while to take the emissivity of the furnace walls into consideration, as it would permit the use of thinner walls.

Dr. Richards also emphasized the importance of keeping the outside surface of a furnace clean and polished. If the radiation from the outside surface is reduced, its temperature rises and the loss of heat by conduction through the walls is also reduced on account of the reduced temperature difference inside and outside.

Experiments with Furnace Electrodes

A paper on this subject was presented by Mr. F. A. J. Fitz Gerald, of the Fitz Gerald & Bennie Laboratories, Niagara Falls, N. Y., and Mr. A. T. Hinckley, of the National Carbon Company, Cleveland, Ohio. Some years ago the National Carbon Company commenced an extensive series of experiments for the purpose of finding the best methods of making large carbons for the larger sizes of electric furnaces coming into use so as to obtain a thoroughly trustworthy electrode that would not fail under the severest usage and at the same time give the smallest wear under the particular conditions which surrounded its use. This study also involved the problem of joining electrodes together so as to avoid the loss in "butts."

The authors describe in their paper the methods used in testing the experimental electrodes and some of the results recently obtained.

The real density, that is, the density of the material free from pores, may be considered as a measure of the chemical composition of the material, and increases as the amount of hydrogen in the hydrocarbons decreases. It is thus an indication of the relative rate of oxidation of the electrode, it being a general rule that hydrocarbons high in hydrogen are most readily oxidized.

The determination is made by grinding the sample to such a fineness that all pores are open to the liquid medium. For ordinary carbon electrodes particles passing through a screen having 100 meshes to the linear inch are sufficiently fine. Carbon ground to pass through a 200-mesh screen does not show any increase in the second decimal place in the density determination. The weighed sample is placed in a small specific-gravity bottle with kerosene oil of known density, and evacuated to 5 mm. of mercury so that the pores are filled with the oil. The bottle is then filled with the oil, brought to a given temperature in a thermostat and weighed. The precision of the measurement is 0.5 per cent.

The apparent density, that is, the density of the carbon including the pores, may be considered as a characteristic of the mechanical structure. It is influenced by every process through which the carbon goes in the manufacture of the electrode, and is in general a measure of the relative rate of disintegration of the electrode in use. For large electrodes the most satisfactory method of determination is to weigh the whole electrode and calculate its volume from its dimensions. To secure a precision of 2 per cent the weight and length should be measured to 1 per cent and the diameter or periphery to 0.5 per cent.

The electrical resistivity of carbon electrodes varies with both the real and apparent densities. Its determination is not

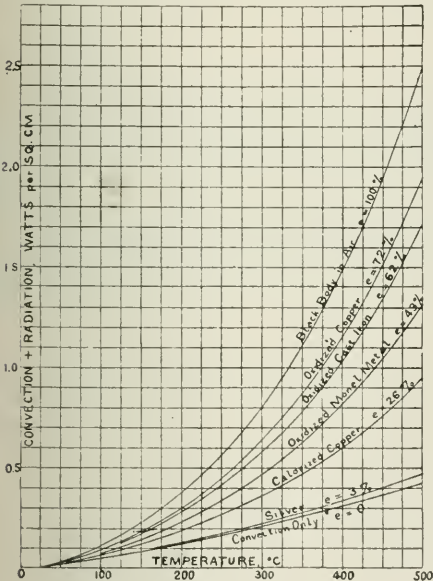


FIG. 3.—TOTAL LOSSES FROM HEATED SURFACES IN AIR

the curves of Fig. 3 are given plots of the total energy losses calculated by adding the radiation (from Stefan-Boltzman law) and convection (from $(\phi_2 - \phi_1)/B$).

Dulong and Petit's law and Lorenz's law for the relation between convection and temperature of the hot body hold very satisfactorily up to very high temperatures. At low temperatures these equations probably give the convection even more accurately than those calculated from the film theory, but at high temperatures the film theory gives much better results. Furthermore, the newer theory enables the effect of the shape and size of the body to be taken into account.

For forced convection, that is, loss of heat in strong air currents, the film theory does not seem to apply. On the other hand, an equation derived by A. Russell seems to agree well with the experimental data available.

For other gases than air, or for air at other temperatures or pressures, the value of B can be assumed to be inversely proportional to the density of the gas. This will give results usually accurate enough for the calculation of convection from small wires.

necessary so much to show the physical properties of the carbon as to provide a means of comparison in electrical terms. With large electrodes its determination is made most readily by passing direct current through the whole carbon and measuring the potential drop through a known distance. For ordinary purposes it is sufficient to measure the resistivity to 0.0001 ohm, a precision of about 5 per cent. This means, approximately, that the volts, amperes and length must be measured with a precision of 1 per cent and the diameter or periphery with a precision of 0.5 per cent. If these rules are followed the magnitude of the testing current is of no importance, provided that it is not sufficiently great to heat the carbon appreciably. Care must be taken in selecting the points for the potential contacts that the distance between them and the current contacts is so great that the equipotential surfaces are substantially plane surfaces at right angles to the axis of the electrode. The resistance of the potential contacts may be practically eliminated by drilling small mercury cups in the carbon or pressing the contact pins against the carbon at a pressure of about 3000 lb. per square inch (210 kg per square centimeter) when using a millivoltmeter having a resistance of 1.07 ohms.

Service Tests.—Useful as the laboratory tests are on electrodes, they do not give sufficient information as to what may happen when the electrode is put in actual service. A serious objection to some of the earlier carbon electrodes used

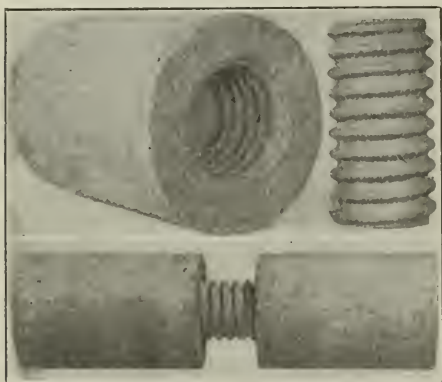


FIG. 4.—JOINING CARBON ELECTRODES

in electric furnaces was the large percentage which broke in service.

Another important consideration in furnace electrodes was the life, some electrodes giving apparently much longer service than others. It is not implied that the astonishing variation in the figures for electrode consumption that is found in technical literature is due altogether to differences in the electrodes used, for this is in large part explained by the irrational method of estimating electrode consumption. For example, however useful from a purely commercial point of view the method of estimating electrode consumption in pounds per ton of steel may be, the important consideration when comparing electrodes is to determine how many kilowatt hours are generated in the furnace per unit weight of electrode. Studied from this point of view it is found that great variations exist.

In experimenting with the manufacture of carbon electrodes it is, therefore, necessary actually to use them in an electric furnace and thus test their behavior under working conditions.

Finally, in working on the problem of making electrodes which can be joined together, laboratory tests of the electrical resistance of the joint, or even tests of its mechanical strength, are not sufficient. It is necessary to use the jointed electrode in a furnace and determine just how it behaves throughout a series of runs.

The authors describe the tests to which the electrode was submitted in an open-arc furnace of the Siemens type, where it worked above a molten bath.

In making a test the furnace was charged with cast iron and steel borings, and the electrode lowered until an arc was started. Then, if it was simply desired to find out if the electrode would stand severe usage without breaking, the power, which should normally have been 150 kw, was increased to 200 kw or 250 kw. In general it is advisable to use an electrode, in a furnace of this sort, with a current density of not more than 40 amp per square inch (6.2 amp per square centimeter), but occasionally higher current densities are necessary or desirable, so that the electrode must be constructed to stand these. Thus, in the experiments to determine the resistance under severe conditions, current densities of 50 amp to 65 amp per square inch (7.8 amp to 10.1 amp per square centimeter) were used. Moreover, for short periods the current density was as high as 77 amp per square inch (11.9 amp per square centimeter).

In making tests of carbon consumption it was soon found that in order to get consistent results the furnace must always be run under identically similar conditions. Before beginning a series of heats the electrode was carefully weighed. Then the furnace was charged with the cast-iron and steel borings, these melted down and the furnace tapped. After a certain number of heats, usually eight, were run off the electrode was again weighed, and the total kilowatt hours used was divided by the loss in weight of the electrode, which gave the kilowatt hours per pound of electrode consumption.

As a result of numerous experiments it has been found that the joint best suited to the average furnace needs is made by using a cylindrical threaded plug, which is screwed equal distances into the ends of the two electrodes to be joined. A shallow rounded thread proves better than a pointed one, or even a truncated one, as there is less tendency to breakage during assembling. A specially prepared paste is used to insure good electrical contact, but this does not interfere in any way with unscrewing the joint when it is desired to replace a section. Fig. 4 shows this method of joining electrodes.

In order to determine the resistance of the jointed electrode to severe usage the method followed was just the same as in the case of the ordinary electrode as regards overloading; but in addition to this a long series of heats were made so that the behavior of the joint as it finally worked down to the bath could be noted.

A very important consideration in the case of jointed electrodes is the resistance of the joint. The method of measuring the resistance of the joint is described in detail.

A large number of electrodes which had gone through different manufacturing processes were tested in the furnace simply for the purpose of determining whether or not they would break in use when subjected to the severest conditions, such as overloading, sudden changes of temperature, etc., and as a result much knowledge has been gained as to the causes of electrode breakage.

In order to study the influence of different manufacturing processes on the consumption of carbon under definite conditions of use a large number of electrodes were manufactured, these being divided into several series. Only one of the numerous factors entering into the manufacturing process was varied in any series, every other condition being kept the same. When the electrodes were made they were tested by the regular laboratory method, and then submitted to the service tests described above. In this particular investigation there have been six series of carbons tested.

As regards the real density no variations have been found, all electrodes giving the same results. As the attempt was made so to adjust conditions of manufacture that there should be no variation in this physical characteristic the results may be considered as highly satisfactory, and give reasons for believing that the great variations found in other characteristics were due simply to the deliberate differences in manufacturing process and not to some unknown causes.

Considerable differences were found in the apparent densities of the electrodes tested, these varying from 1.40 to 1.50. The most interesting results, however, were those obtained in determining the kilowatt hours used per unit weight of carbon, for it was found that the extreme variations were as much as 100 per cent. The importance of this discovery can scarcely be overestimated as showing the great influence manufacturing conditions may have on electrode economy. Such astonishingly large variations were at first viewed with suspicion, for it was feared that they were really due to the way in which the electrodes were used in the furnace and not to inherent peculiarities of the carbon. Experiment showed, however, that when tests were repeated the results checked in a satisfactory way.

In testing the jointed electrodes a great number of heats were made so that the behavior of the joint itself when in the furnace could be noted. Fig. 5 is from a photograph showing a partially consumed joint. In one set of tests an electrode was made by joining four 15-in. (380 mm) sections together, thus making an electrode 60 in. (1520 mm) long and 10 in. (254 mm) in diameter. The electrical connection for the cables was made at the top section, and new sections added to the electrode as it wore away. With this electrode sixty heats were made, the behavior of the electrode throughout being excellent.

From several tests, details of which are given in the paper,



FIG. 5.—CONSUMPTION OF ELECTRODE

it is seen that the resistance of the joint decreases as the electrode gets hotter. It appears that the heating of the joint improves the contact between the sections and the connecting plug.

In the discussion which followed Mr. Petinot referred to the difficulties of joining electrodes in actual furnace practice. At a joint (as they were formerly made) it sometimes happened that all the current passed through the nipple which thus became very hot, while the outside of the electrode at the joint was cold. It may be advisable to make the nipple of graphite, but if satisfactory electrodes 10 ft., 15 ft. or 20 ft. long could be made, they would be better than any method of joining.

Dr. Richards advised to use as long electrodes as possible and make the connection as close to the furnace as possible.

Mr. Hansen agreed that long electrodes are very nice with stationary furnaces, but they may become a nuisance with tilting furnaces. As to the electrode consumption, he pointed out the importance of oxidation by the slag.

In reply to a question by Dr. Whitney as to electrodes with mineral materials, Mr. Hinckley replied that certain mineral materials increase the life of electrodes. The paste used in the joint is a graphite mixture.

Mr. Lidbury said that the most important point in connection with electrodes is that they should not break under "reasonable treatment." But as Mr. Fitzgerald said, there is a difference of opinion between manufacturers and users of electrodes as to what is reasonable treatment.

Aluminium Nitride.

A paper on aluminum nitride and the Serpek process, by Dr. Jos. W. Richards, of Lehigh University, was the first paper of Thursday afternoon. It was essentially the same paper as presented by the author before the New York section

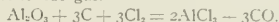
of the American Electrochemical Society and published in full in our March issue, page 137.

New, however, were the following notes on the Serpek reaction.

The decomposition of alumina by carbon in presence of nitrogen to form aluminium nitride is a reaction which is interesting to study and compare with analogous reactions.

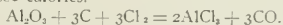
Professor Baron, in 1760, tried to reduce alumina by carbon, but was unable to obtain the temperature required.

Oerstedt, in 1817, devised a way to decompose alumina by carbon, finding that this took place in a stream of chlorine gas, at a low temperature, the products being aluminium chloride and carbon monoxide gas.



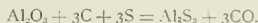
This reaction is practically the exact analogue of the Serpek reaction, using chlorine instead of nitrogen. Deville, for instance, who ran this operation on a large scale, remarks that the retort is heated to a uniform dark-red heat (which would be about 600 deg. C.) and that "however fast the chlorine may pass into the retort, it is so well absorbed during three-fourths of the operation that not a trace is mixed with the carbon monoxide escaping."

If the query be made as to why the reaction is so easy, thermochemistry gives us the answer: the reaction is exothermic 17,880 calories.

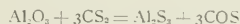


$$-391,600 \quad + 322,000 + 87,480 = +17,880$$

It is profitable to consider in this connection another analogous reaction.

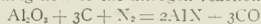


This reaction does not take place at 1200 deg. C., and we find it would be endothermic (heat absorbing) 180,200 calories. However, the reaction with carbon and sulphur already combined as CS_2 does take place, readily, at 1000 deg. C., and this reaction is much less endothermic than with C and S separately:



$$-391,600 + 87,000 + 124,400 + 111,090 = -69,110$$

Coming now to the nitrogen reaction, we have:



$$-391,600 \quad - 90,900^* + 87,480 = -213,220.$$

The figure 90,900 was given as provisional. It is not a measured value.

The above is therefore a highly endothermic reaction and corresponding with that we have the fact that it does not take place to an appreciable extent below 1500 deg. C. Aside from all thermal losses, we can say, therefore, that the reaction itself absorbs 213,220 calories = 248 kw-hours for 82 kg. of aluminium nitride formed; which equals 3000 kw-hours = about $\frac{1}{3}$ kw-year per ton of nitride. At least double this amount would be required in practice.

Tests of the Edison Storage Battery

"Some Tests of the Edison Storage Battery" was the title of a paper presented by Messrs. C. W. Bennett and H. N. Gilbert, of Cornell University. The object was to run a few efficiency tests at different rates of discharge, study the effects of charging at different rates, check the measurements of capacity, and study the cell operating at a low temperature.

The following conclusions are drawn by the authors

1. The energy efficiency, at the normal rate of operation, is about 55 per cent, the amp-hour efficiency being about 85 per cent.

2. The cell is capable of operating continuously at high rates.

3. At the 40-minute discharge rate, with charge at the normal rate, the energy efficiency is about 32 per cent, the amp-hour output being about 60 per cent of the rated output.

4. The cell can be charged at high rates, the efficiency being about normal when a "booster" charge at $\frac{3}{4}$ times the normal rate, for one hour, was given the cell. There is no reason why the cell should not be charged at five times the normal rate for a few minutes.

5. If the cell is to be charged the day before it is to be used, energy is conserved if the charge is maintained until the voltage reaches about 7 volts, instead of carrying it to completion. In other words, no nickel peroxide should be formed on charging, the loss of energy on standing being due presumably to the spontaneous decomposition of this compound.

6. If the charge is stopped before the higher oxidation products of nickel are formed, the efficiency is higher than normal being about 64 per cent.

7. At 4 deg. C. the battery is capable of delivering more than two-thirds the normal output, when charged at the same temperature. The voltage, of course, is lower than normal, for the internal resistance of the cell is higher at the lower temperatures. The efficiency also is low.

8. The cell may unquestionably be allowed to stand an indefinite length of time without injury.

9. The flexibility is so great that the cell can be used under the most adverse circumstances.

10. The method of charging until the voltage has remained constant for one hour gives satisfactory results, but requires watching or the use of a recording voltmeter.

11. The continuous drop in voltage on discharge (about 10 per cent per hour) is a disadvantage against this cell, over the lead cell.

12. When referred to equal amounts of power delivered, the Edison battery weighs about 25 per cent less and costs about 25 per cent more than the Iron-clad Exide battery.

The paper was discussed at some length by Messrs. Clymer, Langmuir, Hering, Smith, Calmus, Schluenderberg, and Smith. Mr. H. H. Smith, of the Edison Storage Battery Company, said that the Edison battery had been placed on the market in a radically changed structure in 1908, and for this reason the literature prior to 1908 could not be relied upon as referring to the present Edison battery. He also said that the paper was based on isolated tests and that there were some inaccuracies and some discrepancies between the drawings and the tables. He criticised, especially, the voltage figures and said that the established average discharge voltage of the Edison battery at normal rate was 1.20 volts per cell. He thought that conclusion 5 of the authors was not substantiated by evidence in the paper and was merely an assumption and that conclusion 11 was hardly accurate. He also claimed that the use of recording meters was not reliable.

Dr. Bennett in his reply acknowledged some inaccuracies in the drawings which were due to a misunderstanding of the draftsman, but said that the figures in the tables were correct, though no very high accuracy was claimed for them, as had been clearly stated in the paper.

A New Type of Concentration Cells

A paper by Dr. Reinhard Beutner, of the Rockefeller Institute for Medical Research, New York City, discussed "concentration cells containing organic liquids immiscible with water."

A year ago he presented a paper before this Society on electromotive forces generated by systems containing immiscible electrolytic conductors. (This journal, Vol. X, p. 297.) He mentioned in that paper some investigations relating to biological applications of the electrochemical researches along this line carried out by him and Dr. J. Loch, in which it had been found that the skin of plants behaves like an electrode reversible with respect to various positive ions.

As he pointed out, a phenomenon of this kind was not yet known in electrochemistry, though certain investigations of Nernst and of Haber (which were there discussed at length) had shown that a phase junction between immiscible electrolytes acts like the surface of a metallic electrode. This did not explain how such an effect was possible.

More recent investigations which the author has carried out have proved that with a number of water-immiscible organic substances similar effects can be produced as with the skin of plants. This has led to further investigations into the electromotive properties of water-immiscible organic substances, and

the surprising result has been reached that cells may be built up from such organic compounds and aqueous solutions showing an electromotive force which nearly equals in magnitude the e.m.f. of galvanic cells with metallic electrodes. An explanation of these phenomena has been found possible on the basis of the well-known theories of chemical thermodynamics. The quantitative mathematical applications involved, however, many difficult questions.

The results of the experiments and theoretical researches of the author are given in four chapters, dealing with salicylic aldehyde as electrode, toluidine as electrode, experiments with mixtures of toluidine and salicylic acid which explain the nature of the e.m.f. under discussion, and e.m.f.'s. due to chemical reactions as distinguished from e.m.f.'s. due to mere concentration differences.

The president, Dr. Miller, congratulated the author on the ingenuity of his indirect methods of checking his results.

Concentration Changes in the Electrolysis of Copper Sulphate Solutions

A paper on the above subject, by Messrs. C. W. Bennett and C. O. Brown, of Cornell University, was presented by Dr. Bennett.

In the electrolysis of copper sulphate solutions with a rotating copper cathode the potential drop across the cell, with constant current, increases with an increase in the speed of rotation. In view of the fact that a more efficient stirring of the solution is obtained by increasing the rotation, concentration differences being thereby lessened, a drop in potential difference might be expected as a random prediction. Since this increase is rather large, and seemingly abnormal, it was deemed advisable to attempt an explanation of this phenomenon. In the present paper the details of this investigation are given. The results are summed up as follows:

1. There is a definite measurable increase in potential difference in the electrolysis of acid copper sulphate solution, as the speed of rotation of the cathode is increased from 1000 to 5000 r.p.m., for instance.

2. From Fig. 2, the increase indicated to maintain zero current is about 0.010 or 0.015 volt.

3. This can be accounted for by the increase in the rotating electrode potential, with increased rate of rotation.

4. The increase in the potential of the rotating electrode with increased rotation is due to the tendency to form cuprous ions, the increase being greater when conditions are favorable for the formation of cuprous ions, and *vice versa*. As the rotation increases, the stirring becomes more efficient, the cuprous ions are removed, and the solubility of the copper is therefore increased. This gives rise to a higher voltage.

5. The potential of the rotating electrode (cathode when current is flowing) may be much higher than that measured here without current, probably due to the removal of cuprous ions from the solution by precipitation.

6. The increase in the drop of potential with increasing rate of rotation of the cathode in the copper-copper sulphate electrolysis is probably due in part to an increased resistance.

7. The decrease in the potential of the electrode in cuprous chloride solution is due to the more rapid removal of the cupric ions with stirring, thus removing the reagent dissolving the copper.

8. The potential of a rotating electrode in nitric acid goes down with increasing rate of rotation, due to throwing off more and more thoroughly the film of nitrous acid, thus decreasing the solution of the copper, and therefore its voltage.

9. The solution pressure of a metal is increased if the size of its crystals be decreased. This is made apparent by an increase in the potential of the metal as the crystal size is decreased, by hammering or burnishing, for instance. It is also shown by the fact that fine-grained metal (electrolytic copper) is electro-positive to more coarsely crystalline metal (cast copper).

The paper was briefly discussed by Drs. Miller, Langmuir, and Richards.

Hyperbasis

On Friday evening Prof. F. B. Kenrick, of the University of Toronto, gave a most interesting experimental lecture entitled "Hyperbasis"—a word introduced to cover such phenomena as superheating, supercooling, and supersaturation. The experiments, which included several new applications of projection, were shown on the screen with the help of a Bausch & Lomb balopticon.

The demonstrations were brilliant and brought out again the wonderful experimental skill of Professor Kenrick.

Although for the most part the study of hyperbasis has led to somewhat irregular and indefinite results, there are a few empirical rules which may be stated in a general way.

In the first place, it has been found practically impossible to supersaturate a liquid in respect to another liquid to any appreciable extent. In illustration of this a flattened tube containing a solution of phenol and water was shown on the screen alongside a thermometer. These were contained in a water cell the temperature of which was alternately lowered and raised. The liquid in the tube became suddenly opaque and transparent, alternately, as the saturation point was passed.

Secondly, liquids when slightly superheated or slightly supersaturated with a gas produce bubbles only on contact with a gas or a porous substance containing a gas. A tube of ether was shown on the screen and heated to 50 deg. C. without any sign of boiling. A capillary tube of air and a grain of pipe-clay caused violent ebullition, while a platinum wire caused only a temporary disturbance.

Thirdly, the crystallization of liquids, slightly supercooled or slightly supersaturated with a solid, is brought about only by contact with the solid itself or with an isomorphous substance. Two experiments illustrated this rule. A dish with CuCl_2 previously painted on the bottom with copper chloride solution and dried was shown by vertical projection. The symbol CuSO_4 was then scratched on the glass with a crystal of blue-stone, and a supersaturated solution of copper sulphate poured into the dish. The CuCl_2 disappeared at once, while the invisible CuSO_4 was gradually developed by precipitation of blue-stone crystals. Next a plate covered with a supercooled solution of m-Chlorinorbenzol in Canada balsam was scratched with the corresponding p-chlor and m-brom compounds; the former had no effect, while every movement of the m-brom compound was followed by a train of crystals.

The next part of the lecture dealt with more extreme cases of hyperbasis in which reaction takes place spontaneously and often in an apparently quite capricious manner. Even here, however, certain irregularities might be observed. The influence of the quantity of substance was shown by gradually heating simultaneously three tubes of ether of different sizes. The contents of the largest tube exploded at about 65 deg.—30 deg. above boiling point. In the second tube the ether did not explode till the temperature had risen many degrees higher, while the smallest tube was heated to 90 deg. without hyperbasis being relieved spontaneously, although the ether could be seen evaporating with great rapidity at the free surface.

This was followed by a number of experiments explanatory of the conditions under which spontaneous crystallization of liquids occur. The relation between temperature and linear rate of crystallization was first shown by allowing melted vanillin to crystallize in a capillary tube. The temperature was measured by the projection thermometer, the distance by the scale divisions on the same thermometer, while the time was indicated by the shadow of a pendulum, heating seconds. At intervals of 20 seconds the temperature of the cell was changed and successive readings were marked on a plate of smoked glass. By the time the crystallization had reached the end of the tube the complete curve had been drawn, showing the initial increase in rate below the melting point, the well-known interval of constant rate, and the final rapid decrease in rate with falling temperature.

The dependence of the number of nuclei on the temperature and duration of supercooling and on the presence of admixtures was shown in a tube of vanillin by Tammann's "exposure

and development" method. A double water-cell was used, each half containing a transparent thermometer visible on the screen. The cell was connected to a hot and cold water system by which each half could be rapidly heated or cooled to the desired temperature. In this cell all the operations of melting, exposure, and development could be followed on the screen.

The various forms of vanillin and the relative number of nuclei of each form were shown in the projecting microscope with polarized light. In these experiments also the effect of time and temperature of exposure was illustrated.

The importance of Tammann's work on nuclei was emphasized in connection with the preparation of amorphous substances and of various crystalline forms of the same substance, and also in its bearing on the structure of metals and alloys.

A curious experiment, the result of which the lecturer did not attempt to explain, was shown by adding a dilute magnesia mixture to a dilute solution of sodium phosphate, and then scratching the bottom of the vessel with a rod at intervals of five seconds. The crystals of ammonium magnesium phosphate appeared only where the glass had been scratched fifteen seconds or more after mixing. It was suggested that this might be due either to the slow formation of nuclei or to the time necessary for a very intimate mixing of the liquids.

The lecturer concluded with an experiment on the electrolysis of potassium hydroxide solution with a mercury cathode (Hulett). The surface of the mercury was shown by the reflectoscope while a current of 8 amperes was passing. Although a considerable quantity of potassium was deposited in the mercury no evolution of hydrogen took place at first except when the mercury was touched with a platinum wire. Finally solid crystals of potassium amalgam separated out, and when these covered the surface of the mercury violent effervescence occurred.

Saturday Sessions

The two sessions of Saturday were devoted to the presidential address of Dr. Miller and the symposium of papers on the electrodeposition of metals. The meeting was practically interesting as the National Electroplaters' Association had been formally invited to attend it and discuss the papers, and some twenty or thirty of its most prominent members were present. The meeting, therefore, gave an opportunity for a very interesting exchange of views and ideas between scientific electrochemists, industrial electrorefiners and practical electroplaters.

Presidential Address on Electrode Reactions

In opening the symposium, the president, Dr. W. Lash Miller, of the University of Toronto, expressed the thanks of the Society to those members who had undertaken the laborious task of collecting and arranging the literature of the subject, and had thus rendered possible the discussions of the day. The subject, he said, had been brought before the Society nine years ago at the time of the St. Louis fair in a paper by Dr. W. D. Bancroft on "the chemistry of electroplating," in which the author, among other matters, emphasized the analogy between the structure of metals deposited electrolytically by a heavy current and the structure of precipitates produced rapidly by purely chemical means. In so doing, Dr. Bancroft laid the foundation of the only theory we possess of the direct effect of current density on structure, and was one of the first to preach the modern view that electrochemistry is a branch of chemistry, with all that that implies. Faraday's law and $\log c_1/c_2$ are, of course, as essential as ever, and electrochemical phenomena have not ceased to obey the laws of energy; but it is now recognized that they obey the Ostwald-Bancroft law as well, according to which the thermodynamically "most probable" reaction is in practice perhaps the least likely to occur.

More than any other one subject, the study of the effect of current density on the reactions at the electrode has held the interest of electrochemists during the past decade. In a few instances a change in the nature of the product of electrolysis with rise in current density has been observed, as when

Skrabal showed the analogy between the electrochemical reduction of chromic acid by heavy currents and the ordinary chemical reduction of the same substance by quick-acting chemical reducing agents; but in most of the cases studied, no difference in the final products with different current densities has been noted, and the interest has lain in the endeavor to account for unexpected polarization.

It has long been known, of course, that electrolysis may cause differences of concentration at the two electrodes, and thus build up a polarizing electromotive force; indeed, until it was shown by an application of the theory of diffusion such concentration differences might be calculated, the tendency was to ascribe all polarization to this cause. The diffusion theory as first used, by Weber, Sand, and Warburg, was applicable only to cells in which convection—whether caused by stirring, gas evolution, temperature changes, or concentration differences—was absolutely excluded. This condition is with difficulty complied with, and rapid progress dates only from the application, by Nernst, Brønner, and Merriam, of Noyes and Whitney's theory of a surface film of liquid adhering to the immersed solid, through which the constituents of the solution can reach the electrode by diffusion and migration only.

It is now possible to calculate the concentrations at the surface of a rotating electrode from the current density, diffusion constant, and other easily accessible data, so that in any given case the amount of polarization due to concentration differences may be calculated and compared with that actually observed. In the case of copper deposition from a copper sulphate solution, Sand observed a polarization much greater than could be accounted for by the concentration effect, and the same is now known to hold for many other metals. Sand ascribed the excess to a "resistance." LeBlanc to the time needed for the dehydration of copper ions, Reichenstein to the deposition of a copper-hydrogen compound as primary product of electrolysis, while Bennett, in his paper of the day before, showed that under certain assumptions the effect might be regarded as due to low concentration of cuprous sulphate at the cathode.

Working under Reichenstein's conditions, with slightly acid copper sulphate solution, Mr. Burt-Gerrans and Mr. Brant were able, like him, to reach the high cathodic overvoltage of 0.7 volt. On adding nickel sulphate to the solution, the overvoltage persisted for ten minutes or so, but no trace of nickel was deposited. Similarly, when the electrolyte contained copper and tin, or silver and copper respectively, and the overvoltage was sufficient, on paper, to deposit tin in the first instance and copper in the second, no traces of these metals were in point of fact deposited. Such negative results discredit explanations based on rates of reaction, intermediate alloy, and ion concentration; the more so that in the experiments next to be referred to positive results were forthcoming.

Ever since Smale's determination of the e.m.f. of the oxygen-hydrogen cell was shown by Haber to be 0.2 volt or so below the thermodynamically reversible e.m.f., the "oxide theory" of Nernst, Luther, and Lorenz—that the e.m.f. of an oxidation or reduction cell with "unattainable" electrodes is due to the existence of oxygen or hydrogen compounds of the electrode metal—has been generally accepted. According to this view, the long delay which Luther found to occur before a "chromic acid electrode" reached its stationary e.m.f. must be ascribed to slow oxidation of the platinum by the chromic acid; the reaction between the electrode and iodine, on the other hand, is practically instantaneous.

This theory can be checked by an experiment analogous to that with the copper and nickel, by introducing a cathode into a solution in which chromic acid is slowly being reduced by an iodide. The e.m.f. thermodynamically "necessary" to reduce the chromic acid is less than that needed to reduce the iodine; experiments carried out by students show, however, that, in fact, iodine is reduced. Similarly when the chromic acid is replaced by arsenic acid; in this case the reverse reaction also is accessible, and in solutions where arsenious acid was being oxidized by iodine, an anode was found to bring about the "less probable" oxidation of the iodide.

The most striking case of this type was that studied by Mr. Welsh some years ago, the details of which are unfortunately not yet in print, in which the cathode was found to reduce iodine, not ferric salt, in solutions where all the time the ferric salt was slowly being reduced by an iodide. This case, even more than the others, emphasizes the oversimplicity of the older electrochemical view of these "ion reactions," and of the graphic method by which on the blackboard a ferric salt, Fe^{+++} , is reduced by rubbing out a +.

In closing, Dr. W. Lash Miller referred to the catalysis of these oxidation reactions by platinum and the absence of catalysis by gold, connecting this with the difficulty experienced in determining the e.m.f. of a "chromic acid electrode" in which the metal used was gold. He also spoke of the application of the theory of diffusion to cases where a precipitate is formed at the electrode, illustrating by a very interesting demonstration the periodic phenomena discovered by Küster during the electrolysis of a solution of sodium sulphide.

Symposium on Electroplating

In the symposium on electroplating five papers were presented, all of which had been prepared at the request of the President of the Society, Dr. Miller. These papers were intended to compile all available literature on electroplating, to compile all recipes for plating the different metals and all other information on electroplating that could be found in the literature. This set of papers, therefore, represents a real compendium on electroplating. As these papers were themselves concise abstracts of the available literature, it is clearly impossible to abstract them again, and only the contents of the different papers will be given here. They will at least show the enormous amount of information stored up in these papers. For the full papers the readers must be referred to the next volume of the Transactions of the Society.

Electrodeposition of Gold and Silver. By Dr. Francis C. Frary, University of Minnesota (73 pages). Part I: Electrodeposition of gold: simple halide baths, cyanide baths, gold chloride in potassium cyanide, gold cyanide in potassium cyanide, fulminating gold in potassium cyanide, miscellaneous cyanide baths containing little or no cyanide, baths to produce special colors, general information. Part II: Electrodeposition of silver: Simple salts, cyanide baths, silver nitrate in potassium cyanide, silver chloride and potassium cyanide, silver cyanide in potassium cyanide, potassium cyanide and miscellaneous salts, baths for bright deposits, general information on cyanide baths, ferrocyanide baths, miscellaneous chloride mixtures, miscellaneous non-cyanide solutions, alloyed deposits, general information on silver plating.

Electrodeposition of Nickel and Cobalt. By Dr. Oliver P. Watts, University of Wisconsin (53 pages). Electrodeposition of cobalt, fifty-one solutions for electroplating with cobalt, deposition of an alloy of cobalt and nickel, electroanalysis (fourteen solutions for estimation of cobalt and nine solutions for separation of cobalt and nickel). Electrodeposition of nickel, purity of nickel anodes, passivity of nickel anodes. Nickel-plating baths: Baths containing single salts of nickel (fourteen neutral, three acid, and nine alkaline baths); baths containing double salts (twelve neutral and eighteen acid baths); thirty miscellaneous baths: malleable nickel (three solutions); three baths producing thick deposits; addition agents for the production of smooth deposits; nickel electrolytes; the nickeling of zinc; electrolytic determination of nickel.

Electrodeposition of Copper. By Dr. C. W. Bennett, Cornell University (18 pages). Eight acid solutions; twenty-three alkaline solutions; addition agents.

Electrodeposition of Brass and Bronze. By Dr. C. W. Bennett, of Cornell University (10 pages). Twenty brass solutions; eleven bronze solutions.

Electrodeposition of Lead. By Dr. Frank C. Mathers, of Indiana University (38 pages). Lead salts of inorganic acids (fluorine acids and perchloric acid); lead salts of nitrogen acids for the deposition of metallic lead; salts of other in-

organic acids; lead salts of acetic, oxalic and other organic acids; alkaline solutions; lead peroxide deposition; non-aqueous solutions; miscellaneous; electroplating and coloring with lead.

Electrodeposition of Tin. By Dr. Edward F. Kern, of Columbia University (23 pages). Electrodeposition of tin by immersion; electrolytic determination of tin, with reference to the use of rotating electrodes and the removal of tin deposits from the cathode; electrodeposition of tin by separate current (nineteen solutions); conditions necessary to obtain bright, dense, adherent deposits of tin; electrolytic refining of tin; effect of organic addition agents in tin electrolysis (sodium-stannous chloride electrolyte, stannous fluoride electrolyte, sodium-stannous fluoride electrolyte).

The president, Dr. Miller, welcomed courteously the members of the National Electroplaters' Association, which had come to join in the discussion. For the National Electroplaters' Association hearty replies were made by Mr. Proctor, founder of association, and Mr. Hogaboom, secretary of the association.

Dr. Wilder D. Bancroft presented the following set of axioms of electroplating.

1. Bad deposits are due to excessive admixture of some compound or to excessively large crystals.
2. Excessive admixture of any compound can be eliminated by changing the conditions so that the compound cannot precipitate.
3. Increasing the current density, increasing the potential difference at the cathode, or lowering the temperature decreases the size of the crystals.
4. The crystal size is decreased when there are present, at the cathode surface, substances which are adsorbed by the deposited metal.
5. If a given solution will give a good deposit at any current density, it will give a good deposit at any higher current density, provided the conditions at the cathode surface are kept constant.
6. Treating is facilitated by a high potential drop through the solution and by conditions favorable to the formation of large crystals.

The discussion was long and at all times animated, interesting, and suggestive. On the side of the American Electrochemical Society the chief speakers, besides Dr. Bancroft, were Dr. Miller, Dr. Bennett, Dr. Richards, Dr. Keith, Mr. Cowles, Mr. Addicks and Dr. Cushman; the chief speakers for the electroplaters were: Mr. Hogaboom and Mr. Proctor; a middle ground was taken by Dr. Kern and Mr. Brown.

It was pointed out by several speakers that the two sides don't yet understand each other. As Mr. Hogaboom said, practical electroplaters don't understand many of the scientific expressions, while scientific electrochemists don't understand the practical conditions under which a plater works. The conditions in the plating shop are very different from those in the laboratory. In the plating shop the electrodes are fixed; there is no physical possibility to regulate the distance between anode and cathode. Then there are perhaps 50 jobs in one big tank and a heavy deposit must be obtained of exactly the right color in a short time. For instance, when bar bottles are to be silver-plated, forty-eight bottles are put in at the same time and they are to be taken out at the same time. It is easy enough in the laboratory to measure and adjust the current density. But in the plating shop that would mean first to answer questions like this: How many square inches is the surface of a gross of table forks? There is a great thirst for knowledge among electroplaters—thirst for chemical knowledge.

Mr. Lidbury suggested that electroplaters should give up the use of terms like ounces per gallon, degrees Baume, and the like and go over to the metric system. Dr. Miller praised Hering's excellent Conversion Tables and recommended them to the Electroplaters' Association, while Mr. Hogaboom suggested that conversion tables may also be used backwards.

As Mr. Hogaboom pointed out, the fundamental unit of the electroplater is the volt. He talks volts because his experience shows that he gets the desired result if he uses so and so many volts.

Dr. Kern pointed out that the difference between the ways of electroplaters and electrochemists is more apparent than real. He showed in detail that while one specifies volts and the other current densities, both are in fact doing the same thing.

Mr. Addicks said that the ordinary plating solutions should be adjusted periodically and brought back to standard as is done in electrolytic copper refineries.

Dr. Kern remarked that if it is said that a solution in a plating shop is 18 years old, this must not be taken literally. Water is added from time to time. Salts are added. The plater really maintains a fair degree of purity. His control is by adjusting the degrees Baume.

Mr. Addicks said that platers should not insist so strongly on absolutely pure salts. For the sake of fair play they should permit the salts to contain a reasonable degree of impurities just as the water and the anodes are not absolutely pure.

Mr. P. S. Brown recommended to encourage the manufacturer to send samples to chemists if he does not want to employ chemists himself. Of greatest importance is the organization of a plating shop on the basis of scientific management—"Taylorizing" a plating shop. If this is done, it can always be shown why a thing pays. It could be easily shown in this way why chemical control pays in a plating shop. The old antagonism of the plater against the chemist is now vanishing and things have been changed greatly.

Dr. Richards remarked that it is often said that the Germans produce things so cheaply. The reason why this is so is that no German plant is without a chemist. The chemist reduces the total cost instead of adding to the expense.

With respect to **gold plating**, Mr. Hogaboom pointed out that a gold deposit reflects the color of the surface on which it is deposited. The reflection of the light from the surface under consideration determines the solution to be used. There is a great difference between inside and outside color if the same solution is employed for inside and outside gold-plating. Outside gold is translucent. If the outside is first plated with copper and the gold is then plated on it, it is possible to get the same color outside and inside. Mr. Hogaboom also referred to additions of arsenic or lead in the deposition of gold and said that it is a disputed question whether arsenic or lead is better.

The importance of getting a right color was also emphasized in connection with **bronze plating**. When the chemist speaks of bronze plating he means deposit an alloy of copper and tin. When the electroplater speaks of bronze plating he means getting a color that is the true color of bronze. Some of the so-called bronze-plating solutions contain copper and zinc, a few others copper and nickel. The much greater cost of tin as compared with zinc is, of course, an item of commercial importance.

With respect to **copper plating** Mr. Hogaboom said that acid solutions are better than alkaline solutions from the electroplater's standpoint and that all heavy copper deposits require an acid solution. The only one satisfactory addition agent for acid sulphate solution for copper plating is, according to Mr. Hogaboom, common alum.

It was to **nickel plating** that quite a considerable part of the discussion referred during the day. Mr. Hogaboom said that the necessity of copper plating as a preliminary basis for nickel plating on iron, steel, etc., was questionable and referred to the success of direct nickel plating on hammer heads. He also emphasized the enormous amount of damage which a small amount of copper can do in a nickel solution.

Mr. Brown said that there was a reason for copper plating before nickel plating. It has essentially a cleansing effect and yields, therefore, better adherence and better deposits.

Mr. Hogaboom emphasized the commercial difficulties in nickel plating by stating that nickel plating of the common three-hole gas plate must be done for 23 cents and that this has to pay for labor, nickel, and everything.

Dr. Bennett thought that pure electrolytic nickel anodes would be better for nickel plating than the ordinary nickel anodes, which always contain iron.

Mr. Hogaboom replied that one of the largest manufacturers of nickel anodes in this country bought a great many tons of electrolytic nickel and could not melt it and cast it into anodes by the ordinary methods.

Dr. Bancroft thought that this fact suggested the advantages of trying something on a laboratory scale before trying it on a ton scale.

Dr. Richards thought that manufacturers of electrolytic nickel might be induced to produce nickel anodes directly.

Mr. Hansen said that electrolytic nickel could be successfully melted in an electric arc furnace and cast with homeopathic quantities of silicon and aluminum and could be forged, and that they had done it with 1000-lb. batches.

With respect to lead plating Mr. Proctor said that electroplaters were interested in it only when they were called upon to produce "royal copper" and metallochromes (iridescent surfaces).

Dr. Keith spoke of his early work with electrodeposition of lead.

With respect to tin Dr. Keith gave interesting reminiscences of electrolytic tinplating in very early days.

Dr. Cushman spoke at some length on the determination of tin in foodstuffs contained in tin cans. This is a matter which is of immense importance at present from a commercial viewpoint.

With respect to addition agents Mr. Addicks referred to the glue in the barrels used in the early days of nickel plating and called attention to the almost universal use of adding some common salt to the electrolytic copper refining solution, though the reason for this is not known. He also emphasized that electrolytic refining, electrotyping and electroplating are very different in character and scope.

With reference to addition agents and metallurgical flotation processes a pretty demonstration was made by Professor Bancroft. A complete account of the discussion will be found in the next volume of the *Transactions of the Society*.

Committee Appointments.

Dr. Baskerville spoke briefly of the prevention of occupational diseases in the chemical industries and made a motion that a committee of the American Electrochemical Society be appointed to act together with a similar committee of the American Chemical Society. The motion was carried unanimously, and President Miller appointed Mr. F. A. Lidbury, chairman of the committee, together with Mr. A. H. Hooker and another member still to be appointed.

In order to continue the co-operation between the American Electrochemical Society and the National Electroplaters' Association a committee was appointed to summarize what has been accomplished so far and especially to work out simple standard methods of analyzing plating baths, to work out standard plating baths, and to account for many of the puzzling phenomena noted by practical platers.

Dr. W. D. Bancroft is the chairman of this committee, and Mr. Hogaboom and Mr. Proctor are the representatives of the Electroplaters' Association on the committee, while the committee has power to add further members of the American Electrochemical Society.

In the following we give a complete alphabetical list of the names of members and guests who registered at the Atlantic City meeting:

Paul O. Abbe, New York; Lawrence Addicks, Chrome, N. J.; Thos. B. Allen, Niagara Falls, N. Y.; Richard Amberg, Harrison, N. J.; G. Aminoff, Baltimore, Md.; W. C. Arsem, Schenectady, N. Y.; W. Aspf, Philadelphia, Pa.; Chas. W. Bailor, Philadelphia, Pa.; Mr. and Mrs. T. F. Bailly, Alliance, O.; Wilder D. Bancroft, Ithaca, N. Y.; Samuel Barr, Philadelphia, Pa.; Chas. Baskerville, New York; E. A. Beck, New York; E. H. Bedell, New York City; C. W. Bennett, Ithaca, N. Y.; R. Heutner, New York; R. S. Bicknell, New York City; H. Bloodworth, Lansdowne, Pa.; Earl Blough, Pittsburgh, Pa.; J. E. Breckenridge, Woodbridge, N. J.; J. P. S. Brown, New York City; de Coney B. Browne, New York; Chas. L. Bryden, Scranton, Pa.; J. H. Byrne, Cleveland, Ohio; W. R. Clymer, Cleveland, Ohio; Mr. and Mrs. A. M. Coney, Chester, Pa.; W. A. Cowan, New York; Mr. and Mrs. Alfred S. Cowles, Sewaren, N. J.; Edwin L. Crosby, Detroit, Mich.; E. Crossman, New York City; C. R. H. Cunningham, Lansdowne, Pa.; Alerton S. Cushman, Washington, D. C.; Newton E. Daholt, Pittsfield, Mass.; Homer T. Darlington, Natrona, Pa.; R. W. Davis, Jr., Aspinwall, Pa.; C. Dittmar, New York City; Earl Eastman, Atlantic City, N. J.; Mr. and Mrs. Francis A. J. Fitzgerald, Niagara Falls, N. Y.; Harry C. Flanagan, New York City;

K. G. Frank, New York City; Milton W. Franklin, Schenectady, N. Y.; A. E. Gibbs, Philadelphia, Pa.; Carl H. Gille, Philadelphia, Pa.; H. W. Gillet, New York; J. S. Goldbaum, Norwood, Pa.; S. L. Goodale, Pittsburgh, Pa.; Mr. and Mrs. Thos. D. Green, Montclair, N. J.; G. A. Hansen, Schenectady, N. Y.; August Heit, Philadelphia, Pa.; H. Hennauer, Philadelphia, Pa.; Carl Hering, Philadelphia, Pa.; A. T. Hinckley, Niagara Falls, N. Y.; Alean Hirsch, New York City; George B. Hogaboom, Newark, N. J.; L. S. Holdstein, Palmetton, Pa.; Mr. and Mrs. A. H. Hooker, Niagara Falls, N. Y.; O. T. Hungerford, Belleville, N. J.; W. R. Ingalls, New York City; Mr. and Mrs. Alois von Isakovics, Monticello, N. Y.; H. T. Kalmus, Kingston, Ont., Can.; N. S. Keith, Philadelphia, Pa.; H. W. Kellogg, Niagara Falls, N. Y.; Philo Kemery, Pittsburgh, Pa.; Frank B. Kerriek, Toronto, Ont., Can.; Edward F. Kern, New York City; A. S. Knight, Niagara Falls, N. Y.; J. Ludwig Koethen, Jr., Atlantic City, N. J.; Frederick L. Koethen, Niagara Falls, N. Y.; W. F. Kuntley, East Orange, N. J.; Mr. and Mrs. Irving Langmuir, Schenectady, N. Y.; F. A. Lidbury, Niagara Falls, N. Y.; Chas. E. Lindsay, Bridgeport, Ct.; Stewart J. Lloyd, University, Ala.; B. E. Love, Baltimore, Md.; D. A. Lyon, Pittsburgh, Pa.; G. M. J. Mackay, Schenectady, N. Y.; Wallace W. Mann, Cleveland, Ohio; C. W. Marsh, Niagara Falls, N. Y.; G. Stanley Merkle, Schenectady, N. Y.; W. Lash Miller, Toronto, Ont., Can.; Mr. and Mrs. Harlan S. and Miss Kern, Minner, Gloucester, N. J.; Walter S. Moody, Pittsfield, Mass.; Mr. J. M. and Miss Helen Muir, New York City; Mr. and Mrs. E. F. Northrup, Princeton, N. J.; J. J. Owens, Newark, N. J.; E. J. Owens, Newark, N. J.; Chas. L. Parsons, Washington, D. C.; N. J. Smith, New York City; N. Y.; William T. Price, Philadelphia, Pa.; Chas. H. Proctor, Arlington, N. J.; Dr. and Mrs. Jos. W. Richards, South Bethlehem, Pa.; E. D. Rippel, Buffalo, N. Y.; E. P. Roerber, New York City; G. A. Roush, South Bethlehem, Pa.; S. J. Sadler, Philadelphia, Pa.; Pedro G. Salom, Philadelphia, Pa.; L. E. Saunders, Niagara Falls, N. Y.; Carl G. Schluederberg, Pittsburgh, Pa.; Wm. J. Schneider, New York City; Job. Schoenawa, Menzschwand, Germany; Frederick F. Schuetz, Newark, N. J.; Mrs. J. S. Scott, Philadelphia, Pa.; J. A. Seede, Schenectady, N. Y.; Philip Stevering, Newark, N. J.; Dyer Smith, New York City; Mrs. G. D. Smith, Montclair, N. J.; Harold H. Smith, Orange, N. J.; H. H. Smith, Newark, N. J.; Mr. and Mrs. Acheson Smith, Niagara Falls, N. Y.; F. D. Steik, Passaic, N. J.; J. E. Sterling, New York; Chas. A. Stuebel, Newark, N. J.; Walter T. Taggart, Philadelphia, Pa.; Mr. and Mrs. F. F. Tombaugh, Alliance, Ohio; H. M. Turner, Jr., Pittsburgh, Pa.; C. H. Vom Baur, New York; L. D. Voon, Detroit, Mich.; M. G. Weber, Perth Amboy, N. J.; Chas. A. Weeks, Philadelphia, Pa.; R. J. Wetlaner, Philadelphia, Pa.; Arthur B. Wells, Philadelphia, Pa.; R. H. White, N. Y.; Mr. and Mrs. A. S. White, Philadelphia, Pa.; W. R. Whitney, Schenectady, N. Y.; F. C. Woodside, Pittsfield, Mass.; F. Zimmermann, Newark, N. J.

Coming Meetings.

The next meeting of the American Electrochemical Society will be held in Denver, Col., early in September, and the principal feature of this meeting will be a symposium of papers on what electrochemistry can do for Western metallurgy.

The spring meeting of 1914 will be held in New York City. For the autumn meeting of 1914 the society has received a courteous invitation to go to Birmingham, Ala., and see the New South.

In 1915 one of the meetings will be held in San Francisco in connection with the world's fair.

Dr. Joseph W. Richards, Lehigh University, South Bethlehem, Pa., is secretary of the society.

Filter Press.—The Berrigan filter press has been taken over by the Hy. R. Worthington Pump Company, at Harrison, N. J.

British Iron and Steel Institute.—The annual meeting of the (British) Iron and Steel Institute will be held in London on May 1 and 2. The Bessemer gold medal will be awarded to Mr. Adolph Greiner, general director of the Société Cockerill, Seraing. The autumn meeting will be held at Brussels.

The lead production of the United States for 1912 has been finally announced by the Geological Survey. The total production of refined lead was 480,894 short tons, which is 71 tons less than the estimate given out on Jan. 2, 1913. This is exclusive of 13,552 tons of antimonial lead. The total refined lead output was less than that of 1911 by 6085 tons, a decrease of 1.2 per cent. Missouri had the greatest output, 162,610 tons, a decrease of about 20,000 from the previous year. Idaho was next with 127,707 tons, a gain of about 10,000 tons over 1911.

The iron ore deposits at Minaret, Madera County, California, are said to be among the largest in California and perhaps in the West. The next in size, but of greater commercial importance, are the Eagle Mountain deposits, in Riverside County, California. The latter have been examined by the Geological Survey and reported in *Bulletin 503*. A moderate estimate of the total quantity of mixed ore and gangue material available in the deposits now exposed, is about 75,000,000 tons. About four-fifths of this is assumed to be ore of very high grade. The principal obstacle to the iron industry in California is the absence of fuel. The future operation of the Alaska coal fields is expected to stimulate iron reduction and steel refining in California.

Notes on Chemistry and Metallurgy in Great Britain

From our Special Correspondent

The Institute of Metals

The annual general meeting of the Institute of Metals was held at the Institution of Mechanical Engineers on March 11 and 12, 1913. In the course of his presidential address, Professor A. K. Huntington said it was usual in a presidential address to call attention to the lines on which the Institute could be most advantageously developed, and a study of the purposes and progress of other similar institutions of longer standing would afford assistance in this direction.

In 1894 in his presidential address to the Institution of Mining and Metallurgy he said "the progress in the iron and steel industries in the last twenty-five years has been extraordinary, and I have no hesitation in saying that a large share of it is due to the existence of the Iron and Steel Institute. The members of those industries are no longer working against one another in a narrow spirit, afraid to confide their ideas to one another lest they should give a competitor an advantage. All this narrowness has disappeared, and now members visit one another's works and discuss everything concerning them at every opportunity. As in everything else the best man wins in the long run, while the rest of the world is benefited by more rapid progress."

At the time that address was made, he had in mind many works in this country where metals were smelted and refined which preserved secrecy as to their doings, and would not admit anyone to their works. Now, notwithstanding that the Institution of Mining and Metallurgy had been in existence twenty-one years, these same works, for the most part, maintain that same attitude: they contribute nothing whatever to that institution. That institution had had its time fully taken up with metallurgy as carried out in direct connection with mines, and probably had hardly given the works in question a thought; and judging from the experience of the Institute of Metals, it would probably have only been rebuffed had it acted otherwise. He did not think the Institution of Mining and Metallurgy would object to such works coming under the Institute of Metals. Perhaps they would come into line in time, but that time was not yet.

The way in which the Institute of Metals had progressed during the few years of its existence left no doubt, to his mind, that the non-ferrous industries would respond to its stimulus just as the iron and steel industries had done in the case of the Iron and Steel Institute. There was no doubt, too, that the Society of Chemical Industry had done good work in the field covered by the Institute of Metals; and they should be grateful to that society, which would be the first to admit that the time was now ripe for specializing more fully in this subject.

It must be remembered that metallography for practical purposes had only come into existence within about the last fifteen years, and the difference it had made in explaining what occurred to metals and alloys in the course of manufacture and in use was incalculable.

It was easy now to appreciate the fact that the knowledge to be gained from chemical analysis of an alloy was as nothing compared with that which could be obtained by studying the internal condition and structure of metals and alloys when their composition was varied, and they were subjected to suitable or unsuitable mechanical and heat treatment.

In dealing with copper-zinc alloys in the alpha-beta range it had been the practice in his laboratory for several years to employ a method of micrographic mensuration which occupied quite a short time compared with the chemical analysis formerly required.

This was only the fifth year of the existence of the Institute, and the membership was 606. At the same period of its existence the Iron and Steel Institute had 587 members, and the Institution of Mining and Metallurgy reached practically the same figure, 585, in its seventh year.

The position between the manufacturer and the user was somewhat delicate. If the manufacturer inquired of the user as to the behavior of the material he had supplied, the user might turn round and say, "Dear me! What is the meaning of this? Evidently he has no confidence in the material himself after all he has said about it." So it came about that the manufacturer waited till the user went for him, and in the very human desire to save his own skin he probably retorted that what had gone wrong was owing to improper usage; and in ninety-nine cases out of a hundred he was most likely right.

What was required was more mutual confidence, and when users realized that other men were quite as straightforward as themselves a great advance would be achieved. It may be taken as certain that manufacturers were in general straightforwardly doing their very best to acquire or maintain a good reputation. His personal experience was that the greatest difficulty he had had to contend with was that of obtaining reliable information from users, who very frequently had not got it.

Accurate observation demanded requisite training from very early youth upwards, and now that metallography and physical chemistry were taught in so many institutions young engineers and works chemists should be required to have competent knowledge of those subjects.

The institution was very often in the position to bring the manufacturer and the user into sounder relations by pointing out that users had everything to gain by carefully and systematically observing what was going on with the metals and alloys they were employing, and by giving friendly information to the manufacturer which would assist him in meeting their requirements. In this way the users would also learn a great deal and would realize that the old saying "Prevention is better than cure," most certainly led to economy and peace of mind.

Papers and Discussions

Abstracts of all the papers presented have already been given in our April issue, page 179. In the following we give a report of the discussion:

Corrosion of Distilling Condenser Tubes

A paper entitled "Contributions to the History of Corrosion (Part 2), having special reference to the Corrosion of Distilling Condenser Tubes" was presented by Mr. Arnold Philips, the Admiralty chemist. (See abstract on page 179 of April issue.)

In the discussion, Dr. Bengough said he thought that the author had proved his case, and that the hydrochloric acid was the cause of corrosion. Experience in connection with brass led him to accept this view. Corrosion of this description was more prevalent in the Mercantile Marine than in the Navy, probably owing to the more careless treatment of condensers in the Mercantile Marine.

Sir Gerard Muntz said that not very long ago it was the general impression that the Admiralty never had any trouble with their condenser tubes, and had attained a state of perfection. The paper gave point to a fact which he had been trying to drill into the heads of people for many years—namely, that the cause of trouble must not always be sought where the trouble is found, and it did not follow that if the cause of corrosion took place in a condenser tube that the trouble arose there. He had recently had the opinions of five distinguished scientists to the effect that they had never found corrosion to arise from anything due to the constitution of the tube.

Dr. G. H. Bailey did not accept the view that hydrochloric acid was necessarily the cause of the corrosion mentioned by the author. It always seemed to him that the dissolved oxygen in the water was an important factor, because when aluminium was exposed to the action of water which had been boiled until all the air was expelled, there was no action. The same thing occurred even with a strong salt solution when air was absent. Did the author say he was not able to determine the presence of hydrochloric acid because it was volatile? His own

experience was that the acid water could be concentrated by boiling so as to enable the amount of hydrochloric acid to be determined.

Mr. Rhead asked whether the author had ascertained that hydrochloric acid would attack copper in the absence of oxygen. The paper did not afford sufficient evidence to combat the accepted idea that copper was not directly attacked by hydrochloric acid, but that in the presence of oxygen it was acted on freely.

Mr. Phillip, in his reply, said that throughout the paper when referring to hydrochloric acid he had meant hydrochloric acid admixed with air. The evidence that the effects had been caused by hydrochloric acid and not by oxygen alone was quite clear because with totally submerged steam heating coils no corrosive effect was produced comparable with that which took place when the tubes were partly above the surface of the salt water. He believed that the greater cause of 80 to 90 per cent of corrosion occurring in the Navy was the presence of acids, together with electrolytic action. If the tubes were in metallic contact with the steel casing that would account for 99.9 per cent of the corrosion which he had seen.

The president remarked that some time ago a boiler stay of similar composition to those mentioned in the paper was sent to him. In places the zinc was removed and he found that the depth to which the zinc had been removed was in direct ratio of the thickness of the deposit, and at the time he came to the conclusion that the action was due to magnesium salts in the deposit.

The Corrosion of Aluminium

Dr. G. H. Bailey read a paper on "The Corrosion of Aluminium," in which he dealt mainly with the effect of water and solution of common salt. (See abstract on page 179 of April issue.)

Dr. Bengough opened the discussion and said that many of the factors affecting the corrosion of aluminium also affected brass, with the difference that in the case of aluminium the oxides were non-adhering, while with brass they adhered firmly. In some respects there seemed to be less difficulty with brass than with aluminium, but on the other hand, secondary reactions occurred necessitating investigation; yet there was not the same difficulty in determining the loss of weight caused by the formation of films of oxide. The idea of direct oxidation was anathema in certain quarters in relation to electrochemical studies; but Dr. Bailey had definitely come to the conclusion that it was an important point in connection with corrosion, and he himself (the speaker) had been gradually coming round to the same view with regard to brass, so that in certain respects he and Dr. Bailey were standing shoulder to shoulder against a great body of physico-chemical opinion.

Dr. Seligman said he did not know a more unscientific method than determining corrosion on the basis of weight. Four of Dr. Bailey's conclusions had already been arrived at by other workers, and he might have referred to those workers more explicitly. He considered that the amount of metal removed was quite unimportant, and, in support of this, produced samples of aluminium corroded by nitric acid and water respectively. He said that estimating the corrosion by the quantity of metal removed they would both be condemned, but the sheet of metal was still in a uniform condition, while the tube was decidedly damaged.

Mr. S. L. Archbutt was of opinion that the times of corrosion adopted by the author were too short. The experience at the National Physical Laboratory was that the formation of a protective coating might exercise considerable influence on the rate of corrosion, which could only be accurately arrived at by much longer periods of immersion than the author had employed. He asked the author how the specimens were immersed, because he had found that whatever supporting materials were used, corrosion was accentuated where the alloy was in contact with them, while friction might also be an important matter.

Mr. Arnold Phillip gave some figures showing that friction had very great influence on the rate of corrosion.

Dr. F. J. Brislee was inclined on the whole to agree with Dr. Bailey's conclusion that oxygen is an important factor in bringing about corrosion. The protective coating was also an important matter. In one case the material had been corroded gradually for three years exposed to spray and only in indirect contact with liquids, but it had subsequently remained practically unaffected. The importance of the protective coating was so fully recognized that patents were being taken out for using it to insulate electrical conductors, and he was astonished to see that a coating of aluminium oxide was flexible enough to permit of the winding of a wire of 18 to 24 gauge on a spool little more than an inch in diameter.

Mr. E. F. Law asked for more data, for he did not think that the results quoted in the paper justified the author's conclusions, although those might still be correct.

Dr. Bailey replied and objected to having been accused of quoting other workers' conclusions without acknowledgment. He had questioned those conclusions, and had therefore worked independently on them and had arrived at the same result. He would not maintain that the amounts of metal removed were the final test, but commercially it was of the greatest importance. He did not think the time of exposure was too short, because longer exposure would mean that the aluminium would be protected by air being driven out of the water.

Engineering Imports and Exports

The returns issued by the Board of Trade for January and February show satisfactory increases as compared with the corresponding period of last year, in respect of all the chief engineering materials except in imports of electrical goods, which are lower. The imports of iron and steel, including manufactures, were £2,683,229, an increase of £661,697; and exports touched £8,866,659, an increase of £959,222. In other metals including manufactures, imports are valued at £5,481,881, an increase of £443,890; and exports are put at £2,421,688, an increase of £543,816. Imports of electrical goods totalled £257,149, a decrease of £17,842; but exports went to £724,583, and increased by £196,503. The imports of machinery reached £1,103,050, an improvement of £169,434; and exports rose to £5,825,103, an increase of £811,805. Imports of new ships, value £2,012, show an increase of £1,505; and exports amounted to £1,123,421, a rise of £411,707. Taking the figures for February alone, the imports and exports of iron and steel show increases of £261,938 and £485,731 respectively; imports of other metals rose £259,360, and exports went up by £238,642; electrical goods exhibited a decline of £14,038 in imports, but exports were £102,076 better; imports of machinery rose £10,654, and the exports increased by £365,973; and new ships showed a rise of £259 in imports, while exports were £104,504 higher.

Market Prices, March, 1913

Copper has been fairly steady over the month, weakening a little about the 6th, but hardening over the last ten days. Opening at £64.7 6, it closes at £67.16 3.

Tin opened at £218, it was at first inclined to go higher, but weakened to £210 on the 13th, since recovering to £219½ and closing at £217.10 0.

Lead opened at £16.12 6 but fell away after the 5th, touching £16.2 6, on the 7th. It has since been stronger, but not above the closing price, £16.15 0.

Haematic opened at 80/6, was 79/- on the 4th, but had reached 82/- by the 15th. It had declined to 79/6 by the 28th and now closes 78 9.

Scotch Pig opened 66/7½ and shows the highest increase of the iron warrants. It rose to 70/- in the first week, then declined a little but recovered by settlement and closes at 72/4½.

Cleveland has improved fairly steadily throughout the month starting at 61/- gaining principally in the second week, it closes at 66/0½.

	£	s.	d.
Aluminium, ton	00.	0	0
Alum. lump, loose, per ton	5.	15.	0
Antimony, black sulphide powder, per ton	20.	0.	0

Borax, refined, cwt.....	17/6 to 18. 6
Copper ore, 10 to 25% per unit.....	11/4½ to 11.10½
Copper sulphate, per ton.....	23.12. 6
Carbolic acid, liquid 97 to 99%, gal.....	3
Caustic soda, ash, 48% ordinary, per ton.....	5.10. 0
Ebonite rod, lb.....	5. 3
Hydrochloric acid, cwt.....	5. 0
India rubber, Para, fine, lb.....	3. 8
Mica, in original cases, medium, lb.....	3/6 to 6. 0
Petroleum, Russian spot.....	9½
Sal Ammoniac, lump, 1st. del. U. K. per ton.....	44. 0. 0
Sulphate of ammonia, f. o. b. Liverpool.....	14. 0. 0
Sulphur, recovered, per ton.....	5. 5. 0
Shellac, cwt., standard orange spot.....	4. 4. 0
Platinum, oz.....	9. 5. 0
Tin ore, 70%, per ton.....	135 to 137. 0. 0
Zinc, Veille Montagne, f. o. b. Antwerp.....	28.10. 0

Higher.

Lower.

Aluminum, ton.....	£ s. d. 1. 0. 0	India rubber, lb.....	4
Alum, ton.....	5. 0	Sulphate of ammonia, ton.....	5. 0
Copper, ton.....	2.18. 9	Haematite.....	1. 9
Copper sulphate, gal.....	1		
Scotch Pig, ton.....	5. 9		
Cleveland.....	5.4½		

Synopsis of Recent Chemical and Metallurgical Literature

Iron and Steel

Concentration of Iron Ores.—In the issue of this journal for December, 1912, page 811, we presented an outline of a paper on this subject, read by Mr. N. V. Hansell, at the Cleveland meeting of the American Institute of Mining Engineers. In the *Bulletin* of the Institute for March, 1913, Mr. C. Q. PAYNE contributes a discussion of Mr. Hansell's paper, from which we take the following extract.

"The high grade of concentrates now produced from magnetic iron ores is particularly striking, as it is not uncommon to find concentrates containing from 67 per cent to 69 per cent iron, representing a recovery of 90 per cent and even 95 per cent. The type of magnetic field producing these results, both in the Gröndal (wet) and Ball-Norton (dry) systems, is one of alternating north and south poles, which was developed by Clinton M. Ball and Sheldon Norton. In the case of wet treatment, the improvement in grade of concentrates is due also to a well-devised scheme of mill treatment, whereby a rough concentrate is first obtained, then reground and re-separated."

In regard to the suggestion of Mr. Hansell to apply the concentrated or condensed-field type of magnetic separator to the wet treatment of feebly magnetic ores, like hematite and limonite, Mr. Payne believes that the suggestion opens an interesting field of inquiry. Mr. Payne did considerable work along this line some years ago, and gives some of the conclusions reached.

"In the first place, hematite particles cannot be so strongly magnetized in a condensed field as to inductively magnetize other hematite particles in their neighborhood, as is the case with magnetite particles. Theoretically, therefore, the sheet of ore fed into the field should be only one particle thick. This means that only a limited capacity may be expected from a relatively expensive machine. Moreover, those ores which have iron-bearing silicates among their gangue minerals, or red colored clays, are difficult to enrich, since for practical purposes such silicates or clays are equally as magnetic in a condensed field as hematite and limonite ore particles, and they are therefore carried with them into the concentrates."

In spite of these obstacles of small capacity and imperfect results, it is unwise to condemn the use of the condensed-field type of electromagnet for the treatment of hematite and limonite ores, either dry or wet, in advance of a careful study of the ore-occurrence in each instance. Mr. Payne inclines to the

opinion that a combination of gravity and magnetic separation will give the best results, commercial and technical, in the enrichment of many low-grade hematite ores which cannot be successfully treated by either system alone. Gravity treatment will give better results on coarse sizes; but on the other hand, this system results in high losses in the fine sizes and slime of such ore. It is in the treatment of the latter that magnetic separation accomplishes its best work. These facts suggest the combination of the two methods, each working on the most appropriate material. In such a combination treatment, the finely crushed middlings or tailings should be given a reducing roast to magnetize the iron ore particles, thereby making magnetic separation more efficient. The concentrates thus produced would be of high grade, and the loss would be low. The degree of concentration would be higher by the combination method than by gravity separation alone.

By way of illustrating the possibilities of the combination treatment, Mr. Payne summarizes the results from six Gröndal plants, and the corresponding results from the gravity separation plant of the Oliver Iron Mining Company.¹

	Crude % Fe	Concen- trates % Fe	Tail- ings % Fe	Re- covery %
Average of six magnetic magnetite-ore plants.....	35.2	67.9	6.2	90.1
One gravity Mesabi hematite-ore plant.....	35.0	58.0	14.1	80.0

Assuming that the less favorable results of the hematite-ore plant are largely due to the difficulties of treating fine material; also by way of illustration, that magnetic separation applied to this portion would raise the grade of the combined concentrates to 60 per cent iron, then it would be possible to secure 1 ton of such concentrates from 1.96 tons of crude ore instead of 1 ton of such concentrates from 2.1 tons of crude ore. The recovery would thus be increased to 87.4 per cent. Such assumed results would show an increased yield of 1191 tons of concentrates from the 35,000 tons of crude material treated in the mill per day; and taking \$2 per ton as the value of concentrates at the mill, would provide an increase of nearly \$2500 per day in the gross value of the concentrates from the same amount of crude material.

"The importance of these results will be realized when it is recalled that those iron-bearing deposits of the Lake Superior region alone which contain 35 per cent and more of metallic iron have been estimated to contain 67,640,000,000 tons."

Corrosion of Iron and Overvoltage

The meeting of the (British) Faraday Society, held on April 4, 1913, in Manchester, was devoted to a general discussion of the corrosion of iron and steel.

A paper by Mr. Bertram Lambert on an electrolytic theory of the corrosion of iron was first presented. This is published in full elsewhere in this issue.

Prof. W. W. Haldane Gee, of Manchester, delivered a lecture, illustrated by experiments and lantern slides, on "The Electrolytic Methods for Preventing the Corrosion of Metals."

He showed that the corrosion of metals can be lessened or prevented in two ways: (1) By connecting the metal to be protected to a more electropositive metal, so that a primary cell is produced; (2) by making the metal to be protected the cathode in an electrolytic cell supplied by an external electrical pressure. Various types of primary cell arrangements that could be employed in practice were classified. The efficiency of the cell for protection will depend on the current density at the cathode, and this will be controlled by the resistance of the cell and the effective voltage. The importance of overvoltage in determining the effective voltage was discussed.

The history of Sir Humphry Davy's application of zinc and iron protection for the prevention of the corrosion of the copper sheathing of ships and subsequent inventions for the protection of condensers and pipes were detailed. The patents of

¹See this journal, November, 1912, p. 17.

Harris and Anderson, in which aluminium alloys are used for the prevention of the corrosion of condenser tubes, were shown to be primary cell methods. In the case of the use of zinc in boilers was involved a knowledge of the electrolytic resistance of the boiler waters and the effective voltage at temperatures from 150-200 deg. C., concerning which there is great need of experimental data. The amount of zinc used in some marine boilers is as great as from 400-600 lb. of rolled zinc per annum. If the zinc is efficient in producing electrical currents then the average current may be from 17 to 25 amperes. It was obvious that such currents would be obtained much more economically by the use of a dynamo.

The direct use of electrical currents has been the basis of a number of patents. Those of Mr. Elliott Cumberland were especially described. Iron anodes are placed in the water of the boiler, which latter is made the cathode. A low voltage of supply provided by a small motor-generator is used. The method has proved effective not only in preventing corrosion, but also in removing scale from the heating surface and preventing its formation. Experiments carried out at the Manchester School of Technology have shown that the current densities necessary to protect iron, copper, and other metals from corrosion of fresh and salt water are of low value, and hence in the cases of boilers and condensers the annual cost of electrical energy required is a small item. The chief cost will be in the renewal of anodes. Harris and Anderson have also applied electrical currents for the prevention of the corrosion of condensers. They find that a condenser with a cooling surface of 1025 sq. ft. requires only 2 volts and 2 amperes, and the special anodes used by them cost from £3 5s. per 1000 sq. ft. per annum. The use of electrical currents may also be applied in chemical works to prevent the corrosion of metallic screens and vessels from acid liquids.

Mr. J. T. Crabtree read a paper on "The Nature of Overvoltage." This may be regarded either as the excess of the anodic or cathodic decomposition voltage of a dilute acid with a given electrode over that for platinized-platinum, or as the excess of the back e.m.f. set up as an electrode (anode or cathode) after polarization over that set up by a platinized platinum plate under identical conditions.

This back e.m.f. was determined by alternately polarizing an electrode and measuring its single potential difference by means of a potentiometer, alternate connection between the latter and the primary circuit being made by means of a rotating commutator.

Overvoltage is affected by the time of application and current density of the polarizing current, the nature of the electrolyte and electrode employed, and by the thickness of the latter. Since these factors also affect the back e.m.f. of platinum, a comparison electrode was employed consisting of a thin deposit of platinum on Jena glass, the back e.m.f. of which is constant with time.

The overvoltage of a metal was taken as the difference between the back e.m.f. set up after an application of the polarizing current for thirty minutes at a given current density, in $\frac{N}{10}$ acid, and that set up by the above electrode under similar conditions.

It is probable that overvoltage is simply a manifestation of the difference between the rate of production of ions at the electrode and of the combination of these form complexes. Probably after being discharged the ions pass into a metastable condition and give rise to a back e.m.f. The catalytic activity of the electrode is probably an important factor as affecting this velocity of reaction, and the difference in catalytic activity of various metals under different conditions of texture, temperature, etc., would explain why the overvoltage varies largely with different metals under different conditions.

Overvoltage may be of importance in the case of the corrosion of a metal as either:

(a) Having a tendency to retard the deposition of hydrogen or oxygen on its surface, which might tend to assist or prevent corrosion.

(b) By setting up a high back e.m.f. which would diminish the effect of a decomposing current in cases of corrosion due to electrolysis.

(c) By assisting or preventing the solution of a metal. The condition that a metal may dissolve in an acid and evolve hydrogen is given by the relation

$$\text{e.m.f. of metal} - \text{back e.m.f. of hydrogen} > \eta$$

where η = overvoltage of metal. If η is large, no solution of the metal can occur. This is apparent with galvanized iron, where, owing to the overvoltage of the zinc, very little solution of iron takes place, the latter assuming the overvoltage of the external metal.

In view of the absence of any suitable factor to indicate the tendency of a metal to corrode, future experiments may indicate a parallelism between the overvoltage of a metal and its corrosion factor.

Dr. W. Rosenhain read a paper entitled "Note on a Specimen of Ancient Iron from Ceylon."

The reading of the paper was followed by a long discussion. A number of specimens of corroded metals was exhibited.

Electric Steel in Germany

In *Elektrotechnische Zeitschrift*, April 3, 1913, an interesting review of the present status of electrometallurgy in Germany is given by VICTOR ENGELHARDT, the chief engineer of the electrochemical department of the Siemens & Halske Company. As mentioned before in these columns, the large 25-ton Heroult furnaces have been put in operation on the Deutscher Kaiser. The largest size of induction furnace is at present 13 tons. As a result of the competition between induction furnaces and arc furnaces, the opinion seems now to have been firmly established that both types of furnaces have a right of existence. For this reason the Gesellschaft für Elektrostahl Anlagen, which is affiliated with the Siemens & Halske interests and which owns the principal patents for induction furnaces, has now also acquired the rights for the Girod arc furnace for Germany. Otherwise, no change has come in the exploitation of the different furnace types by the large electrical manufacturing companies, as the Heroult-Lindenberg electric furnace interests are affiliated with the Allgem. Elek. Ges. and the Nathasius furnace interests with the Bergmann concern. The use of electric furnaces for melting ferromanganese has been widely introduced since the advantages are evident in this case. The cost of operation can be reduced so as to show a clear saving without taking into consideration the technical advantages which are doubtless in existence, but cannot be easily stated in exact figures. In the thirty large German steel works which use large quantities of ferromanganese in their converter plants, twelve had electric furnace plants for melting manganese in operation or course of erection at the end of 1912. Five of these plants were erected by the induction furnace-Girod interests, three use Heroult furnaces, two Keller furnaces, and two Nathasius furnaces. As to the use of the electric furnace for copper and zinc smelting, Mr. Engelhardt points out that the electric furnace has to meet in this field not only competition from the old-established metallurgical processes but also from leaching processes with electrolysis. While ten or fifteen years ago the potentialities of wet electrolytic processes were overrated, there is at present a tendency to pay new attention to their possibilities.

Gold and Silver

Cost of Dredging.—In the *Mining Magazine*, March, 1913, Mr. C. W. PURINGTON gives, among other data on the Seward peninsula, Alaska, the cost of dredging at the Plein dredge, on Otter creek, tributary to the Nome river. At this dredge the ground is thawed by means of steam-points. The equipment for this purpose consists of two boilers, totaling 80 hp, mounted on skids covered with tents. They are moved ahead as fast as the dredge works the ground. Forty steam-points are used to thaw the ground to a depth of 14 ft. The duty of each point is 16 cu. yd. in 24 hours, and the cost is under 13 cents per yard. The cost of power generated, with oil costing 7.27 cents

per gallon delivered at the plant, is 3 cents per horsepower-hour.

The total cost of dredging and thawing at this plant is said to be 27 cents per cu. yd., handling about 800 yards per day. The cost at this plant, however, is not representative of what can be done on a larger scale. In the Nome tundra area, it is not impossible that, where a sufficiently large yardage is available and a proper equipment installed, the entire cost of stripping, steam-thawing and dredging will be under 20 cents per cu. yd.

Coarse Crushing in Cyanide Solution.—Some gold ores are readily amenable to cyanidation, even when crushed as coarse as $\frac{1}{2}$ in., and when such is the case, it is possible to mill low-grade ore at a profit, even though the percentage extraction is not as high as that of some mills in which fine grinding is practiced. Two good examples of this class of ore, and of the improvement in milling made by changing from fine to coarse crushing, are given in the *Mexican Mining Journal*, March, 1913, by Mr. ERNEST H. SIMONDS.

The first ore is described as very porous, oxidized, and of medium milling grade. It would absorb and readily retain 15 per cent moisture. It was being crushed in stamps to 20-mesh and finer, the stamp duty being about 4 tons per day. About 55 per cent of the stamp product was coarse enough to be leached and the balance was agitated and filtered. Laboratory tests indicated that good extraction could be made with much coarser ore. One of the batteries was then fitted with a 4-mesh screen, and a lot of ore crushed. The stamp duty rose to 7.8 tons per 24 hours, and 76 per cent of the product could be leached. The extraction was as good as when ore was crushed to 20-mesh; the capacity of the mill was almost doubled by making an addition to the number of leaching tanks; and the slime department needed no enlarging.

The second ore was a medium hard quartz containing about 6 per cent of coarsely disseminated pyrite. Little slime was formed when the ore was crushed. Little concentrate could be obtained by crushing to 4-mesh, but at 8-mesh it was possible to secure a concentrate containing 73.5 per cent of the gold in the ore. At 12-mesh the concentrate yielded only 70 per cent of the gold in the ore. The final mill treatment was as follows: Crush to 8-mesh in cyanide solution, size the pulp on a 30 or 40-mesh screen and concentrate the oversize on jigs and the undersize on tables. All coarse and fine sand tailings are washed to recover cyanide solution, and then discharged to waste. The slime is thickened, agitated and filtered. Coarse crushing is expected to give a stamp duty of 7 tons if no re-grinding apparatus is used, and more than 10 tons with regrinders. The test data show that the jigs will handle more than two-thirds of the ore; that the screen undersize, or sand size, will amount to less than one-quarter of the ore and contain more than one-half of the concentrate in it; that the same recovery can be made by coarse crushing and concentrating as by sliming all the ore to 150-mesh and cyaniding it without concentration.

The New Shamva Mill, Rhodesia.—The following outline of the equipment and process to be employed in the new Shamva mill in Rhodesia is interesting as showing the trend of milling in that country. It is taken from the *South African Mining Journal*. The ore is to be sized before going to the stamps, all $\frac{1}{4}$ -in. particles being passed directly to the tube mills. The larger sizes will be crushed in cyanide solution by fifty-six 2000-lb. Nissen stamps fitted with $\frac{1}{4}$ -in. mesh screens. Classifying cones will be placed ahead of the tube mills, and amalgamation plates and blankets after the tubes. The tube mill product is returned to the cones. The overflow of the tube mill cones will pass to eight sand cones superimposed above four others, where the sand will be classified and sent to leaching tanks. The slime overflowing the last cones will be thickened in 35 ft. x 10 ft. Dorr thickeners and agitated in Pachuca tanks, 45 ft. x 10 ft., connected in series. A Butters filter of 336 leaves will remove the valuable solution from the slime. Precipitation will be effected in ordinary zinc boxes, and the gold slime melted in gas-fired furnaces.

Copper

Historical Note on Gas-Fired Reverberatories.—The application of producer gas in firing reverberatory copper smelting furnaces is regarded as a recent development, although it is true that certain metallurgists foresaw some years ago the value of the method and predicted its extended use. It appears, further, from an article on mining in the Argentine Republic, by Mr. W. EVAN SIMPSON, in *Mining Magazine*, March, 1913, that one Fournet, "an unknown but progressive metallurgist," actually used gas-firing on a small reverberatory furnace at a place called Corrales, more than twenty years ago. "The plant consisted of a water-sealed producer, a reverberatory capable of holding a charge of about one ton, and a brick stove built on the best regenerative principles." A furnace charge required about six hours for reduction. The ore contained from 7 to 12 per cent copper, and the slags assayed only 0.5 per cent copper. The only fuel available for the gas producer was wild mountain bush, locally known as *jarilla*, which, though very resinous, is so small that difficulty is experienced in finding a stick of it larger than 1 in. in diameter. The apparent success of this venture in the wilds of South America, is worthy of the consideration in similar inaccessible localities.

Grinding-Pan Practice

Notes from the Great Fingall, Australia.—In the monthly *Journal* of the Chamber of Mines of West Australia, Mr. E. JENSEN, metallurgist for the Great Fingall company, presents some notes on the efficiency of the grinding pan, and of the relative merits of flat and corrugated shoes and dies. The latter first came to his notice about four years ago when he was metallurgist at the Oroya Black Range, through Mr. W. Jordan who was then mechanical engineer at the same mine. The first trial indicated a greater grinding efficiency, owing to the increased grinding surface offered by the corrugated surfaces. The corrugations are spaced $1\frac{1}{2}$ -in. centers, and take the form of concentric rings on the whole set of shoes and dies, each shoe and die being cast with a proportionate section of the concentric corrugations.

Mr. Jensen gives a number of tables showing the daily efficiency of the pans during the life of a set of shoes and dies. The life of a set of flat-surface shoes and dies appears to be about 21 days, while with corrugated surfaces the same sets lasted for 37 or 38 days. In calculating the grading analyses and grinding efficiencies in these tests, Mr. Jensen has used Stadler's method,¹ of which the following are the essential points. The mechanical work necessary for reducing the weight of a unit from one grade to the next following, is a constant for each grade, and is called the energy unit, denoted by E. U. The useful work done per unit by any crushing machine is the difference between the mechanical values of samples taken from the inlet and discharge of the machine. The relative mechanical efficiency is the value obtained by dividing the total work done by the horse-power consumed. The following table is a summary of the different tests made according to Stadler's formula:

	Test Numbers			
	1	2	3	4
Fan speed, r.p.m.	54	54	54	54
Shoe and die surface	flat	flat	corr	corr.
Pan feed, E. U.	20.50	20.54	20.77	20.84
Pan discharge, E. U.	23.21	23.38	23.80	23.48
Work done, E. U.	2.62	2.84	3.03	2.64
Total E. U., 24 hours	60.82	107.02	118.78	111.41
Relative mechanical efficiency	10.51	11.36	12.49	11.73
Consumption iron per ton ore, lb.	3.75	3.56	3.64	2.88
Tons water per ton sand	5.33	5.33	5.50	5.10

"It appears that better results can be expected by having an increased grinding surface, as per medium of corrugated shoes and dies. Another factor worthy of consideration is pan speed. We are now undoubtedly getting, with a pan speed of 54 r.p.m., better results, with a lower power consumption per pan, than those obtained when the pans were running at 58 to 60 r.p.m."

¹Grading Analyses and Their Application. Stadler. *Trans. Inst. Min. & Met.*, Vol. XIX, pp. 471-485.

Recent Chemical and Metallurgical Patents

Gold and Silver

Treatment of Precious-Metal Ores Containing Copper.—Under certain conditions the presence of copper in gold and silver ores renders such ores not amenable to the cyanide process. According to the specifications of a patent recently granted to Mr. JOHN COLLINS CLANCY, of New York City, the use of a cyanamide in the cyanide solution overcomes the usual difficulties and makes it possible to treat successfully gold and silver ores by cyanidation. Mr. Collins has discovered that the addition of a cyanamide prevents the copper from consuming the cyanide, that is to say, the cyanamide forms an insoluble compound with the copper which is then soluble in excess of cyanide solution without destroying it. By precipitating the copper from the solution the cyanide can be regenerated and used as a solvent.

Mr. Clancy's discovery further includes the fact that copper thus dissolved in a cyanide solution containing cyanamide, is not precipitated on metallic zinc, while the precious metals are thus deposited from the same solution.

The assumed reaction between copper cyanamide and cyanine is as follows: $n\text{CuCN}_2 + n\text{KCN} = (n\text{CuCN}, n\text{KCN})$. The last complex compound is a solvent for precious metals and copper compounds. By simple electrolysis, it may be caused to yield regenerated cyanide.

As an example of carrying the invention into effect, the following figures are given by the inventor: 2000 lb. pulverized ore is suspended in 2000 lb. water containing 2 lb. sodium cyanide, 2 lb. calcium cyanamide and 2 lb. lime. This mixture is agitated for 8 or 12 hours, or until extraction is complete, after which the solution is displaced from the solid matter, and the precious metals therein contained are precipitated on zinc, without deposition of copper. The solution may be used over and over again until its copper content is such as to warrant removal of the copper by electrolysis. (1,057,936, April 1, 1913.)

Method of Treating Ores.—In Fig. 1 is shown a representation of an apparatus recently patented by NIELS C. CHRISTENSEN, JR., of Salt Lake City, Utah, for the treatment of gold and silver ores. The apparatus comprises a novel form of roaster in which oxidizing, reducing, chloridizing or sulphatizing processes can be carried on. The device consists of a shaft 1 having a suitable lining 3. Two floors or grates 5 and 6 are composed of staggered members 10 and 11, between which a reciprocating member 12, 13 can move for the purpose of delivering the ore continuously from one compartment to the next. The shaft is thus divided into compartments in which different conditions of temperature and chemical action can be maintained, and the ore successively subjected to these conditions. Ports 4 and 14 below the upper grate, and 7 and 15 below the lower grate, provide for the introduction of air, fuel, steam or other gas, to cause the proper reaction with the ore and added sodium chloride or sulphate or other salt.

The inventor outlines various combinations that can be made in treating ore in this furnace, and gives results that he

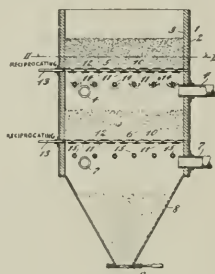


FIG. 1.—APPARATUS FOR TREATING ORES

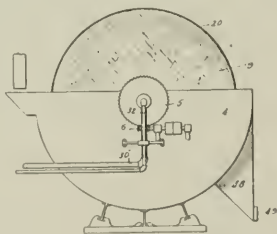


FIG. 2.—VACUUM FILTER

has obtained on ores from Colorado and Utah districts. "Practical tests have demonstrated the success and efficiency of this process of heating ores in stages at temperatures and for periods under positive control of the operator, and in atmospheres adapted to the particular kinds of ores treated." (1,058,034, April 8, 1913.)

Vacuum Filter.—A new form of continuous vacuum filter for ore slime is shown in Figs. 2 and 3, being the invention of Mr. GEORGE J. YOUNG, of Reno, Nev. The device operates on the same general principles as other vacuum filters, viz., submerging a filter surface in slime pulp, withdrawing the valuable solution through the filter fabric, and building up a cake of slime on the fabric. These operations are performed successively and continuously on a plurality of disc-like filter units mounted on a hollow rotatable shaft, within which is a second hollow shaft which is stationary and which serves as a suction chamber, being connected with a vacuum pump. Suitable ports connect the different cells of a disc with source of reduced or excess pressure, so that vacuum and pressure can be applied alternately at the proper times.

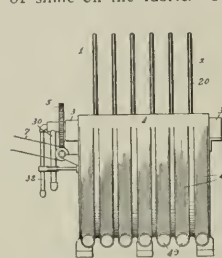


FIG. 3.—VACUUM FILTER

The apparatus consists of a slime tank 4, within which are mounted the filter discs 19. The latter are rotated by means of a gear drive as shown. The valves in the hub are so arranged that about seven-eighths of the disc is under reduced pressure, while one-eighth is subjected to excess pressure at the time of discharging the cake. The cake can be washed prior to discharge by means of sprays from pipes suitably arranged. Scrapers are used to divert the discharged cake into chutes 48, from which the slime is withdrawn through doors 49. (1,057,475, April 1, 1913.)

Agitator.—A modification of the Pachuca tank system of agitating ore slime has been patented by A. W. WARWICK, of Colorado Springs, Colo. One form of his invention is shown in Fig. 4. It consists of a cylindrical tank *a* with a conical bottom *a'*, in the center of which is supported an air-lift pipe *b*, supported at *b*, and capped at the top by *b'*. Rows of holes *b''* are provided for the exit of the lifted slime. A source of compressed air is shown at *c'*, connected with the nozzle *C* under the lift pipe, and with secondary air injectors *d'* arranged near the bottom of the conical bottom. The assumed level of material in the tank is the line *N*.

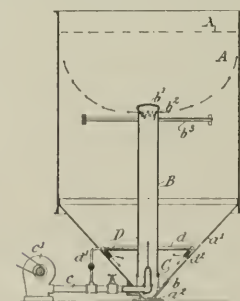


FIG. 4.—AGITATOR

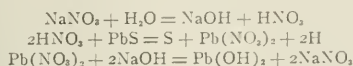
In operation the compressed air lifts the slime through the central pipe, from which it issues through the holes in a lateral direction. The rising air serves to aerate the slime above the top of the lift-pipe, while the heavier solids fall to the bottom for further agitation.

The auxiliary air injectors, near the bottom keep the slime there well aerated and agitated, and serve also to break up any cakes that may form during suspension of operations. (1,054,629, Feb. 25, 1913.)

Lead, Copper and Zinc

Electrolytic Production of Lead Hydrate.—According to a patent issued to THEODORE G. TIMBY, of Chicago, Ill., lead ore may be fused and cast into bars or cakes of convenient form, and used as anodes in an electrolytic cell containing as electrolyte a neutral solution of sodium nitrate in water, producing under electrolysis lead hydrate. The lead ore is fused in a reducing atmosphere and cast into bars about 18 in. long,

12 in. wide and 2 in. thick. One or more such anodes are immersed in the electrolyte, with suitable cathodes, and electrolyzed. The following reactions show the cycle of operations.



With a potential between one and six volts the current separated some free sulphur at the anode, forming a coating which can be brushed off, or which can be retained in the diaphragm when the latter is used. (1,056,382, March 18, 1913.)

Electrolysis of Argentiferous Lead Ores.—By ordinary smelting methods, silver-lead ore is reduced to silver-lead bullion which must be refined for the recovery of silver and the production of soft lead. It is proposed by THEODORE G. TIMBY, of Chicago, Ill., to fuse such ore and cast it into blocks that can be used as anodes in an electrolytic cell. The electrolyte is a hydrous solution of sodium nitrate, slightly acid with nitric or acetic acid to prevent the precipitation of lead hydrate. When a cell is thus equipped and electric current applied in the usual manner, silver and lead are liberated as a result of electrolysis. The lead may be collected upon the cathode in metallic form, or precipitated as, for example, lead chromate. The silver may be collected upon a filter cloth surrounding the anode, and into which is slowly fed a solution of alkaline halide, such as sodium chloride, which precipitates the silver. If no filter cloth is used, the precipitant may be fed into the bath between the electrodes and the deposited silver salt removed from the bottom of the cell. Some lead halide may temporarily precipitate, but will again dissolve and the lead will be deposited on the cathode. (1,056,383, March 18, 1913.)

Fusing and Purifying Copper.—A furnace and process for fusing cathode or electrolytic copper, and maintaining its purity during fusion, have been patented by MESSRS. WALTER S. ROCKEY and HILLIARY ELDRIDGE, of New York City. The object of the invention is to expose the metal to a reducing flame in the presence of the products of combustion, protecting the metal from oxidation after fusion by an atmosphere of the products of combustion, also while the metal is flowing from the fusion chamber to the ladle and from the ladle to the casting molds.

It has been found that electrolytic copper of high purity is contaminated by oxygen in the fusing process, forming oxides such as CuO and Cu_2O . By the process of the patentees these impurities are eliminated. (1,057,882-3, April 1, 1913.)

Electrolytic Recovery of Zinc and Manganese.—According to a process of ANDREW G. FRENCH, of Nelson, B. C., for the treatment of complex lead-zinc ores, the inventor obtains as a final result a strong aqueous solution of sulphate of zinc with neutral sulphate of sodium and more or less sulphate of manganese. This process is described in a former patent No. 1,041,060. More recent work by the same investigator has shown him that these three salts, viz., sulphates of zinc, sodium and manganese, are easily electrolyzed with the production of pure metallic zinc on the cathode and pure manganese dioxide on the anode.

The success of the process depends on the favorable combination of these three salts, and when insufficient manganese is present this element should be added.

The solution of the three sulphates is treated in an electrolytic tank having lead anodes and thin sheet zinc cathodes. Direct current is used, of from 2½ to 4½ volts with a current density of 20 to 30 amp. per square foot of cathode surface. Under these conditions pure zinc and manganese dioxide are formed. In the practice of the process, the solution flows continuously from the ore leaching tanks to the electrolytic tank, and after electrolysis and the removal of a portion of the metals, back to the leaching tanks.

The solution flowing into the electrolytic tank is practically neutral, while the electrolyzed solution is slightly acid and suitable for further reaction on ore. (1,055,157-8, March 4, 1913.)

Vanadium

Extraction of Vanadium from Carnotite.—SIEGFRIED FISCHER, of Golden, Colo., has recently patented a process for removing vanadium from carnotite concentrates or rich carnotite ores, leaving a residue containing all the uranium, radium and a negligible quantity of vanadium. The process consists in mixing the carnotite with caustic alkali in the ratio of about 5:1, adding water to form a plastic mass, and then evaporating and possibly re-evaporating until the reaction is complete, forming sodium vanadate. The mass is then lixiviated, the vanadium solution drawn off and made slightly acid with sulphuric acid, and the vanadium precipitated as iron vanadate by means of ferrous sulphate. This product, after drying can be reduced to ferrovanadium by the electric furnace or by the Goldschmidt process. Mr. Fischer's process was fully described by him in an article in this journal, August, 1912, page 469. (1,054,102, Feb. 25, 1913.)

Metal Alloys

Alloys for Cutting Tools.—In two patents granted to ELWOOD HAYNES, of Kokomo, Ind., the inventor describes new alloys which are valuable for forming cutting implements in which hardness, toughness and elasticity are demanded. The alloys also are capable of taking a high polish and receiving and retaining a sharp cutting edge. These alloys are formed from cobalt, chromium and one or more of the elements of the chromium group, such as tungsten, molybdenum and uranium. An example of a ternary alloy is one containing 15 per cent or less chromium, 15 per cent or less tungsten, and cobalt. Numerous other formulas are given in the patent specifications, for alloys having special application. (1,057,423-828, April 1, 1913.)

Government Specifications for the Purchase of Coal

Large users of coal will be interested in Bureau of Mines Bulletin No. 63, "Sampling of Coal Deliveries, and Types of Government Specifications for the Purchase of Coal," which has just been issued. The Federal Government, which purchases \$8,000,000 worth of coal annually, buys more than half of it under specifications and has gone deeply into the question of sampling and analyzing coal. George S. Pope, the engineer in charge of such investigations and the author of the bulletin, makes the following statements:

To determine with utmost accuracy the ash content and heating value of a quantity of delivered coal would require the burning of the entire quantity, and special apparatus arranged to measure the total heat liberated, or would require crushing the whole quantity, and reducing it by an elaborate scheme of successive crushings, mixings, and fractional selections to portions weighing approximately 1 gram, the minute quantity which the chemist requires for each determination. Either of these procedures is obviously impracticable if the coal is to be used for the production of heat and power.

The method actually employed is to select portions from all parts of a consignment or delivery of coal and to systematically reduce the gross sample, obtained by mixing these portions, to quantities that the chemist requires for making ash determinations or that can be burned conveniently in the calorimeter, an apparatus for determining the heating value. The gross sample should be so large that the chance admixture of pieces of slate, bone coal, pyrite, or other impurities in an otherwise representative sample will affect but slightly the final results. Increasing the size of the gross sample tends toward accuracy but the possible increase is limited by the cost of collection and reduction. In reducing the gross sample by successive crushings and halvings or fractional selections, the object is to procure a small laboratory sample that, upon analysis, will give approximately the same results as the gross sample itself, or, in fact, the entire quantity of coal from which the gross sample was obtained.

Recognizing the importance of the method of sampling as being a definite commercial procedure and of having the method clearly set forth in the specifications to become a part of the

contract, and recognizing also the desirability of insuring uniformity and similarity in the specifications used by the different branches of the federal service for the purchase of coal, representatives of the executive departments and independent establishments of the government held a conference under the auspices of the Bureau of Mines in February, 1912, for the purpose of discussing these and other features of the specifications. At this conference committees were appointed to prepare specifications in accordance with the views of the members. It was recognized at the conference that in general specifications, such as were recommended, certain requirements had to be of wide application, as the specifications cover such a wide variety of conditions, not only as to character and quality of coal but as to type of furnace equipment, size of deliveries, methods of delivering, etc.

The specifications which were used for the purchase of coal on the heat-unit basis prior to the fiscal year 1912-13 were on the B.t.u. (British thermal unit) "as received" basis; that is, payment for delivered coal was directly affected by the moisture content of the sample received by the laboratory. This method was based on the assumption that the moisture in the samples collected at the time of weighing and delivery could be preserved with slight loss during the storing and subsequent working down of the gross sample to a quantity convenient for transmittal to the laboratory and in its later treatment in the laboratory. From experiments that have been made and from a large mass of data, it is known that the moisture content of coal does not remain constant, and that the moisture content reported by the laboratory may be as much as a 5 to 10 per cent lower than that actually contained in excessively wet or high-moisture coal at the time of weighing.

* * * As a sample loses moisture, its B.t.u. "as received" value correspondingly rises, with the result that the price for delivered coal determined on the "as received" value is, with rare exceptions, higher than that warranted by the quality of the coal at the time of weighing. As a general statement, payment based on the "as received" B.t.u. value will be higher than warranted, unless the sampling and laboratory work can be carried on under conditions that minimize moisture loss, as under freezing temperatures.

Recognizing the uncertainty involved in taking the moisture determination in the laboratory as representative of the moisture content of the delivered coal and the consequent possibility of payment of a higher price than is warranted, the Bureau of Mines recommended to the executive departments and independent establishments of the federal service that the heating value in the coal specifications for the fiscal year 1912-13 be on the "dry coal" basis.

In preparing these specifications the fact was recognized that the amount of moisture contained in coal produced from day to day from the same mine, or group of mines working the same bed, is largely accidental, and is a matter over which the buyer and seller have only slight control. However, in order to place a negative value on high-moisture coals and to protect the government against the delivery of coals containing excessive amounts of moisture, the specifications require the bidders to specify the maximum moisture content in coal offered. This value becomes the standard of the contract.

If coal of uniform B.t.u. "dry coal" value is delivered on a contract, the contractor receives the advantage on any delivery in which the moisture content approaches the maximum specified, because he is paid for the weight of water contained in the coal in excess of a normal amount, whereas if the coal is very dry, containing less than the normal amount of moisture, the purchaser receives the advantage.

* * * As an example of the effect of a heavy rain on a car of coal in transit, a precipitation of 3 in. of water on a loaded 50-ton car, area of top about 360 sq. ft., would increase the weight of the coal 5.01 per cent, provided none of the water drained out or evaporated. It is obvious that if this coal is weighed and delivered immediately, special samples for moisture determinations should be collected and prepared at once and sent to the laboratory as a basis for equitable adjustment

of payment on account of the excessive amount of water in the coal. As the weight of the coal was increased by the excess water, there should be a corresponding decrease in the price to be paid.

If a railroad car or wagon so rained on should not be unloaded immediately after weighing and special moisture samples should not be properly collected, prepared, and sent hermetically sealed to the laboratory, it is obvious that the purchaser would pay a higher price than warranted, especially if the car or wagon stood for some time before sampling and some of the water drained out. Further, if the coal was not immediately unloaded and sampled or if the car continued in transit after weighing, then the coal at the top would soon dry; and in either case the effect of the 3-in. rainfall, as indicated by the analysis, might be only a fractional percentage of the moisture contained in the coal at the time of weighing.

The determination of the moisture of coal delivered from stock piles is often of great importance, for the proportion of moisture contained in the small sizes, which are most abundant near the center of a stock pile and which absorb the rains, and melting snows in districts of heavy snows, may be from 10 to 15 per cent higher than when stocked. It is apparent, therefore, that special moisture sample determinations are necessary for equitable adjustment of payment on amount of excessive moisture in coal stocked in piles exposed to the weather.

The specifications provide for the collection of "special moisture samples" if, in the opinion of the government officials sampling it, the delivery contains moisture in excess of that guaranteed by the contractor. The "special moisture samples" are prepared in a manner to minimize moisture losses and may be taken and prepared independently of the gross samples collected for the determinations of heating value (B.t.u.), ash, and other specified data. If the analysis of the special sample shows a moisture content in excess of the contractor's guaranty, a proportionate deduction is made from the price to be paid for the coal.

The Joseph Dixon Crucible Company, of Jersey City, N. J., have issued a very complete "production catalog" of over 100 pages of type and illustrations, giving a full description of the entire Dixon graphite line. This catalog should be very effective in acquainting those who are already users of one form of graphite with its many other forms and uses.

Buffing and Grinding Motors.—The Charles J. Bogue Electric Company of New York City have placed on the market a new design of direct-current buffing and grinding motors in five sizes from $\frac{3}{4}$ to 5 hp, for connection to 110-volt and 220-volt circuits.

New Pulp Mill Project.—The Chesapeake Pulp & Paper Company, of West Point, Va., it is said, will have the first sulphate pulp mill in the United States built entirely by an American engineer with modern American methods of design and construction. This mill is now being built by Mr. George F. Hardy, New York; while Mr. Philip B. Sadtler, assistant general manager of the Swenson Evaporator Company, Chicago, has secured the contract for the entire machinery layout in the recovery end of the plant.

Zinc Dust Precipitation.—Experience with this form of precipitating cyanide solution at the Brakpan mill, South Africa, is related in *Bulletin* 101, Inst. Min & Met., by C. B. Brodigan. The equipment consisted of three Merrill presses with belt feeders. The latter caused trouble at first on account of irregular feeding, but this was overcome by alterations. Clear solution also was found essential, as slightly turbid solution choked the cloths, causing a rapid rise in pressure in the press, and necessitating by-passing and resting of the press. Strength of solution had to be higher than with zinc-box precipitation. There was a feeling of some anxiety about the reliability of precipitation, which led to the use of continuous testing. In spite of these troubles, which were largely incidental to starting a new process, it was generally believed that the press system offered advantages over the use of boxes.

Oil Extracting and Washing Centrifugal

It has so long been the custom of many plants and manufacturing establishments to regard the burning or discarding of their oily waste and other machinery-wiping material as a matter of course that they have ceased to consider what such a practice actually cost them, or, more properly speaking, perhaps have considered such expense as unavoidable.

The consumption of waste, towels and other wiping mate-



FIG. 1.—CENTRIFUGAL STEAM-TURBINE DRIVEN COMBINED OIL-EXTRACTING, WASHING AND DRYING MACHINE.

rials by every plant or manufacturing establishment is, of course, subject to the amount and character of its power and industrial equipment. However, when reckoned by the year it is by no means the least of the items of operating expense.

When to this is added the value of the oil and grease that is lost in the discarded waste and other machinery-wiping material, the figure approaches a total that would surprise those who have not given the matter serious consideration.

This condition presents great opportunity for a successful process of waste and oil reclaiming, and where machinery-wiping towels are used a satisfactory method of cleaning them. Naturally such a promising field has not been neglected and several methods have been employed to reduce the loss of waste and oil, beginning with drip pans, presses, chemicals, etc., to more modern and efficient machines with increasing success.

The accompanying illustrations show the most recent and highly efficient and successful machine that has been devised for this purpose by the D'Olier Centrifugal Pump & Machine Company, of Philadelphia.

The machine is a combined oil-extracting and washing centrifugal, steam-turbine driven and manufactured in sizes 10-in., 15-in., 20-in., and 36-in. baskets.

The basket, which is entirely enclosed within the curb of the machine, is directly connected to the steam-turbine wheel. The following description of the operations of the machine can be readily understood by reference to the accompanying line drawing. The waste, towels or other machinery-wiping material to be reclaimed is placed in the basket and the steam turned on, rotating the basket at high speed. The small arrow heads indicate the course of the steam from the point of entrance (nozzle), through the turbine wheel into the bottom of

the basket, thence through the waste, acting thereon, and out through the exhaust.

The effects of the steam on the waste, or other contents of the basket, are to heat and open up same, thinning the oil and grease, which in turn are thrown off through the perforations in the sides of the basket to the inside of the curb of the machine and are then conveyed by gravity through the oil trap to precipitate any solids and discharged into an oil receptacle. Heavy arrow heads indicate the course of the oil from the contents of the basket.

The waste or towels having been freed of oil and grease are then ready for cleaning; the water tap is opened and the basket and its contents submerged. Special washing powder is placed in the bath and by means of a steam jet the water is boiled while the basket is slowly revolved, thoroughly and continually agitating the contents. After the washing process, all free water is drained from the machine and the basket run at high speed to extract remaining moisture. The waste or towels are then ready for air-drying and further use. This reclaiming process can be repeated many times at a great saving of machinery-wiping materials.

The oil-reclaiming process will save from 90 to 95 per cent of the oil and grease in the soiled material. The complete process is thus one of dual economy in the recovery of oil and reclaiming of waste and towels. The economy thus effected can be readily appreciated.

This machine, by the use of an interchangeable basket, furnished by the manufacturers, can also be successfully used for the recovery of oil from automatic machined parts, metal chips and turnings, scrap metal, etc. Also, for general laundry purposes and like economic uses, and further, supplies a long-felt want in many laboratories for an efficient hand or power operated centrifugal for testing or experimental work.

The entire machine bears evidence of thorough and efficient design and construction. The manufacturers lay particular stress upon the enforced steam circulation whereby the steam acts on the entire contents of the basket; also upon the design and position of the main bearing, located above and not below the water level. This machine is also equipped for electric motor or belt drive for special service.

The D'Olier line includes a modern and highly efficient machine of the centrifugal type for the recovery of oil and grease from metal chips, turnings, scrap metal, etc., electric motor and belt driven. It can be fitted with an interchangeable

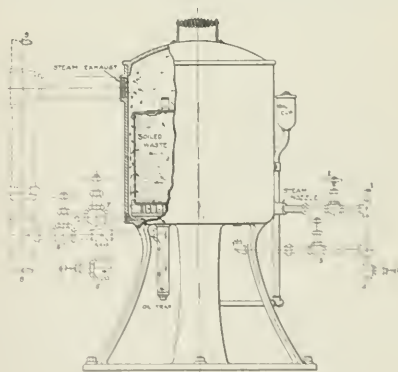


FIG. 2.—CROSS-SECTION OF CENTRIFUGAL

basket for extracting oil from machinery wiping materials for installations where steam is not available.

The machine is of the "Weston" self-balancing type, which is a decided advance in machine construction for this service.

The D'Olier Centrifugal Pump & Machine Company is continuing and enlarging its well-known line of centrifugal machines and pumps, manufactured for many years by D'Olier

Engineering Company, and is prepared to furnish centrifugal machines for every possible purpose, ranging from 10-in. to 72-in. baskets of varied design and for steam, water, electric and belt drive.

New Float-Type Recording Differential Pressure Gauge

In the April issue (page 222) a description was given of the spring pressure tube designs of recording differential pressure gauges made by The Bristol Company, of Waterbury, Conn. This company is now putting on the market another invention of its president and founder, Prof. Wm. H. Bristol, as illustrated in Fig. 1.

This new float type differential recorder was patented



FIG. 1.—FLOAT-TYPE RECORDING DIFFERENTIAL PRESSURE GAUGE

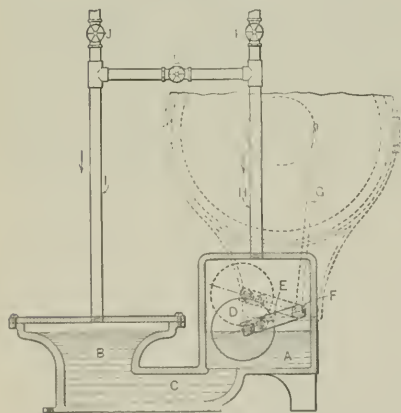


FIG. 2.—SECTION OF FLOAT-TYPE RECORDING DIFFERENTIAL PRESSURE GAUGE

August 15, 1911, and has been tested under practical operating conditions in steel works. It was developed to meet requirements for which the spring pressure types of differential recorders are not suitable, that is, for very low ranges of differential pressure in air, gases, liquids or steam, for applications where the static and differential pressures fluctuate rapidly, and also to satisfy the demand for instruments to record the rate of flow or volume directly on charts having uniform graduations in units of flow or volume.

The construction and principle of operation of the instrument will be readily understood by referring to the sectional diagram Fig. 2. There are two pressure chambers *A* and *B* in communicating through the connection *C*. A cylindrical float *D* is located in the pressure casing *A* as indicated, and

is connected by arms to the shaft *F*, which extends through the casing. The recording arm *G* is directly connected to the end of the shaft *F*.

Connections are made by means of the pipes *H* and *I* between the pressure chambers *A* and *B* and the two pressures whose difference it is required to record. Mercury or water are employed in the pressure chambers according as the differential range to be recorded is high or low. When the higher pressure is applied to the chamber *B* through the pipe *I*, the level of the liquid in this chamber is lowered and that in the float chamber is raised, carrying the float and the attached recorder arm with it and making a record on a circular chart as it is revolved by a special clock at the desired speed.

By making cross-sections of the pressure chamber *B* of certain proportions as indicated, it is possible to produce a scale whose graduations are uniform for equal increments of flow or volume. The Bristol-Durand radii averaging instrument can therefore be used to obtain total flow or volume from the chart record of this instrument for a period of twenty-four hours.

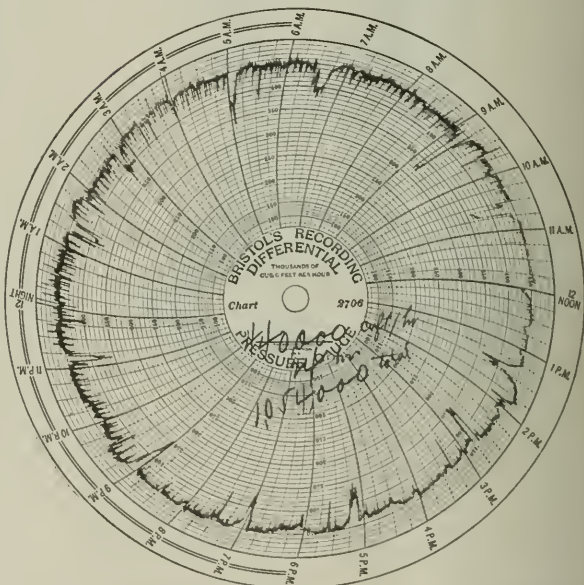


FIG. 3.—CHART TAKEN FROM DIFFERENTIAL PRESSURE GAUGE, RECORDING RATE OF FLOW AT BLAST FURNACE GAS

As these gauges will operate on extremely low differential ranges they are particularly well adapted for use with Pitot tubes for recording volumes of liquids, air or gases.

Fig. 3 is a reduced reproduction of a 24-hour record of the flow of blast furnace gas, made by one of these instruments in connection with Pitot tubes, at one of the largest blast furnaces in the United States. It will be noted that this chart is graduated in thousands of cubic feet of gas per hour, the range being from 0 to 500,000 cubic feet per hour. The working part of the chart above 150,000 is provided with uniform volume graduations. The actual differential pressure for the total range of the chart shown was 0 to 1.46 in. head of water. From this it will be seen that with a simple inexpensive Pitot tube it is possible to obtain sufficient differential pressures at slow rates of flow to directly record the amount and variations of the volume.

This instrument may be used to record the head of water in tanks under varying pressures, as, for instance, height of water in steam boilers and condensers. Another special application is that of recording the flow of water through a notch or over a weir, even though liquid is under pressure or a vacuum.

Magnetic Separation of Minerals

Although a number of machines have been designed and used successfully in the magnetic separation of minerals, less attention has been paid to the preparatory process of roasting. Recently this part of the work has been investigated more thoroughly, and a number of patents have been granted on apparatus and methods for making a proper magnetizing roast.

In the Campbell system of magnetic concentration the separation is accomplished by the conversion of the pyrite into a magnetic sulphide of iron in the Campbell furnace, and the subsequent removal of the magnetic sulphide by the Campbell separator. It is in the production of magnetic sulphide without loss of valuable metals in the roast that the Campbell process differs from all other magnetic separation processes. The operation of roasting to a magnetic sulphide is simple and easily controlled in the Campbell furnace. The pyrite is transformed superficially only, which means a minimum loss of sulphur, or gold, silver or copper which may reside in the pyrites. The resulting magnetic product obtained by the separator is therefore a valuable one for smelting or for the manufacture of sulphuric acid.

The Campbell roast is comparatively so slight that no metals are lost by volatilization, and a chalcopryite ore, with gold or silver, may be roasted without material loss.

Both the furnace and the separator are small, light in weight, simple in construction, durable, automatic in operation and saving of power. Unlike other furnaces, this one can be started and stopped without affecting the character of the roasted product, and need not be operated continuously, but may be started in the morning and shut down at night regularly or may be run continuously.

The roasting begins five minutes after the oil fire is lighted in the roaster.

The roaster is without brick or other lining, and only the front section needs replacing after a considerable time, which replacement is made from duplicates kept on hand.

Chances of uneven roasting are largely eliminated. The heat is controlled by the amount of oil and air admitted, and the feed is controlled by an automatic feeder. The ore is presented to the heat always in the same way, consequently the ore is uniformly roasted and to just such a degree as may be desired.

Everything about the plant is automatic, and a shovel never touches the ore from the time it is unloaded from car or wagon. It is first passed through a revolving dryer, heated by fuel oil fire and elevated to a dry storage bin with hopper bottom. As required, it is discharged from the bin, elevated to a screen, the undersize of which passes to the roaster, oversize to rolls, and back to screen.

The roaster product is conveyed to a revolving cooler with jet of water inside, through cooler, and elevated to conveyor to feed separators.

The zinc from separators is conveyed by spiral conveyer and elevated to zinc storage bin. The pyrites from separators are conveyed in similar manner to iron bin. Both of these bins have hopper bottoms, and with conveyors and elevators discharge the ores into cars for shipment.

The process has been very successful in the treatment of the marcasite-blende concentrates in the Wisconsin field. A fully-equipped plant, capable of handling 75 tons per day, is situated at Cuba City, Wis. This plant is so arranged that lots of ore, varying from a few hundred pounds to carloads, can be treated successfully; also, a plant is now in operation at Linden, Wis., by the Linden Zinc Company.

The process is calculated to make a very high-grade zinc concentrate carrying a high percentage of the total zinc in the ore; also a pyritic product carrying a high percentage of sulphur and such gold, silver and copper as may reside in the pyrites.

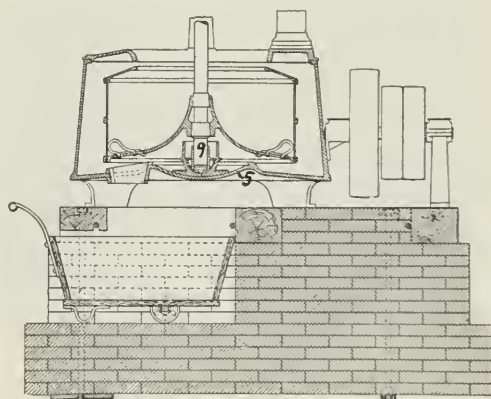
The Campbell Magnetic Separating Company, of Boston, Mass., in addition to the original Campbell patents covering the processes and apparatus for concentrating ores, has recently acquired other valuable patents relating to the magnetization of iron sulphides and apparatus for accomplishing the same.

A Self-Balancing Centrifugal

There has been much written on the subject of centrifugals, especially on their applications in certain chemical industries, but there still remains an important factor to be considered. That is, the question of taking care of an unbalanced load. As all users of centrifugals know, a basket or retainer out of balance to any extent when running at normal speed will rack itself to pieces in a short time or will do some damage to the building in which it is in operation.

The adjoining illustration shows a section of one of the various types of centrifugals built by the Tollhurst Machine Works, of Troy, N. Y., all of which are designed to be automatically self-balancing. For this purpose the Tollhurst centrifugal does not depend upon bumpers, rubber rings or springs, but by a very ingenious construction its basket and load supplies the means of balancing through the force of gravity in a flexible bearing that absorbs every particle of vibration.

The step in which the shaft or spindle revolves, resting in the saucer of base 5, is free to adjust itself to any position taken by the rotating spindle, but as the saucer forms a quick circle in relation to the top bearing, the step when leaving the center meets the resistance or force of gravity which is always tending to pull the step back to the lowest point of the saucer.



SECTIONAL VIEW OF BOTTOM DISCHARGE EXTRACTOR

In this way all vibration is absorbed, the movable or flexible step taking all the shock.

The heavier the load the smoother it will run, although it may be 5 to 15 lb. out of balance.

The illustration shows the Tollhurst self-balancing hydro-extractor in the special type which is arranged with bottom discharge. For such subjects as sugar, salts, crystals, etc., this method of unloading is very desirable. The foundation may be so constructed as to allow a truck to stand under the extractor.

After the liquor has been removed from the materials, the lid is raised, and the material falls down into a receptacle, thus doing away with the necessity of shovelling it out.

The Tollhurst Machine Works build centrifugals with the self-balancing feature for all kinds of work, especially for use in chemical laboratories, chemical works, laundries, silk mills, etc., and suitable for any kind of drive, also under-driven centrifugals having removable baskets specially designed for nitrated cotton or chemicals that must be kept free from any and all contaminations.

According to the chemical nature of the material to be treated the basket and case may be of plain metal or nearly any required composition.

The basket may be rubber-covered and the case may be lined with porcelain, lead or rubber.

A New Foundry Laboratory

The new laboratories of The H. M. Lane Company were formally opened by a reception to the Detroit foundrymen on March 22, which was attended by about one hundred people.

These laboratories are the outcome of the research work which has been carried on by Mr. H. M. Lane during the past dozen years.

They are located at 18 East Piquette Ave., Detroit, Mich., and occupy some 5000 square feet of floor space.

Mr. E. J. Woodison, of The E. J. Woodison Company, co-operated with the laboratory staff, furnishing the foundry supplies necessary in the laboratory and the material for the lunch.

Refreshments were served from six to seven o'clock, the menu consisting of baked beans, baked potatoes, baked sausages, rolls, pickles, doughnuts, and coffee. Everything that could be baked was baked in a core oven, and the hosts even declared that the doughnuts were cooked in core oil. The oven used was a Crawford gas-fired oven, and the temperature was regulated by a number of attached thermometers furnished by the Taylor Instrument Companies, of Rochester, N. Y. Each man's lunch was placed in a new E. J. Woodison 16-qt. foundry pail, and, according to the most modern core-room methods, these were delivered to the visiting foundrymen on this occasion by



A CORE MAKER'S LUNCH

a gravity carrier. This full dinner bucket scheme goes the serve-self lunch one better, as the bucket first delivers the lunch and then becomes the diner's seat.

As to the plant and equipment, the following notes will be of interest.

At the front of the building there are located the offices and draughting room, back of these there is the chemical laboratory, which is 15 by 40 feet. The remainder of the main building and the adjoining building are given up to the working core room and model foundry plant, containing the latest appliances in foundry equipment.

The objects of the laboratory are: To study core sands and core binders, so as to find the best mixture for the conditions present in different plants; to study new core binders in order to ascertain their exact value to the foundry; to determine the best method of mixing sands and binders for different classes of cores or molds. A careful study is also made of the proper baking temperatures, baking times, and different core oven fuels.

Special machinery is being installed in the laboratory for cleaning and recovering old core sand.

In the foundry department there is a melting furnace and appliances for pouring molds so as to test both the cores and the molds.

A room at the rear of the building is fitted with microscopes and cameras for the study of core problems.

A permanent exhibit of apparatus and supplies is also provided and for this purpose Mr. H. M. Lane has had the co-operation of a large number of manufacturers.

A Proposed Standard Method for Sampling Ferro-Vanadium

By Wm. W. Clark

It has been requested that a standard method for sampling ferro-vanadium be given, as considerable discrepancies in the analysis result from poor and careless sampling. It seems that the average chemist goes on the supposition that if one or two pieces are taken from each keg of alloy (ferro-vanadium comes on the market in kegs weighing from 150 to 200 lbs. each), that this sample is a good representative average of the material. Such is not the case.

Segregation is a commonly known term, and has caused considerable discussion in the steel business, where a few tenths or even hundredths of a per cent of one constituent are present and segregate considerably. How much more will segregation be when elements are present up to 40 per cent, such as vanadium, or to 2 or 3 per cent, such as aluminum or silicon?

Commercial ferro-vanadium, like every other complex alloy, on cooling, goes through a solidifying interval and not a sharp solidifying point. The aggregate solidifying first will rise or sink in the molten mother liquor, according to the difference between its specific gravity and that of the molten magma. The next and successive solidifying aggregates will follow the same course until the remaining molten mother liquor will have reached the eutectic composition, when it will solidify as a whole. During this process of solidification, it is thus apparent that the chances for the separation of the various constituents are very great, especially if the solidifying interval stretches over a great range of temperature, and if the difference in specific gravity of the crystallizing constituents is great.

Besides this tendency of the alloy to separate various constituents on cooling, we have to bear in mind the fact that ferro-vanadium is manufactured in small quantities in crucibles, which will naturally vary in composition from crucible to a certain extent, and will thus enhance the importance of careful sampling for obtaining the true average of a large quantity.

The method used in sampling is one which has been adopted after numerous experiments, and has been found to give uniform results on checking. The alloy is shipped in several different sizes, going from lumps 6 inches in diameter down to 10 mesh, according to requirements. When sampling the lumps a couple of chips are knocked off each lump with a chipping hammer. The regular size, which is crushed to lumps about the size of a walnut, is sampled during the process of kegging by running the hand along the front edge of each shovelful of the alloy, which has been well mixed after crushing, and taking off all that can be held in one hand of both fine and coarse material. The finer grades of alloy are sampled by mixing and quartering until a sample is obtained of the requisite size. A sample of at least 3 per cent of the bulk of the alloy should always be taken; however, when the alloy is very coarse this should be increased considerably.

The samples are first put down to rice size in a small crusher, the jaws of which are hardened steel. This crushed sample is quartered twice, and then put down to 20 mesh. This is quartered until about 2 ounces remain, which is all powdered in a hardened steel mortar, until it will pass through a 100-mesh sieve. This sample is well mixed for use in the laboratory.

It might be well to mention here the fact that ferro-vanadium is not uniformly magnetic, so that if a magnetic spatula is used in weighing weird results are the rule.

American Vanadium Co.,
Bridgeville, Pa.

Mexico has always been the world's largest producer of silver, the next largest output coming from the United States, Canada, Australia and South America.

A New Molten-Metal Pyrometer

The Brown Instrument Company, of Philadelphia, manufacturers of pyrometers, are placing on the market an improved type of thermo-electric pyrometer for measuring the tempera-

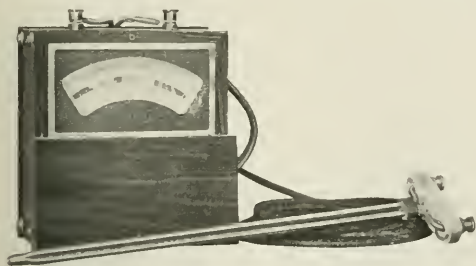


FIG. 1.—MOLTEN-METAL PYROMETER

ture of metals such as copper, brass aluminum, bronze, etc., in their molten state.

This company has been experimenting for some time in an effort to produce a thermo-couple which will withstand for a reasonable length of time the action of the molten metal. They have been able to produce a thermo-couple formed of nickel

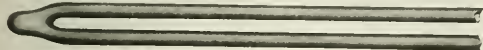


FIG. 2.—THERMO-COUPLE

alloy rods $\frac{1}{4}$ in. in diameter which are inserted directly into the metal. On account of this large cross section (Fig. 1) in comparison with thermo-couples heretofore used which have a cross section of about $\frac{1}{8}$ in. in diameter only, the life of the thermo-couple has been greatly increased.

In conjunction with this thermo-couple a high-resistance



FIG. 1.—AUTOMATIC CONTINUOUS DISTILLING MACHINE

portable indicating instrument is used which is unaffected by changes in the length of the wire connecting the thermo-couple and the instrument, and which is sufficiently robust in construction for ordinary foundry service.

It has long been recognized that if a reasonable life could be secured from a pyrometer, all foundries would adopt them on account of the importance of knowing the temperature at which the metal should be poured. In the case of this new Brown pyrometer the thermo-couple is inserted directly into the molten metal before pouring, and the temperature instantly noted. When it has once been definitely determined what the proper pouring temperature is for a certain metal a concern can reproduce results continually, producing uniformly good castings from maintaining the same temperature.

Distilling Machinery

In our April issue, page 224, we noted the new industrial laboratory and demonstration plant of Mr. Walter E. Lummis of Boston, Mass. The accompanying illustrations show some interesting machines installed in that laboratory.

Figs. 1 and 2 show an automatic continuous distilling machine for refining turpentine produced by destructive distilla-

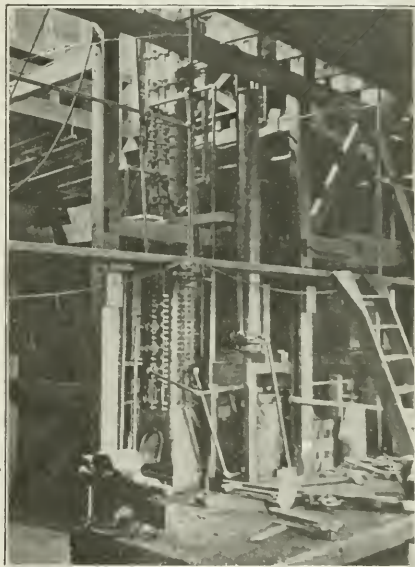


FIG. 2.—ERECTING SHOP, ASSEMBLING AN AUTOMATIC CONTINUOUS DISTILLING MACHINE

tion of long-leaf pine. Turpentine and pine oil are completely refined in a single operation. Fig. 2 shows the assembling of this machine in the erecting shop.

The continuous distilling machine shown in the cuts illustrates a small refining unit, equipped with positive controlling devices that regulate temperature, pressure, and cooling water, with absolute precision, obtainable for brief intervals in apparatus that is regulated by hand.

The machines are generally equipped with heat-exchanging devices or recuperators that utilize waste heat to warm the incoming raw material.

These distilling machines combine simplicity with large yields of products of uniform quality.

The economy of steam, in the case of very dilute or low-grade material, is sometimes equal to 80 per cent of the total that would be required with ordinary stills. The capacities range from 5 to 500 gallons per hour.

Figs. 3 and 4 show a continuous extracting and distilling outfit for simultaneous extraction of gums, oils, etc., and the recovery of the solvent in dehydrated or rectified condition. Fig. 4 shows the dumping, the extractor being inverted to discharge the spent material.

This experimental apparatus is capable of handling a great variety of materials by a diversity of processes.

It consists of a copper extraction cylinder of the dumping type, capacity 1 cubic foot, equipped with juice warmer and connected to tanks for supplying the solvent, and collecting the extract by gravity, and may be operated by either upward

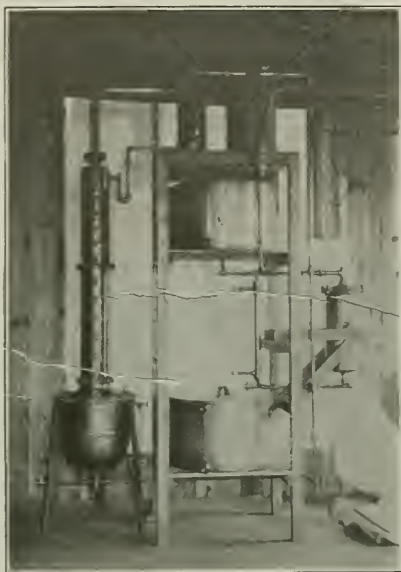


FIG. 3.—CONTINUOUS EXTRACTING AND DISTILLING OUTFIT

or downward extraction, in batches or by continuous process, with precise temperature regulation.

A still serves to concentrate the extract and recover the solvent. The still is provided with a plain still head and also with a very efficient rectifying column, the latter serving to repurify solvents that have become dilute or impure.

The extractor and still may be operated independently or in circuit by manipulation of valves.

The Lummus laboratory has 4000 sq. ft. floor space. There are two rooms for the assembling of distilling apparatus, 22 ft. high and 40 ft. long, also laboratories and work rooms. In the center of the building there is a trestle for supporting tanks and storing materials. The tanks are at different levels so that liquids may be run by gravity.

Occupational Diseases in Chemical Trades

A committee on occupational diseases in the chemical trades was appointed by the New York Section of the American Chemical Society in February, 1912.

The objects of the committee may be specifically stated as follows:

1. To hold itself ready to advise the Legislature of the States of New York and New Jersey in reference to matters pertaining to occupational diseases in the chemical trades.

2. To study various bills presented in the Legislatures in an effort to avoid unwise legislation; especially that which might be inoperative or ineffective from one or many reasons resulting from lack of technical knowledge at the time of writing the laws.

3. To inaugurate and superintend such investigations as might be decided upon which look toward improvement of conditions of labor in the chemical trades.

The personnel of the committee may be seen by a perusal of the membership, which is given herewith.

Dr. Charles Baskerville, Professor of Chemistry and Direc-

tor of the Laboratory, College City of New York, chairman.

Mr. R. C. Uhlig, Chief Chemist, Brooklyn Union Gas Company, Brooklyn, N. Y., secretary.

Dr. Geo. P. Adamson, Baker & Adamson Chemical Company, Easton, Pa.

Mr. W. H. Bassett, Technical Superintendent and Metallurgist, American Brass Company, Waterbury, Conn.

Dr. Wm. F. Doerflinger, Consulting Chemist, 52 Beaver Street, New York City.

Dr. H. M. Kaufman, General Manager Mutual Chemical Company of America, 55 John Street, New York.

Dr. Chas. F. McKenna, Chemical Engineer, 50 Church Street, New York.

Dr. A. C. Langmuir, Chief Chemist, Marx & Rawolle, 9 Van Brunt Street, Brooklyn, N. Y.

Dr. Charles L. Parsons, Mineral Chemist, Bureau of Mines, Washington, D. C.

Dr. Geo. A. Prochazka, General Manager, Central Dye Stuff & Chemical Company, Newark, N. J.

Dr. Geo. D. Rosengarten, Powers, Weightman & Rosengarten, Philadelphia, Pa.

Mr. A. H. Sabin, Consulting Chemist, National Lead Company, 129 York Street, Brooklyn, N. Y.

To date the following work has been undertaken or completed, namely:

Dr. Charles L. Parsons, Mineral Chemist of the Bureau of Mines, Washington, D. C., reports that the Bureau's experts engaged in studying mineral resources and utilization under instructions from Dr. J. A. Holmes, Director of the Bureau of Mines, are noting health conditions and collecting information on occupational diseases in chemical trades in so far as they relate to mining and metallurgy. When funds are available, bulletins will be issued on the subject from time to time.

Dr. Charles F. McKenna was retained by the Factory Investigating Commission of the State of New York, as chemical

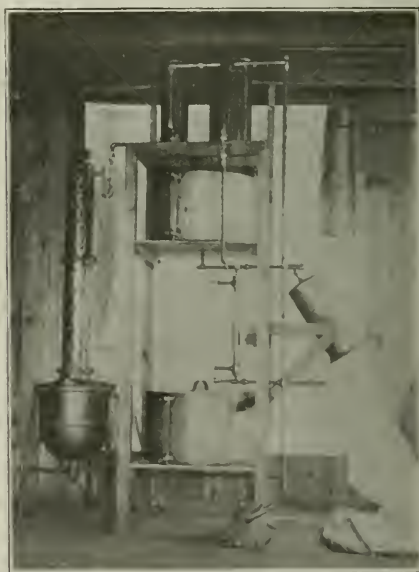


FIG. 4.—EXTRACTOR INVERTED TO DISCHARGE SPENT MATERIAL

adviser to Dr. Charles M. Price, Medical Director, who supervised the investigation of the chemical trades in New York State. Dr. McKenna's work was associated with the manufacture of "commercial acids."

Dr. Charles Baskerville prepared a report for the Factory Investigating Commission of New York on "Wood Alcohol"

These reports, which are now in press and will soon be available, contained suggestions for some new legislation. Bills have been introduced carrying these suggestions. Among them is one which involves the appointment of a chemical engineer as one of the four members of the Division of Industrial Hygiene of the proposed reorganized Department of Labor.

Vulcanized Fibre for Chemical and Electrical Purposes

Vulcanized fibre—a modified form of pure cotton cellulose—was first made in this country some forty years ago in Pennsylvania and West Virginia. In 1873 a plant was erected in Wilmington, Delaware, and since that time the American Vulcanized Fibre Company, of Wilmington, has developed the uses of this material for the most various purposes, for fibre trunks, fibre suit cases, fibre washers, for spigot packing and fibre valves, for telephone insulation, for gears, frictions, bearings, etc.

These different applications are made possible by the characteristic properties of vulcanized fibre which has the consistency of horn, has great mechanical strength, is tough, pliable, can be turned, bent, embossed, drilled, threaded, easily machined, and is insoluble in all ordinary solvents.

Its low electrical and thermal conductivity makes it the best insulating material.

With respect to chemical influences vulcanized fibre is said to be insoluble in all ordinary solvents, and is not deleteriously affected by alcohol, turpentine, naphtha, benzene, petroleum, or any of the animal, vegetable, or mineral oils, and by very few ordinary chemicals.

The special shapes in which it is manufactured include gaskets for steam, water, and oil; washers, gears, gear blanks, switch bars, and bases, valves, packings, etc., also tubes, sheets, and rods.

Personal

Dr. Edward G. Acheson, on his recent visit to Russia for the purpose of lecturing before the Russian Imperial Technological Institute on the nature and uses of deflocculated graphite and oilad and aquadag, was the guest of honor at a dinner at the American Embassy in St. Petersburg. The Russian government made the dinner a special demonstration of friendship between the United States and Russia. Count Witte and Premier Kokovtsoff, who seldom meet, were both present. Premier Kokovtsoff, with the Czar's approval, treated Dr. Acheson as a State visitor. Dr. Acheson was presented with signed photographs of several members of the nobility, while the Czar had conferred to him the insignia and sash of the Order of St. Ann. This order compares with the highest orders conferred in other European countries and conveys nobility to the recipient in the Russian empire.

Mr. Kosaku Asano, metallurgical engineer for the Ashio copper mines, Japan, has returned to that country after making a tour of the principal mines and metallurgical plants in the United States.

Mr. Frank C. Bowman, metallurgist for the New Reliance Company, Trojan, S. D., has returned to the Black Hills from a visit to Denver.

Mr. William A. Carlyle, until lately professor of metallurgy at the Royal School of Mines, London, has resigned his position and formed a commercial partnership with Mr. John F. Allan. Consulting engineering offices have been opened at 62 London Wall, E. C., London, England.

The firm of Emory & Eisenbrey, civil, chemical and industrial engineers, 2 South Fifteenth Street, Philadelphia, Pa., announce that Mr. J. R. Taft, formerly with Coverdale & Company, Inc., New York City, has become a member of the firm. A New York office of the firm will be conducted at 50 Church Street.

Mr. Milton M. Kohn, the inventor of the multiple re-

placeable unit system of electric laboratory furnace, has just returned from a business trip to Europe.

E. J. Lavino & Co., of Philadelphia, Pa., importers of ores and ferro-alloys have been appointed sole American agents for the Société Anonyme des Bauxites de France, who are the largest producers of bauxite in France, and although they have been without previous representation in the United States it is said that they have been shipping from France the bulk of the bauxite which has come to the United States.

Mr. Henry Schmitzel, general manager for the Golden Reward Mining Company, Deadwood, S. D., was in Denver recently, enroute to New York City on a business trip.

Mr. Edwin G. Steele, of the engineering firm of Sutton, Steele & Steele, Dallas, Tex., has returned to Dallas after spending several weeks in Leadville and Denver, Col., investigating the ore situation at the former place, and supervising the erection of a large testing plant at the latter.

Mr. Charles E. VanBarneveld, until recently professor of mining in the University of Minnesota, has been appointed chief of the Department of Mines and Metallurgy for the Panama-Pacific International Exposition, at San Francisco, 1915. Mr. Van Barneveld is a graduate of McGill University and well qualified for the position.

Mr. Leslie H. Webb, secretary of the Wedge Mechanical Furnace Company, of Philadelphia, has returned home from a two-months trip to the Pacific Coast.

Mr. Pope Yeatman, consulting engineer for the Braden Copper Company, Chile, South America, is at the property on a trip of inspection. He will also visit the property of the Chile Exploration Company at Chuquicamata.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Continued).

703,857, July 1, 1902, Constantin Jean Tossizza, of Paris, France.

Refers to his application for patent 710,346, and relates to the electrolytic deposition of such metals as zinc, nickel, cobalt, and cadmium, from their sulphates, by charging the solution with sulphurous acid and electrolyzing under conditions that do not decompose this acid or liberate hydrogen. With zinc sulphate soluble anodes of pure copper are used, metallic zinc separating at the cathode, while copper sulphate forms at the anode. The voltage necessary for the operation is 1 1/10, and with currents of 40 amperes per square meter rises to about 1 5/10 volts. The copper sulphate is subsequently saturated with sulphurous acid and electrolyzed with 31/100 volt. The liquid remaining from the copper electrolysis serves to dissolve more zinc ore.

704,639, July 15, 1902, Carl Hoepfner, of Frankfort-on-the-Main, Germany; Henry Orth, Jr., administrator of said Hoepfner, deceased.

Refers to his prior patent 507,130, and relates to the extraction of metals, such as copper, from their ores, by a warm solution of cupric chloride containing less than its saturation quantity of alkaline chloride. This solution will dissolve the lead and silver from the ore, which are precipitated by refrigeration, and the cuprous chloride formed by the solution of the lead and silver is converted by chlorine into cupric chloride. The iron present is precipitated by oxy-chloride of copper, and the remaining solution heated and returned to the ores to dissolve the copper with some zinc. Any lead present may be precipitated by zinc oxide. The copper and zinc are now preferably precipitated electrolytically.

706,436, August 5, 1902, Frank Townsend Mumford, of Kalgoorlie, Western Australia, Australia.

Relates to apparatus for the treatment of electrolysis of ores and slimes, also clay-like and slimy ores, or such as require

fine crushing, of the precious metals. The apparatus consists of a long steel cylinder lined with copper which is amalgamated, and has a layer of mercury at the bottom. Through the center of the cylinder and concentric therewith are a number of carbon or iron rods connected as anodes; the copper lining serving as the cathode. The electrolyte may be potassium or sodium cyanide, chloride, or bromide, or a mixture of them. It should preferably contain a salt capable of yielding oxygen to accelerate the solution of the gold. The pulp is discharged after the operation is completed, the mercury removed and the gold amalgam treated in the usual manner to recover the gold.

709,817, September 23, 1902, Clinton Emerson Dolbear, of Boston, Massachusetts; assignor to American Mining and Metal Extraction Company, of Boston, Massachusetts.

Relates to a method of electrolytically extracting metal from ores such as copper, etc. The ore is suitably crushed and electrolyzed in a vat with an electrolyte consisting of sulphuric acid containing nitric acid, or a nitrate, such as sodium nitrate. Insoluble anodes, preferably of carbon are used, and a cathode of the same metal to be extracted. During the process the nitric acid dissolves the ore, the oxygen liberated by electrolysis oxidizing the lower oxides of nitrogen back to nitric acid, whereby no nitric acid is lost; the metallic nitrate formed is converted into sulphate, from which a reguline deposit of metal can be obtained on the cathode.

710,346, September 30, 1902, Constantin Jean Tossizza, of Paris, France.

Relates to the electrolytic deposition of copper or other metals from solutions of their sulphates using insoluble anodes and depolarizing the anode by sulphurous acid, or the introduction of sulphur dioxide gas through a hollow anode. The roasted ore is dissolved in sulphuric acid, and the sulphur dioxide given off in the roasting is collected and used for depolarizing the anode. The solution of ore obtained is electrolyzed with a voltage of 6/10 of a volt, whereas heretofore electrolysis of such a solution with insoluble anodes required at least 1 3/10 volts.

714,598, November 25, 1902, Sidney Theodore Muffly, of Bowdre, Georgia; assignor of one-half to Runyon Pyatt, of New York, N. Y.

Relates to an apparatus for the separation of metals from their solutions by electrodeposition, and refers to the application for his process patent 714,599. The apparatus consists of a plurality of precipitation boxes, in cascade, with the electrolyte, flowing through all the boxes in series, a filled box being removed from the beginning, and a fresh, empty one inserted at the end of the series. The boxes are divided into two compartments, one constituting a conduit, and the other an electrolytic compartment. The latter contains an anode of iron or other suitable material, and a mattress-cathode made of cellular porous carbon casings containing a carbon plate, and filled with a filiform packing of lead and zinc composition. The electrolyte consisting of a cyanide solution containing gold and silver, or a similar solution, flows through the box, down the conduit, and up through the mattress-cathode, in which the metals are deposited both by the electrolysis and also by the filiform lead-zinc packing. Compressed air is blown through the cathode during deposition. From this cathode, the solution overflows into another similar box.

Book Reviews

The Metallurgic Industry in Italy.—Description of the principal works. LXI and 430 pages, 7 x 10. Associazione fra gli Industriali Metallurgici Italiani, Milano.

This book was written in honor of the (British) Iron and Steel Institute and was intended for distribution among those of its members who would join the trip to Italy, in connection with the industrial exposition at Turin. It will be remembered that after all plans and preparations had been made, this trip of the Institute to Italy was finally abandoned.

The young and flourishing Italian industries had done everything in their power to tender a worthy reception to their British guests, and the above "Associazione" or, as we might say, the "Italian Society of Industrial Metallurgists," had prepared this instructive description of the country and the industry, with details on nearly every plant, where the metallurgical profession helps producing goods for commercial purposes or for national defense. The book is beautifully illustrated.

Although it is written in English we cannot help mentioning that we would mark this book with the mysterious sign of the cable companies L C O ("language of the country of origin") and not L C D as it is intended. The reviewer being acquainted with the Italian language had no trouble in understanding the text, and after all, what counts in a book of this kind, is the material—and this is all in the book—and the fine spirit in which it is written.

An Expensive Experiment: The Hydro-Electric Power Commission of Ontario. By Reginald Pelham Bolton. New York: The Baker & Taylor Co., \$1.25.

This book is an amplification of the author's evidence before the New York State Committee on the subject of the activities, operation and results of the Hydro-Electric Power Commission of Ontario. Its appearance is particularly timely on account of the bearing of the subject on the schemes at present being advanced for the entrance of the State of New York into the business of power development and supply.

While giving the Ontario Commission credit for conducting "a difficult problem in an honest and sincere manner," the author's investigations lead him to the conclusion that its operations have resulted in a considerable deficit, which is being disguised by the temporary suspension of the sinking fund, by the omission of provision for depreciation, and by "some method by which a part of the cost of operation, maintenance, and general expense, upon the system, is dispersed into the cost of new construction." These conclusions are abundantly supported by figures which include a rather damaging analysis of the financial statement issued by the Ontario Commission in January, 1913.

There appears to be little escape from the author's view that the Commission is selling power not at cost, but at prices considerably below cost as determined upon ordinary business principles, and that the difference is being provided for at the cost of the taxpayers of the Province as a whole. The further extension of loading can only result, in the author's estimation, in a still greater loss; but while this is probably true, we feel that he has scarcely taken proper account of the relief in overhead charges that such increased loading may bring.

Less satisfactory are those portions of the book which deal with the general subject of the development, transmission and utilization of hydro-electric power. The author sees quite clearly that for municipal and similar purposes there is, except in exceptional cases, no economic advantage in hydro-electric power over steam or gas-generated electric power; but when he is found assuming an efficiency of 54 per cent as representative of the transformation of water power into electric power, producing once more the bewildered fallacy that the expenditure in travel induced by Niagara Falls is an economic asset, and insinuating, by means the photograph opposite page 64, that a phenomenon which has been observable as a result of low water at Niagara Falls before power developments were undertaken, is "the effect in part, of the expensive experiment on the Canadian Falls," he is writing, not with the knowledge and detached judgment of an engineer, but with the lack of knowledge or prejudice of a public agitator of the dime magazine type.

We deplore these blots on the work all the more since there is no question but that the author has rendered a useful public service in throwing a clear light upon the Ontario Hydro-Electric Commission's position and in dispelling to a considerable degree, by definite figures, the secrecy surrounding its financial operations.

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The Willard Gibbs Medal

The report of the presentation of the third Willard Gibbs Medal to Dr. Leo H. Backeland, which is published elsewhere in this issue, suggests a brief comment on medals in general and on the Willard Gibbs medal and its present recipient in particular.

There are medals and medals. Their variety is unlimited whether we consider them as pure works of art or as standing for a purpose. From a purely artistic viewpoint the renaissance of the medal has been one of the phenomena of art evolution during, say, the last forty years, and a study of modern medals may be made a most interesting source of instruction, as it will reveal some principal leitmotifs in the evolution of art of different nationalities and different schools.

But medals are far more important by what they stand for. And in this respect the modern renaissance of the medal is due to the complete change in public sentiment from exclusive war-hero worship to a broad recognition that honor is due even more forcibly to those who have accomplished works of peace and contributed to the positive progress of human civilization. This country has the Civil War Medal, the Indian War Medal, the War With Spain Medal, and so on. But it has now also a series of medals, founded in recent years by scientific and engineering societies, such as the Edison Medal, the John Fritz Medal, the Willard Gibbs Medal, and the Perkin Medal. And if the competition between different societies in founding new medals seems at times almost to degenerate into a fad or a craze, it still emphasizes the healthy psychological attitude that is at the bottom.

We could easily add to the list of prominent medals conferred for engineering or scientific attainments. But those we mentioned—the Edison, John Fritz, Willard Gibbs, and Perkin medals—emphasize by their very names one and perhaps the most prominent function which these medals have to fulfill—to link together the name of the man in whose honor the medal was founded with the name of the recipient of the medal, to link together the past with the present and the future, or in a single word to give inspiration.

To make our point clear with especial reference to the Willard Gibbs medal and its award to Dr. Backeland, we may refer to the words of Dr. Richardson in his happy presentation speech. "The man who accepts a medal takes upon himself a heavy responsibility. Like the man in the story he places himself in the hands of his friends in a most absolute way. There is no relief or rest for him from the time he arrives in town until he is safely aboard a train and on his way home. There is every likelihood of his being over-entertained and over-feasted. . . ." All this is very true, but the concluding sentences would seem to over-

shadow a deeper meaning intended, we think, in the first sentence quoted.

Willard Gibbs died ten years ago and Dr. Wilder D. Bancroft in the obituary written for this journal (Vol. 1, page 313) made some pointed and impressive remarks which are fundamentally as true to-day as they were ten years ago and which we quote because they came back at once to our memory in connection with the sentences of Dr. Richardson given above. With respect to Gibbs' Equilibrium of Heterogeneous Substances Dr. Bancroft wrote:

"While the fundamental importance of this paper is now generally recognized by chemists, it must be admitted that this is very largely lip service still. There is no man who pretends to understand Gibbs. There are some who have understood small portions of this classical work, but that is all. As one distinguished scientific man has said, it is easier to discover a thing oneself than to find it in Gibbs. This is perfectly true. But when one has, after much effort, discovered a thing, one finds it all in a more general form in Gibbs. The trouble is not that Gibbs is obscure. After one has mastered a small portion, one wonders that there ever should have been any difficulty in understanding that particular passage. But the difficulty reappears with the next passage. Gibbs' mind works along different lines from those of the chemist of to-day and the latter cannot follow. We all know that Faraday's reasoning seemed obscure until Clerk-Maxwell put it into mathematical form. When some experimental chemist succeeds in doing for Gibbs' mathematics what Clerk-Maxwell did for Faraday's experiments, then for the first time shall we appreciate the real greatness of the man whose loss we mourn."

We need hardly draw, in many words, the logical conclusion from our connection of these sentences with the presentation of the Willard Gibbs Medal to Dr. Baekeland. We do not know anyone in the world who by ability, enthusiasm, and perseverance would be more predestined to do for Gibbs' mathematics what Clerk-Maxwell did for Faraday's experiments, than Dr. Baekeland. Will he do it? We should not wonder.

Zinc-Dust Precipitation

The continued and increasing use of zinc dust as a precipitant for gold solutions in the cyanide industry is gradually extending our practical knowledge of the operation, even though all the chemistry involved may not be definitely known. As in the development of any novel process the original advocates of zinc dust precipitation hedged the method about with certain precautions and restrictions which were dictated by their best judgment and reason. But likewise, as in the further application of such novel processes, the actual use of zinc dust in the hands of various metallurgists and under a variety of conditions has gradually altered the original practice. Step by step some of the negative precautions have been disregarded with apparent impunity, while at the same time the original positive instructions have been altered with consequent improvement. Thus, both by accident and design, fallacies have been exposed and our practical knowledge increased.

It will be recalled that one of the points strongly emphasized in the early literature on zinc-dust precipitation was the necessity of preventing oxidation of the dust prior

to its mixture with the gold solution. The idea was strongly held that efficient precipitation would occur only when the dust particles were fresh and uncontaminated by oxidation products. In fact, small tube mills were used to brighten the zinc just before using and successive charges of dust were constantly undergoing this treatment in order that fresh material might be available for precipitation.

This feature was embodied in patents granted to one metallurgist, who proposed to use a tube mill or other grinding apparatus in the precipitation circuit so that the mixture of dust and solution should pass through it. This procedure was adopted because such precipitants as zinc dust "are rendered less efficient and active by reason of the presence of a detergent coating upon the surfaces of the particles thereof, caused in part at least by their well-known property of oxidation." The mechanical removal of the coating of oxide by abrasion was regarded as essential to efficient precipitation, because otherwise it must be removed by solution before good contact could be obtained.

It has come about, however, in at least one instance that the protection of zinc dust against oxidation by exposure to air is wholly disregarded. At the Homestake slime plant where zinc-dust precipitation was first installed on a large scale the dust is exposed to air in shallow pans for periods ranging from several days to a week or more. No provision is made for subsequently grinding the dust to remove any coating of oxide that may have formed, and it is fed dry to the gold solution and pumped to the press. Nor have any ill effects followed this disregard of a generally accepted precaution, all of which suggests the probable fallacy of the original idea.

Another feature of zinc-dust precipitation, which was early adopted and later abandoned, was the practice of forming an emulsion of dust and solution which was then added to the main body of the latter and pumped to the presses. The method usually adopted for forming the emulsion was to mix zinc dust with a small stream of barren solution in an inverted cone and to agitate the mixture by a jet of air. The emulsion overflowed from the cone and joined the gold solution on its way to the press. An investigation into this method of precipitation made some time ago by Mr. A. J. Clark, metallurgist for the Homestake company, led to the condemnation of air agitation in connection with gold solutions prior to precipitation. The reaction of the carbon dioxide of the air with the lime dissolved in the solution, formed calcium carbonate which was found to exercise a genuine deterrent effect on precipitation by coating the zinc particles and preventing contact with the solution. On the removal of the air-agitated mixing cones and the substitution of dry feeding direct to the precipitation line the troubles due to lime coating disappeared.

The ill effects of calcium carbonate as first determined at the Homestake are now generally recognized and the emulsifying cone with its air agitation is no longer in general use. The original theory seems to be well founded and has not been changed. On the other hand, the exposure of zinc dust to the air before using is not yet generally practiced, but it may be only a short time until the old opinion on this point may be broadly regarded as a fallacy.

The Detailed Pig-Iron Statistics

Supplementary to the general pig-iron statistics for 1912, issued Jan. 30 by the Bureau of Statistics of the American Iron and Steel Institute, a detailed statement is made under date of May 15, showing the distribution as to grade and also the distribution as to condition in which used, the latter being a very commendable and interesting improvement over the presentation of previous years.

A slight correction (involving about 1,000 tons) is made to the total pig-iron production of 1912, placing the figure at 29,726,937 gross tons, against 23,649,547 tons in 1911 and 27,303,567 tons in 1910, the best previous record for a calendar year. Each year one likes to test the old trade dictum that pig-iron production doubles every decade, hence we observe that the 1912 production is 1.67 times the best production made through 1902, 3.24 times the best production made through 1892 and 6.42 times the best production made through 1882. The rule of doubling held good through 1902, but has not held good through 1912. The rule easily held good through 1906 but has since broken down and cannot be reinstated except by extraordinary gains in future.

The statistics of production by grades shows a continuance of the change shown in recent years, that the production of foundry grades does not increase as rapidly as the production of steel-making grades. Despite the large gain in total pig-iron production in 1912 the production of foundry and ferro-silicon, 5,073,873 tons, was less than the corresponding production in 1907, 1909 and 1910. This is a part of the natural evolution whereby if rolled steel does not actually supplant cast iron it at least enjoys by far the major part of the increased demand as the general use of ferrous products increases.

The production of Bessemer and low-phosphorus iron was 11,664,015 tons against an output of 11,417,886 tons of basic iron. The small difference was probably exceeded by the production of low-phosphorus iron—limited to 0.04 per cent in phosphorus—leaving it that the production of Bessemer iron—limited to 0.10 per cent in phosphorus and invariably made very close to that limit—fell slightly below that of basic iron, whereas ten years earlier 5 tons of Bessemer iron were made to 1 of basic. The maximum production of Bessemer and low-phosphorus iron occurred in 1906, with 13,840,518 tons, and obviously in the present order of things that tonnage will stand as record.

The condition in which iron was delivered for consumption is reported for the first time, the figures for 1912 being as follows:

Molten	16,466,722
Sand cast	6,309,495
Machine cast	6,214,121
Chill cast	726,017
Direct castings	10,582
Total	29,726,937

To the blast-furnace manager it may be interesting to note that 726,017 tons were "chill cast," which we understand to mean simply that the chills were placed by hand and were not in a casting machine, representing probably a limited demand for chill-cast iron from furnaces which

have not found it desirable to install machines. The metallurgist will simply add the "machine cast" to the "chill cast" finding that 6,940,138 tons were cast in chills. It would have made the statistics more interesting to report how much of each grade was delivered in the different forms, but one can make fairly close surmises. We know that all the basic iron was either molten or chill cast, and that very little Bessemer is sand cast, most of it being used molten and a comparison of figures indicates that nearly all the foundry and malleable iron was sand cast. In other words, the foundries have not taken to chill-cast iron as it was thought a dozen years ago they might.

The molten-iron deliveries of 16,466,722 tons were exclusively to the Bessemer converter and the open-hearth steel furnace, outside of a limited tonnage delivered for the direct casting of ingot molds, whereby it may be taken that 16,000,000 tons of Bessemer and basic iron were used for steel making out of a total production of about 23,000,000 tons. The steel mills which regularly practice the direct metal process must necessarily use some cold iron, including "Sunday metal," iron shipped from detached furnaces, broken molds, etc., so that their proportion is more than three-fourths of the total steel industry and may easily be more than four-fifths.

This represents a rapid growth in a relatively easily practiced economy.

Shape and Rod Production

Production statistics of the American iron and steel trade make their appearance in rather irregular order. Thus early in the year the rail production of 1912 was reported, while in the past fortnight the statistics of rods and structural shapes have been returned but no statistics have yet appeared of steel ingots.

Rails were reported at 3,327,915 gross tons, a gain of 18 per cent over 1911, but a loss of 16 per cent from the record year of 1906; rods are reported at 2,653,553 tons, a gain of 8 per cent over 1911, which furnished the record, while structural shapes are reported at 2,840,487 tons, a gain of 48 per cent over 1911 and of 25 per cent over 1909, the previous record year. Thus outputs in the different finished products do not hold well together at all. The different products contribute quite different proportions to the total in different years. Were it the case that rails, rods and shapes furnished the same proportion in 1912 as in 1911, three-eighths, the total output of rolled iron and steel in 1912 would prove to be about 23,400,000 gross tons, against a record output of 21,021,270 tons in 1910 (of which 1,740,156 tons was rolled iron) but we expect the 1912 total when finally reported to be considerably larger than this conventional amount.

The structural shape production in 1912 was extremely heavy as compared with previous years, when it is considered that the year was not regarded as one of exceptional activity in the erection of large bridges and buildings. In 1906, on the other hand, there was a great deal of such work, yet the 1912 tonnage exceeds that of 1906 by 34 per cent. From 1892, the first year of structural shape statistics, to 1912 there is an increase of 525 per cent, the tonnage was more than sextupled.

Readers' Views and Comments

New York Hydroelectric Development

To the Editor of *Metallurgical and Chemical Engineering*:

Sir:—There is no doubt that a public service of the first magnitude can be rendered by the engineering press in pointing out the ill-considered nature of the present proposals for the operation of power plants by the State of New York. The consummation of these schemes can only result in a reduced price of power at the cost of the taxpayers of the State as a whole. To Mr. Reginald Pelham Bolton much credit is due for his thorough, albeit somewhat one-sided and prejudiced exposure, in his recent book entitled "An Expensive Experiment" and reviewed in your last issue (p. 302), of the actual results of the operation of similar schemes in the Province of Ontario.

It is not clear, however, what good purpose can be served by such "economic" arguments as that contained in Mr. Bolton's letter to the *Electrical World* of April 12 to the effect that in the case of Niagara Falls "the financial benefits derived from the vast number of visitors who bring to the channels of general trade, as reported by the Committee of Engineers appointed by the United States Government, an annual revenue exceeding \$20,000,000" are far greater than the value of the power that could be obtained by a more complete utilization of the Falls.

This argument is either unintelligent or insincere; its logical strength can be shown by the *reductio ad absurdum* that, if the economic value of anything is to be measured by its power of inducing unproductive expenditure, we shall have to conclude that prostitution is one of the greatest commercial assets of New York State!

Nor are Mr. Bolton's estimates of the economic value of Niagara power more reliable than his economic logic. Its value is only to be measured by a realization of what the products of the enormous group of electrochemical industries, which have sprung up at Niagara Falls as a direct result of the power development and which constitute a most efficient consumptive outlet for the bulk of the power, have done for the advance of industry in general. Possibly Mr. Bolton has no such realization; and if this is so, I shall be glad, if he will call upon me when his investigating journeys next bring him to this neighborhood, to see that he gets from men connected with these industries some idea of their enormous and far-reaching economic importance. It will further be demonstrated to him by the historic method (this ought to appeal to him particularly) that without the development of Niagara power the development of these industries would have been set back ten years, if indeed it would have been at all possible.

The trouble with Mr. Bolton is that he is partly an engineer and partly a "trustee of the American Scenic and Historic Preservation Society." Niagara Falls is his "King Charles' Head." Pending his receipt of instruction on the points mentioned above, let him bury it, if possible, when discussing subjects to which it has no relation.

F. AUSTIN LIBBURY.

Niagara Falls, N. Y.

The "Thermist"

To the Editor of *Metallurgical and Chemical Engineering*:

Sir: I note with interest in your May, 1913, issue Mr. H. M. Boylston's suggestion to call "thermist" the technical specialist in charge of the heat treatment of metals in industrial plants. The recommendation seems a happy one to me. The term is short and suggestive and infinitely superior to such periphrasis as the metallurgist, or metallographist, or metallurgical engineer, or chemist, or engineer of test "in charge of heat treatment."

There certainly is a growing need and demand for engineers and metallurgists possessing an expert knowledge of the heat treatment of metals and especially of iron and steel and capable of applying theoretical principles in a practical and effective manner to the daily commercial production of finished or semi-finished articles. And may I not seize this opportunity to express a thought that has come to my mind more than once, namely, that technical schools teaching metallurgy should recognize the need and demand to which I have just referred by providing in their last year's curriculum special and practical laboratory courses in metallurgical pyrometry and heat treatment of metals, subjects which are at present but scantily treated?

Harvard University.

ALBERT SAUVEUR.

Simple Rules for Temperature Conversion

To the Editor of *Metallurgical and Chemical Engineering*:

The methods given by Mr. Childs for temperature conversion in your May issue (page 230) seem of dubious value. Reduced to formulas they are:

$$\begin{array}{r} 5 \\ (F + 40) - 40 = C. \\ 9 \end{array}$$

$$\begin{array}{r} 9 \\ (C + 40) - 40 = F. \\ 5 \end{array}$$

The old rule seems simpler.

$$\begin{array}{r} 5 \\ (F - 32) = C. \\ 9 \\ - C + 32 = F. \\ 5 \end{array}$$

In view of the enormous waste of brain tissue going on in evolving "long cuts" and approximations for metric conversion allow me to suggest a rule I have found most satisfactory for all such work: Learn the correct formula and use it.

DONALD M. LIDDELL.

Elizabeth, N. J.

Inverse Weighing of Light Objects on Heavy Weight Scales

To the Editor of *Metallurgical and Chemical Engineering*:

Sir:—It often happens that one desires to weigh with moderate accuracy a small object, when only scales for heavy weighing are available. On such an occasion the writer conceived the following method of accurate weighing, which may prove useful to others.

The principle involved is based upon the simple relation of the beam weight to the platform weight, which in the scales used was one to one hundred, i.e., 1 lb. attached to the suspended beam weight was just equipoised by 100 lb. on the platform. Having adjusted the empty scales to equilibrium, object (X) to be weighed was then attached to suspended beam weight, and any convenient material (M) was placed on platform until scales were again in equilibrium. Object (X) was then removed from beam, and the weight of the material (M) determined as usual. Then the weight of the material (M), divided by 100 equalled the weight of the object (X).

The beam being graduated to and sensitive to 0.25 lb., made the scale sensitive to 0.04 oz. by this inverse method of weighing.

Homestead Steel Works,
Munhall, Pa.

WALLACE B. CROWE.

Effect of Power Development on the Canadian Falls at Niagara

To the Editor of *Metallurgical and Chemical Engineering*.

Sir: Facing page 64 of Mr. Reginald Pelham Bolton's book entitled "An Expensive Experiment," and reviewed on page 302 of your May issue, is a photograph with this legend:

"The Effect, in Part, of the Expensive Experiment on the Canadian Falls." (This picture is reproduced herewith as Fig. 1.)

The implication is that the rather small amount of water flowing over the portion of the Falls shown is due to the power development. In order to make this evidence conclusive it is, of course, necessary to make a comparison with a photograph taken before the power development.

On the title page of Mr. Bolton's book is a list of various societies of which he is a member, the character of which would indicate that he is a man of strict mental probity and scientific attainment. The first quality would exclude the possibility of his having deliberately used a misleading photograph; the second excludes that he should have failed to make the necessary comparison.

I send you herewith a photograph showing the same portion of the Goat Island end of the Canadian Falls. (This is reproduced in Fig. 2.) In Mr. Bolton's illustration (Fig. 1) may be plainly seen a portion of the intake of the Ontario Power Company, thus establishing the date of the photograph as *after* that power installation, while in the photograph I send (Fig. 2) this feature is missing, thus establishing its date as *before* the power installation. This may well be, since the photograph I send is from a well authenticated negative by Nielson, at least 20 years old.

It will now be interesting to see upon which horn of the dilemma Mr. Bolton chooses to impale himself.

P. MCN. BENNIE.
Fitzgerald and Bennie Laboratories,
Niagara Falls,
New York.

The Chapman Engineering Company, Mt. Vernon, Ohio, report considerable activity in the gas producer field. Among the recent sales of their standard 10-ft. producers are six for the Forged Steel Wheel Company, Butler, Pa., two for the Railway Steel Spring Company, Inter-Ocean Works, Chicago Heights, Ill., one for the Carnegie Steel Company, Mingo Works, Mingo Junction, O., two for the Nova Scotia Steel & Coal Company, Sydney Mines, Nova Scotia. The Bethlehem Steel Company, South Bethlehem, Pa., ordered six more Chapman producers for open-hearth work.

Iron and Steel Market

New buying of pig iron and steel products came to almost a complete stop in May, while buyers continued to take full pig-iron shipments on old purchases, and in finished steel products specifications continued at a fair rate, averaging 60 to 75 per cent. of the current shipments.

Almost all the mills had accumulations of specifications for



FIG. 1.—GOAT ISLAND END OF CANADIAN FALLS AFTER POWER DEVELOPMENT
(Reproduced by Photography from Mr. Bolton's Book)



FIG. 2.—OLD PHOTOGRAPH TAKEN BEFORE POWER DEVELOPMENT

finished steel products, some of the accumulations being quite large, and as a consequence some mills can stand the reduced rate of specifying much better than others. There is disclosed a striking difference between the position of different mills,

which seems clearly ascribable to the difference in sales policies last autumn. At that time, when the general buying movement was in progress, some mills quoted slightly lower prices than others, but were very particular as to the character of business booked. Others were stiff in their views as to prices but very accommodating as to the kind of contract they would write. Had demand continued extremely good, the latter would have fared the better, but as it is the more conservative and painstaking mills have occasion to reflect upon the truth of the adage referring to the relative desirability of a bird in the hand as against two in the bush.

Production throughout the iron and steel industry has been maintained at the full rate, involving practically record production and shipments. According to all the evidence there is no accumulation of finished product in the hands of buyers, and apparently therefore the experience of 1910 is not being repeated, when buyers received material simply because it was cheap. It is barely possible, however, that such judgment is premature. There is undoubtedly much low-priced material still being delivered.

The future of the market is altogether in doubt, and particularly so when it is observed that there is the widest divergence in the views expressed in the trade as to when improvement is to be expected. The least hopeful view expressed is that next year as a whole will be a poor one, there being no opinion expressed that a period of years will elapse before recent favorable conditions are duplicated. In some quarters an improvement in the early autumn seems to be expected. While no predictions are made, it is probably generally believed, even by producers themselves, that another upturn cannot occur without a decline in prices first furnishing the foundation.

Pig Iron

The decline in pig iron prices has been proceeding at a more rapid rate than in the early months of the year, prices having started to yield fractionally toward the close of December. Averaging all grades, the rate of decline since early in April has been about 75 cents per ton per month, against a rate one-third or one-half as great in the first quarter of the year. The decline has been fairly uniform geographically, but not so as to grades, since foundry iron has declined considerably more than steel making grades, Bessemer and basic. The latter have not been supported by market purchases to any extent, but are supported in great measure by the heavy deliveries on what are practically requirement contracts, such contracts being entered into for an extended period, the furnace to keep the buyer supplied and the iron to be settled monthly according to the disclosed market price for the month. Foundry iron buyers are fairly well covered to July 1, few contracts running beyond that date, while in quite a number of instances deliveries have been anticipated so that contracts will be filled before July 1. Buyers are all holding off until the last moment, but an active point must be reached soon, probably before the end of June, with prices continually declining and consumers reaching the end of their supplies. The market is quoted in the following figures as closely as can be done, but in some instances there is no basis for close quotations and in such cases a price is named which it is known could be done, but which in all probability could be shaded very materially should definite inquiry be made, such cases being indicated by the word nominal: Bessemer (nominal) \$17; basic (nominal) \$15; No. 2 foundry and malleable, \$14.50; forge, \$14, all at valley furnaces, 90 cents higher delivered Pittsburgh or Cleveland; No. 2 foundry, Philadelphia, \$16.75; Buffalo, at furnace, \$15.50; No. 2 foundry, delivered Cleveland (nominal) \$15.25; Chicago, at furnace, (nominal) \$16; Birmingham, \$11.50.

Steel

A complete change has occurred in unfinished steel. For several months during the winter the mills refused absolutely to sell for forward delivery, while brokers, and an occasional small mill, offered odd lots for prompt shipment, prices being generally about \$20 for billets and \$30 for sheet bars. Buyers

could not have afforded to pay similar prices for large tonnages for extended delivery, but as none such were offered even at those prices, the general market, such as it was, had to be quoted at \$20 for billets and \$30 for sheet bars. Early in April limited sales began for third quarter, \$27.50 being accepted for third quarter sheet bars. The prompt market declined slightly, but there were two distinct markets, prompt and forward. In May the mills began not only to shade third quarter prices, but to undertake to anticipate some deliveries, until toward the close of the month the June steel offered at third quarter price could be considered prompt steel, and the disappearance of the prompt market, at a premium, was complete. We quote billets at \$26.50 and sheet bars at \$27, f.o.b. maker's mill, Pittsburgh or Youngstown, with buyers holding off and evidently expecting still lower quotations. An interesting feature is that the offerings are chiefly open-hearth and it might be impossible to obtain Bessemer as cheaply, whereas in the winter Bessemer was in better supply and if wanted at all could frequently be obtained at a concession from the open-hearth level.

Finished Steel

The remaining premiums for prompt deliveries of finished steel have practically disappeared, though small lots of plates sometimes bring \$1 a ton advance, and steel bars frequently command \$1 to \$2 a ton, but only in quite small tonnages. Regular prices are not shaded to any material extent, there being slight irregularities in pipe and wire. The shading of \$2 a ton in galvanized sheets noted in April has produced a regular market at 3.40 cents, and this price is now sometimes shaded. We quote regular prices as follows, f.o.b. Pittsburgh unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Common iron bars, 1.65 cents, Pittsburgh; 1.67½ cents, Philadelphia; 1.57½ cents, Chicago.

Wire nails, \$1.80 per keg, base; plain wire, 1.60 cents.

Sheets, blue annealed, 10-gage, 1.75 cents; black, 28-gage, 2.35 cents; galvanized, 28-gage, 3.40 cents; painted corrugated, 28-gage, 2.55 cents; galvanized corrugated, 28-gage, 3.50 cents.

Tin plates, \$3.60 for 100-lb. cokes.

Merchant steel pipe, ¾ to 3-in., 70½ per cent off list.

Steel boiler tubes, 3½ to 4½ in., 70 per cent off list.

Standard railroad spikes, 1.80 cents.

Button-head structural rivets, 2.10 cents; cone-head boiler rivets, 2.20 cents.

The Non-Ferrous Metal Market

The situation in the non-ferrous metal market since our last report is not greatly improved. The price for lead was raised temporarily, but reduced again without apparent reason. Spelter has been unusually dull, with prices tending downward. An upward tendency has been noted in copper, and tin has held its own.

Copper.—The market for this metal improved steadily in the early part of May. Foreign demand was greatly increased by reason of the more settled political conditions in Europe, and domestic business also was better. The last quotations are: Lake 1534, 16 cents; electrolytic 15.55, 15.60 cents.

Tin.—Domestic consumers have not shown great activity, and spot tin has been offered below the importation price. May tin is quoted at 49¼ cents, New York.

Lead.—Business in this market has been very quiet following the unexpected reduction in price by the American Smelting & Refining Company. Recent quotations have been unchanged at 4.20@4.25, St. Louis, and 4.30@4.35 cents New York.

Spelter.—This market has been without feature except the steady decline in prices and the unwillingness of consumers to purchase even at the prevailing low market. The last quota-

tions are 5.25@5.30 cents, St. Louis, and 5.40@5.45 cents, New York.

Other Metals.—The aluminium market is not active and there is little new business. Quotations are 26¼ cents, New York. Antimony has been dull, and quotations have not changed much. The last are 7.70@7.79 cents for various brands. The market for quicksilver is quiet, with prices unchanged. The New York price for flasks of 75 lb. is \$30; San Francisco, \$38.50 for domestic, and \$36 for export orders.

The Western Metallurgical Field

Varied Activity in Nevada

Since the destruction by fire of its cyanide mill over a year ago, the Florence Goldfield Mining Company has confined its activities wholly to mine exploration work in the hope of developing new ore which would warrant the rebuilding of the mill. This work has been conducted on the upper levels of the mine, and no effort has been made to search for ore on the lower levels. The net result is that sufficient ore has been found to warrant a resumption of milling operations, although no new plant will be erected at this time. Arrangements have been made with the Jumbo Extension Company to use half of its 20-stamp mill, the Bonnie Clare. Half of the mill has been remodeled, and both companies will treat their ores separately. This arrangement will permit the Florence company to treat about 1500 tons of ore monthly, and in the meantime continue mine exploration.

A new consolidation is reported to be under way at Ely, Nev., involving a combination of the Giroux Consolidated, Coppermines Company, Butte and Ely, and Chairman Consolidated. The Giroux Consolidated is reported to be the nucleus of a company with \$8,000,000 capital, in which the various companies will enter on the basis of their present stock issues and values. The Giroux is controlled by the Thomas F. Cole interests, which in turn controls the Butte and Ely. The other two properties are owned by the Gunn-Thompson company. It is hoped by the promoters of this consolidation that it will prove the means by which the Giroux may be made the success it was originally expected to be. At present one section of the Nevada Consolidated mill at McGill is devoted to the concentration of Giroux ore.

The Nevada Cinnabar Company will erect a reduction furnace on its property this summer, for the treatment of ore from its quicksilver claims. Mine development has been under way for some time past, and a sufficient tonnage of ore has been found to warrant the erection of a plant to treat 50 tons of ore per day.

Idaho

The state law of Idaho requires mining companies to file with the assessor complete reports of their production, expenses and profits from annual operations. These reports for 1912 have now been made public, and it appears that the lead mining companies in the Coeur d'Alene district made a gross production valued roundly at \$14,000,000. The total cost of extraction and treatment was \$5,678,000, and the net profit \$3,407,000. The Bunker Hill & Sullivan again holds the first rank among Coeur d'Alene companies, having net earnings slightly in excess of \$1,000,000, which is \$300,000 ahead of the next largest producer. The Stewart made the best showing compared with the record for 1911, its net earnings exceeding those of last year by 100 per cent, amounting to \$395,000 out of a gross production of \$1,471,500. The Federal M. & S. Company had a gross yield valued at \$4,645,000, but netted only \$953,387.

Minerals Separation Suit at Butte, Mont

The Minerals Separation Company, controlling patents for the concentration of ores by flotation, brought suit against the Butte & Superior Copper Company for infringement by the use of the so-called Hyde process in its mill at Butte. The case was heard in April, but no decision has as yet been handed down. The Butte & Superior company has been using flotation

for some time, having installed a process patented in the United States by James H. Hyde, who, it is claimed, is using the Minerals Separation process.

The first annual report of the Butte & Superior company has been issued, showing that ore blocked out and ready for mining and treatment amounts to 1,200,000 tons, containing 21.7 per cent zinc and 7.9 ounces silver per ton.

Effects of Depression Felt in Utah

The general depression in the Western mining industry, first felt keenly by Colorado, is now being observed in Utah, and steps are being taken to revive an interest in legitimate mining in that state. The conditions there are practically a repetition of those existing in Colorado, namely, a lack of prospecting and consequent failure to discover and open new mining districts. Interest lags among those who have already made money in mining, and the commercial clubs of Salt Lake and other cities are considering ways and means for attracting local moneyed men back to mining as a business. Salt Lake City feels the local depression the more keenly because it maintains a local mining stock exchange, in which activity has been steadily decreasing. The small investor is not attracted to mining stocks, with the consequence that few new companies are formed of local men, and the largest operations are carried on by foreign corporations which remove the bulk of their profits from the state. The general consensus of opinion is the same as that reached in Colorado, and is to the effect that if the state is to profit to the greatest extent by mining, then more local money must be expended in the industry. Mining propositions must be presented on a sane business foundation, and the public encouraged to invest on a business basis.

The May meeting of the Utah Society of Engineers was held at Salt Lake City on the 16th ult. Mr. L. M. Bailey, general manager for the Portland Cement Company, of Utah, presented a paper on the manufacture and uses of cement. On the day following the meeting, the Society visited the plant of the Utah company.

Flotation at the Inspiration

In this journal for March, 1913, editorial comment was made on the fact that the Inspiration management had abandoned further work on its proposed new concentrator until experimental work could be completed on flotation by the Minerals Separation process. Since that time, and without waiting for the decision of the suit of the Minerals Separation mentioned above, the Inspiration company has practically decided to adopt the process. The first experimental unit was of 50 tons capacity only. This has been followed by a plant of 600 tons capacity, and on the results obtained in this mill the determination to build a larger plant will be based. If the results are as favorable as expected, a 7500-ton mill will be erected. The earliest experiments indicated a saving of from 93 per cent to 94 per cent of the copper in average Inspiration ore, the concentrate carrying up to 40 per cent copper. This is an excellent result, and one believed impossible with copper ores of this character. It may be said that Inspiration is fortunate in delaying its original plans, for in case flotation is adopted the original cost of plant will be far less than would have been expended for the standard type of concentrating mill.

Company Reports

Nipissing Mines Company.—The annual report for 1912 states that the high-grade mill ran continuously and successfully through the year, treating 1752 tons of Nipissing ore averaging 2212 oz. silver per ton, and 90 tons of custom ore. Bullion shipped amounted to 4,258,641 oz. A sampling plant was added and a blast furnace installed in the refinery for the reduction of slags, flux dust and precipitate. A new reverberatory furnace was built for refining the precipitate from the low-grade mill, so that practically the entire silver product of the mine is now shipped as bullion over 997 fine.

The low-grade mill was completed in 1912, but was not in full operation until 1913. During February, 1913, the mill handled over 200 tons ore per day. A feature of the practice is

precipitation of silver on aluminium dust. The precipitate, after drying, contains 93 per cent silver, and can be melted bullion 999 fine in one operation. The total cost of the mill and accessory plant was \$324,650, and it is expected that this cost will be returned in six months. Sufficient ore is in sight to keep the mill running at present capacity for 2½ years.

Porphyry Copper Companies.—The annual reports of the Utah Copper Company, Ray Consolidated Copper Company, Nevada Consolidated Copper Company, and Chino Copper Company, have been issued, from which we take the following data, conveniently arranged for comparison:

	Ray	Utah	Nevada	Chino
Mill capacity, tons per day...	8,000	21,000	8,000	6,000
Ore reserves, tons.....	82,904,368	337,700,800	38,853,351	90,000,000
Copper in reserves, %.....	2.19	1.5	1.67	1.8
Ore milled, tons.....	1,565,875	5,315,321	2,852,515	1,120,375
Copper in ore milled, %.....	1.68	1.36	1.692	2.077
Ratio of concentration.....			9.09	16.56
Copper in concentrate, %.....	18.95	20.75	10.49	21.20
Extraction, %.....	68.28	66.32	68.25	61.63
Cost of milling, cents per ton.....	46.88	41.58		58.57
Copper produced, lb.....	35,861,496	96,175,090	63,063,261	28,684,208
Cost copper per lb., cents.....	9.83	9.02	8.86	7.69

Boston Meeting of the American Institute of Chemical Engineers.

The sixth semi-annual meeting of the American Institute of Chemical Engineers will be held in Boston, Mass., from June 25 to 28. All the professional sessions will be held at the Engineer's Club.

On Wednesday, June 25, a business session will be held at 9:30 a. m., which will be followed by the reading and discussion of the following papers:

Effect of climate on plant location, by Mr. William M. Booth.

The power plant, by Mr. Barker, of Arthur D. Little, Inc. Relation of the manufacturer to the patent system, by Dr. William M. Grosvenor.

In the evening of Wednesday a session will be held at 8 o'clock devoted to the presidential address of Dr. T. B. Wagner and a paper on general efficiency in dye houses and bleach houses by Dr. Louis J. Mathos.

On Thursday, June 26, an evening session will be held devoted to the reading and discussion of the following papers:

Depreciation and obsolescence, by Mr. Richard K. Meade.

Some peculiar functions of the retained expert, by Dr. William M. Grosvenor.

Legal control of dangers to health in factories, by Dr. Charles F. McKenna.

On Friday, June 27, an evening session will be held at 8 o'clock, devoted to the reading and discussion of the following papers:

Import duties on chemicals and their influence on the chemical industry, by Dr. F. W. Frerichs.

The drying of linseed oil and red lead with special reference to painting steel, by Dr. J. C. Olsen and Mr. A. H. Callaghan. This will be followed by the final business session.

The program of visits and excursions is very full. In the afternoon of Wednesday an excursion will be made to the works of the Hood Rubber Company, at Watertown (manufacturers of rubber shoes, automobile tires, etc.). Alternate visits have been arranged for the same afternoon to the U. S. Arsenal at Watertown and to the laboratories and experimental paper mill of Arthur D. Little, Inc.

On Thursday morning a train will take the members to Lawrence, Mass., where two parties will be formed to visit the Pacific Mills (cotton goods and prints) or the E. Frank Lewis' Wool Scouring Establishment. In the afternoon a visit will be paid to the works of the Russell Paper Company (wood pulp by soda and sulphite processes, and manufacture of book and print paper).

For Friday an all-day excursion on a special ocean-going tug-boat has been arranged as follows: The party will be first

taken to the New England Gas & Coal Company, Everett, Mass. After the inspection of this plant the party will be taken to Marblehead, Mass., where dinner will be taken at the Eastern Yacht Club. On the return trip of the boat to Boston a stop will be made at the new power plant of the Boston Elevated Railroad if time permits.

On Saturday an excursion will be made to Providence, R. I., where two parties will be formed to visit either the works of the Gorham Manufacturing Company (silversmiths and goldsmiths, plated ware, bronzes, etc.) or the plant of the United States Finishing Company (large cotton finishing plant, bleaching dyeing, printing).

Dr. J. C. Olsen, Polytechnic Institute, Brooklyn, N. Y., is the secretary of the American Institute of Chemical Engineers.

International Engineering Congress at San Francisco, 1915

In connection with the Panama-Pacific International Exposition which will be held in San Francisco in 1915, there will be an International Engineering Congress, in which engineers throughout the world will be invited to participate.

The Congress is to be conducted under the auspices of the following five societies: American Society of Civil Engineers, American Institute of Mining Engineers, The American Society of Mechanical Engineers, American Institute of Electrical Engineers, and The Society of Naval Architects and Marine Engineers.

These societies, acting in cooperation, have appointed a permanent Committee of Management, consisting of the Presidents and Secretaries of each of these Societies, and eighteen members resident in San Francisco.

Thus constituted, the personnel of the committee is as follows: For the American Society of Civil Engineers: Geo. F. Swain, president; Chas. Warren Hunt, secretary; Arthur L. Adams, W. A. Cattell, Chas. Derleth, Jr., Chas. D. Marx.

For the American Institute of Mining Engineers: Charles F. Rand, president; Bradley Stoughton, secretary; H. F. Bain, Edw. H. Benjamin, Newton Cleaveland, Wm. S. Noyes.

For the American Society of Mechanical Engineers: W. F. M. Goss, president; Calvin W. Rice, secretary; W. F. Durand, R. S. Moore, T. W. Ransom, C. R. Weymouth.

For the American Institute of Electrical Engineers: Ralph Davenport Mershon, president; F. L. Hutchinson, secretary; J. G. De Remer, A. M. Hunt.

For The Society of Naval Architects and Marine Engineers: Robert M. Thompson, president; D. H. Cox, secretary; Geo. W. Dickie, W. G. Dodd, Wm. R. Eckart, H. P. Frear.

The committee has effected a permanent organization, with Prof. Wm. F. Durand as chairman, and W. A. Cattell as secretary-treasurer, and has established executive offices in the Foxcroft Building, 68 Post Street, San Francisco.

The ten members of the committee, consisting of the presidents and secretaries of the five national societies will constitute a Committee on Participation, through whom all invitations to participate in the Congress will be issued to governments, engineering societies, and individuals. Mr. Chas. F. Rand is the chairman and Mr. Chas. Warren Hunt the secretary of this committee.

The papers presented at the Congress will naturally be divided into groups or sections. During the Congress each section will hold independent sessions.

"The scope of the Congress has not as yet been definitely determined, but it is hoped to make it widely representative of the best engineering practice throughout the world."

Importations of graphite largely exceed domestic production, according to the 1912 figures of the Geological Survey. The imports in 1912 amounted to 25,643 tons, valued at \$1,709,337; domestic production of natural graphite, 2455 tons, valued at \$207,033; production of artificial graphite, 6448 tons, valued at \$830,193.

Award of the Willard Gibbs Medal to Dr.

L. H. Baekeland

Reception and Banquet

The presentation of the third Willard Gibbs medal to Dr. Leo. H. Baekeland of Yonkers, N. Y., formed the occasion for a congratulatory banquet at the Hotel Sherman in Chicago on May 16. The arrangements for the entertainment and banquet were under the auspices of the Chicago section of the American Chemical Society. Mr. William A. Converse, founder of the Willard Gibbs medal, was present at the banquet. The medal is awarded each year to a man who has gained pre-eminence in chemical research.

Preceding the dinner was a reception, which afforded the younger chemical fraternity an opportunity to meet some of the most distinguished educators and industrial chemists of this country. About 150 ladies and gentlemen were present.

After the reception the guests repaired to the Louis XVI room to inspect the exhibit of the bakelite products which were displayed on tables. The room was specially decorated with American flags. Charts bearing graphic representations of chemical compounds, which require the use of polysyllabic names to designate them in chemical nomenclature, lent a distinct color to the meeting which was much in keeping with the profession represented at the banquet.

There was a most interesting display of the many articles which have been made from bakelite, illustrating the wonderfully manifold applications which these new infusible and insoluble phenolic condensation products have found. These applications vary from cigar holders to switchboards for battleships; from grind-stones to self-lubricating bearings; from the most effeminate jewelry to acid pump valves; from brass bedstead lacquer to phonograph records; from billiard balls to automobile magnetos; from unbreakable dolls to electrical machinery; from buttons to newspaper stereotyping matrices.

Following the banquet, Professor Julius Stieglitz, chairman of the Chicago Section of the American Chemical Society, addressed the guests. In his address, which was on "The Willard Gibbs Medal," he said that the two medals which were awarded previously were presented to pure chemists; that is, to men occupied entirely in theoretical science—Svante Arrhenius and Theodore W. Richards. This is the first time that the medal has been awarded to a man pre-eminent in applied chemistry.

Presentation Speech of Dr. W. D. Richardson

Dr. W. D. Richardson in presenting the Willard Gibbs medal to Dr. Baekeland spoke in part as follows:

"We have gathered here to-night in recognition of the services of one who has materially aided human progress, and in adding to the sum of knowledge has increased the sum of human happiness. To emphasize or memorialize this recognition we are to present our guest of honor with a medal, founded by Mr. Converse, and named in honor of Willard Gibbs, who, although known as a mathematical physicist, was

in fact one of the most illustrious chemists of this country.

"The man who accepts a medal takes upon himself a heavy responsibility. Like the man in the story, he places himself in the hands of his friends in a most absolute way. There is no relief or rest for him from the time he arrives in town until he is safely aboard a train and on his way home. There is every likelihood of his being over-entertained and over-feasted. But worst of all people inquire what he has done to merit the award and look up his record as though he were a politician of another party. And by accepting the medal and placing himself at the mercy of his friends he may even have to sit patiently and listen to a recital of some or all of his scientific labors and results and appear to be at ease. Dr. Baekeland, I hope you will be able to bear the worst that is to come with fortitude and a smile. Where the record is clear, where the work has been well and permanently done, there should be no embarrassment to the doer in the admiration of his friends. In such work a man may most properly show his pride without fear of the charge of egotism.

"The needs of the world increase with the age of the world.

It was not long ago that people got along very well without railroads and steamships, automobiles, and air ships; not long ago that they got along without the tungsten, arc, carbon-filament and mercury-vapor lamps; without gaslight and kerosene light—indeed, it was only 100 years ago that there was no stearic acid candle! and the animal and vegetable fats and oils were used for illumination. The chemical inventor of to-day fills a new want as often as he does an old one. Or he may even create a want where there was none before."

Dr. Richardson then outlined the wonderfully successful work of Dr. Baekeland in three different fields of chemistry. First, in the field of photo-chemistry he invented the developing-out paper or gaslight paper and thus revolutionized photographic print making. "Velox" was the very felicitous name applied to the invention—the same velox made by the Eastman Company to-day and known to all photographers.

Second, in electrochemistry Dr. Baekeland perfected the

Townsend cell for the manufacture of electrolytic alkali and chlorine which is now in commercial operation at the works of the Hooker Electrochemical Company at Niagara Falls, N. Y., where 34 tons of caustic soda are manufactured daily and within a year the capacity is to be increased to 64 tons per day. Recently the Farbentfabriken of Elberfeld has adopted the Townsend cell in its works.

Dr. Baekeland's last, and as he looks upon it, his most important work has been in the field of organic chemistry, and consists in the invention and manufacture of condensation products of phenol and formaldehyde. At present the product bakelite is worked up in twenty-four different industries. In the button industry alone about one thousand workmen are employed on the product.

Dr. Richardson then continued:

"But if I were asked to sum up in a few words the genius of Dr. Baekeland I should say that it is comprised in his ability to follow an idea to its conclusion. In the case of each of his discoveries he has not been satisfied with the mere discovery,



L. H. BAEKELAND

however important it was; but in every instance he has carried his work to a practical conclusion—from the laboratory to the factory and from the factory to the public. Let us see what this entails.

"In the first place a man must have faith in his idea; not only that but faith in himself. Then he must have the scientific knowledge and technical skill necessary to carry his discovery through the laboratory stages and to the small manufacturing stage. Following this the larger factory must be built and started in operation. In the meantime, if the inventor's future rights are to be protected, the inventions must be carried through the patent office—a matter of no small importance, and requiring no small amount of information. Then the company must be organized and capitalized and directed. Finally, if the inventor is to be completely successful, he must be a good publicist in order to bring his discoveries before the great consuming public. In all these departments of activity Dr. Baekeland stands pre-eminent.

"And now, having exposed thus briefly the splendid record of our guest, in this barefaced fashion, before him and you—an ordeal which he has borne with the utmost fortitude—I have only the duty left of presenting the Willard Gibbs medal.

"Dr. Baekeland: This medal can add nothing to your achievements. Your work and your inventions have already received their merited recognition by your scientific colleagues and by the world of practical affairs. But in presenting you with this medal the Chicago Section of the American Chemical Society not only desires to give recognition to your discoveries in a very special way, but more than that, to link your name with those of the famous men who have preceded you—Arrhenius, Richards, and the one after whom the medal is named, Willard Gibbs, as well as with many more who will follow you in after years.

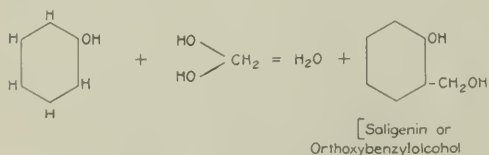
"In the name of the Chicago Section and the founder, Mr. Converse, I now present to you the third Willard Gibbs medal."

Dr. L. H. Baekeland's Address on the Chemical Constitution of Resinous Phenolic Condensation Products

Dr. Leo. H. Baekeland replied with a few gracious felicitous words of thanks, and with the modesty which is so characteristic of the man said that "friendship and good feeling often bring about an alteration of good judgment."

Dr. Baekeland then presented a long and very interesting address on "The Chemical Constitution of Resinous Phenolic Condensation Products," dealing with the purely theoretical side of the subject. Only an abstract of this address can be given here. He pointed out that the study of the chemical reactions involved in these processes is, by no means, an easy one. Bodies of a resinous or colloidal nature, which neither crystallize, nor melt, nor distill, nor dissolve, do not belong to the class of substances easy to purify nor to be studied with some hope of accuracy.

By conducting the action of formaldehyde on phenol under suitable conditions, a very clean-cut reaction takes place, giving the simplest phenolic condensation product, oxybenzylalcohol or saligenin. In this case, formaldehyde reacts as methylglycol.

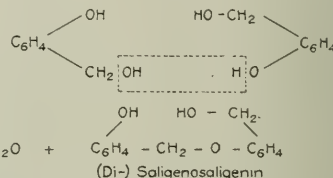


In this same reaction, more or less large amounts of para-oxybenzylalcohol are formed at the same time. But saligenin or oxybenzylalcohol is very easily disturbed by dehydration, and then changes into a resinous mass. This dehydration can occur by simple application of heat, and is much hastened by the presence of strong acids or other chemical agents. This

is the reason why in the processes of the preparation of oxybenzylalcohol, low temperature and general careful treatment are essential.

Unless such careful precautions be observed, the reaction gets beyond control and resinous bodies are sure to be produced in predominant quantity. This is especially the case when strong acids are used as condensing agents; but even under such conditions Dr. Baekeland has succeeded, in his laboratory, in establishing the presence of noticeable quantities of oxybenzylalcohol. In the process of preparation of phenol-alcohols, the conditions of procedure and necessary precautions are, therefore, widely different from those required in the production of synthetic resins.

The resinous dehydration products, derived from phenol-alcohols, are known and have been described long ago as saliretin products. Some have proposed a chemical structural formula for these products. For instance, it has been assumed that the first dehydration product, disaligenosaligenin, is formed as follows:



In the same way a trisaligenosaligenin and a heptasaligenosaligenin are formed.

However attractive all such formulas may appear to enthusiastic novices in organic chemistry, the sobering fact remains that every one of above mentioned substances is of a highly resinous nature. Whether their constitution is as simple as indicated by above formula, is a mere matter of conjecture. All that can be stated with any degree of certainty is that oxybenzylalcohol, by a process of dehydration, loses water and turns into ill-defined resinous bodies designated under the generic name of saliretins.

To try to establish nearer the structural constitution of their molecules, is out of the question. That the gradual elimination of water should decrease the fusibility of these bodies and accentuate their insolubility, seems almost self-evident to any chemist and requires no further explanation.

Dr. Baekeland discussed at considerable length the work done by many investigators on fusible soluble condensation products. All these fusible, soluble phenolformaldehyde resins described by Blumer, Delaire, Lebach, Knoll-Wetter, Bayer & Co., Aylsworth and others, have practically the same chemical and physical properties, differing only in minor details by the way in which they are prepared and by the more or less large amount of free phenol they may contain. Dr. Baekeland had proposed heretofore to designate all these relatively simple fusible soluble phenolic condensation products under the generic name of Novolac or they may just as well be called fusible saliretin products.

From different sources many attempts have been made as far back as 1902, to utilize these soluble artificial resins, as shellac substitutes, or as substitutes for other natural resins in the manufacture of varnish. In this respect they were of special interest, a few years ago, when the market price of shellac was unusually high. But it seems that none of these products has fulfilled the expectations of industrial use.

Nowadays, the main importance of these shellac substitutes resides in the fact that they bear an immediate relationship to another class of bodies, the so-called infusible insoluble phenolic condensation products, which have been designated by such names as bakelite, resinit, condensite, and which Dr. Leblat called resites.

The second part of Dr. Baekeland's address dealt with the chemical constitution of these infusible condensation products

of maximum hardness and resistivity. He pointed out that the very fact that these bodies are infusible and insoluble, and that they are characterized by a much more pronounced chemical and physical inertness than the so-called Novolak resins or saliretin products, makes it still more difficult to study their chemical constitution.

All that can be said now with some probability is this:

First, that these bodies are phenolic condensation products of formaldehyde, or of equivalent methylene-group-containing substances, this condensation process having for result a corresponding enlargement of the so-called carbon nucleus of the molecule.

Furthermore, it is a well-accepted fact that in these reactions, formaldehyde can be replaced by its many equivalents: methylal, methylenacetate, hexamethylenetetramine, the polymers of formaldehyde, paraform, etc., have all been tried with success. Even products like monochloromethylether may lend their methylene-group in reacting with phenol and form hard, insoluble condensation products.

Secondly, that after the so-called condensation has taken place, polymerization sets in, with the result that the molecules of the condensation product form by aggregation or regrouping so-called polymerized molecules of much higher molecular weight; and the chemical and physical dynamics of these enlarged polymerized molecules are proportionately lessened. This at once explains the higher specific gravity and contraction of the polymerized product as well as the greater inertness or resistivity.

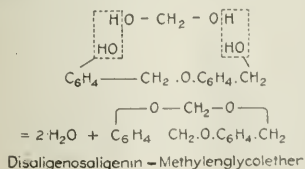
This also explains why many other aldehydes or phenolic substances may react with another and give condensation products, and yet not undergo further polymerization, and on this account, do not possess the properties of hardness, strength and resistivity, which here lend special technical value.

If we go beyond these more general theoretical conceptions and try to interpret the intimate chemical constitution of these bodies in about the same way as we are able to do with the relatively much simpler crystalline or volatile organic compounds, then our flights of fancy are justified to take as well one direction as another, because our flying machine brings us in zones where readable charts, etc., are unavailable.

Dr. Baekeland reviewed the different hypotheses which have been proposed in succession, the first one by himself in 1909, when he proposed the constitution as oxybenzylmethylenglycol-anhydride or hexasalignosalignimethyleneglycolether. This simple way of interpretation has the advantage that it accounts for the possible existence of any number of similar bodies made with a varying proportion of formaldehyde. For instance, it leaves the possibility of imagining that a product may exist which is derived from more than six molecules of oxybenzylalcohol and one molecule of methyleneglycol, which then would show less chemical resistivity. In the same way, it allows to conceive varieties of the same product with a smaller number of molecules of phenol-alcohol entering into reaction with one molecule of formaldehyde or methyleneglycol.

Numerous facts point to the existence of such modified bodies of which the chemical and physical inertness increases with the lesser amount of molecules of phenol-alcohol, which enter into reaction with one molecule of methyleneglycol; or to put it simpler, of which the resistivity increases with the larger amount of formaldehyde, which reacts with a given amount of phenol.

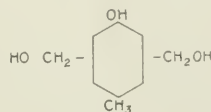
Dr. H. Lebach called attention to the existence of a condensation product resulting from the action of one molecule of methyleneglycol (formaldehyde) on disalignosalignin.



The latter corresponds, in practice, to condensation products resulting from the direct reaction of two molecules of phenol on three molecules of formaldehyde.

Dr. Baekeland then discussed at some length the hypothesis of Dr. F. Raschig, who has taken up the subject from quite a different point of view and has tried to show the possible relationship of these infusible condensation products with diphenylmethan derivatives.

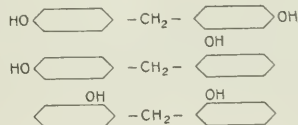
Dr. Raschig supports the opinion that the first and simplest stages in the reaction are the formation of ortho- and para-oxybenzylalcohol. He points out, however, that in case paracresol is used a well-known dialcohol phenol may be formed by further action of one molecule of formaldehyde



which again on heating sets free formaldehyde. In the case of phenol, the first resulting phenol-alcohol may react on a further molecule of phenol and give dioxydiphenylmethan.

Or one molecule of phenol-alcohol may react either with another molecule of the same phenol-alcohol, or of another phenol-alcohol, and produce, for instance, a dioxydiphenylmethan with a carbinol group, and all these reactions may occur simultaneously. Such a new phenol-alcohol may react on another molecule of phenol, and this condensation process may continue still further.

Dr. Raschig calls attention that the phenol-alcohols when reacting on phenol may produce at least three different isomeric bodies:



and that by further proceeding steps in the reaction the number of possible isomeric bodies and different compounds engendered by them becomes so limitlessly large that we know of no scientific methods which would enable us to identify them all. Commenting on this fact, he states that the amorphous character, as well as the valuable technical qualities of the end products, are probably due to the great variety of the different bodies which are all contained therein, including, at the same time, extraneous bodies like free phenol, etc.

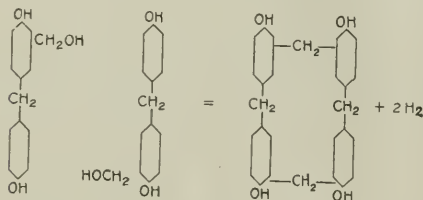
Further, Dr. Raschig seems to infer that the production of the soluble variety condensation products is merely determined by the relative amounts of phenol and formaldehyde, and he argues that in case insufficient formaldehyde be used, a mixture of the three isomeric dioxydiphenylmethans should be the result, while if more formaldehyde is present, then the corresponding phenol-alcohols of those dioxydiphenylmethans will be formed; the latter can react further upon another and in this way build up a much larger molecule which accounts for the insolubility and great resistivity of these large molecules, of which the dynamics are correspondingly decreased.

Dr. Baekeland expressed the opinion that this theoretical interpretation only holds good to a certain extent. While it is possible, and even probable, that some of the bodies mentioned by Dr. Raschig, exist in the first phases of the reaction, or may exist in the final products together with an endless variety of other chemical individuals, we know next to nothing of the relative importance thereof, whether they exist as traces or whether they play a fundamental role.

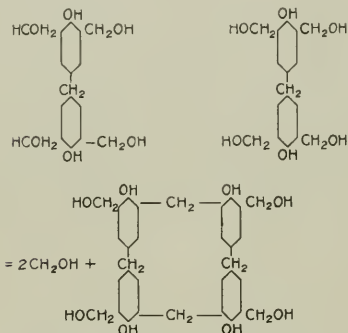
Further, Dr. Raschig seems to overlook the very important and well-established fact that the nature of the condensing agent has an enormous influence on whether fusible or infusible products will be formed. For instance, it is known that if we operate in presence of small amounts of bases, in-

fusible condensation products will be the result whether we use an excess of phenol or not.

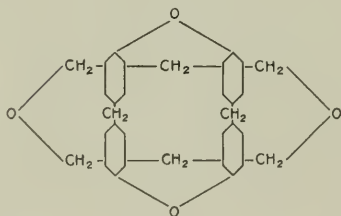
Dr. Raschig has outlined some further theoretical interpretations from the standpoint of the assumption that diphenylmethan derivatives are the main products in this process. He has shown for instance, how under certain conditions, a body may be conceived which he calls bakelite I:



and then again another body resulting from a larger proportion of formaldehyde,



which after dehydration gives a molecule which Raschig calls bakelite II.



Dr. Backeland concludes that much as these speculative considerations presented by a scientist of reputation, may be able to give food for thought, we must not forget that one hypothesis is about as easy to propose as another as long as we are unable to use any of the methods for determining molecular size and molecular constitution.

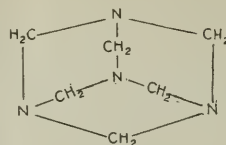
Lately Wohl has advanced a radically new hypothesis which considers these bodies as polymerization products as methylenederivates of the tautomeric phenol.

In whatever theoretical speculations we engage on this subject, we must not lose sight of the fact that the nature of the condensing agent may have an enormous influence on the way the reaction takes place, even if at the end the products may look very much alike and be more or less identical.

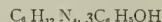
For instance, when an acid condensing agent is used the process seems to proceed very differently from what happens when alkalis are used. Then again, when ammonia is used, the reactions which succeed each other seem to be quite different. In the latter case they can be followed to quite an extent (Lebach).

For instance, every chemist knows that ammonia, on acting

on formaldehyde produces immediately hexamethylenetetramine



In the same way, if ammonia be added to a mixture of phenol and formaldehyde, the same hexamethylenetetramine is formed independently whether the phenol or the ammonia is added first or last. In presence of the phenol, one molecule of hexamethylenetetramine adds to three molecules of phenol and gives:



hexamethylenetetramintriphenol.

Under the action of heat this latter substance decomposes and sets free large amounts of ammonia gas, leaving finally a hard infusible body. But the ammonia can act immediately on new amounts of formaldehyde and repeat the same reaction so that a small amount of ammonia is sufficient to act as catalytic agent for all the formaldehyde present, and to carry on the whole reaction to the finish, after which free ammonia is found in the product.

The general reaction seems to be the same whether ammonia and formaldehyde be used in conjunction with phenol, or whether phenol be made to react directly on hexamethylenetetramine or whether hexamethylenetetramintriphenol be destroyed by heat. In the latter case, however, the so-obtained product is very porous and does not present a sufficiently homogeneous colloidal mass as required for technical purposes. On this account the use of a somewhat larger amount of phenol or other suitable similar body becomes imperative so as to obviate this defect. Hence some excess of phenol or suitable solid solvent will accomplish technical results which are not otherwise possible when hexamethylenetetramine is used alone.

Dr. Backeland showed a cylindrical tube made of bakelite which was ruptured longitudinally. The fracture was so regular that it shows that bakelite is not a heterogeneous substance.

Dean **James R. Angell** of the University of Chicago, entertained the guests by the remarks which he made concerning the points which he said he had absorbed as an experimental psychologist from Dr. Backeland's technical paper. In a more serious frame of mind he said that the attitude of the chemical departments of American universities toward industrial chemistry are too harshly criticized. He added that it is only by applying the mind to the highly theoretical phase of the science that the student can set himself on an unbiased plane furthering the development of chemistry.

Prof. **George B. Frankfurter** of the University of Minnesota, gave a short talk and expressed his pleasure at being able to attend a ceremony in honor of Dr. Backeland.

The jury of award which selected Dr. Backeland as the third Willard Gibbs Medalist comprised Prof. Alexander Smith, Dr. W. R. Whitney, Dr. E. C. Franklin, Prof. W. A. Noyes, Dr. J. D. Pennock, Prof. G. B. Frankfurter, Prof. John H. Long, Prof. Julius Stieglitz, Dr. William Brady, Mr. E. B. Bragg, Mr. S. T. Mather and Dr. G. Thurnauer.

Electroplating with Trisalz is the title of an interesting little pamphlet explaining the advantages of trisalz (a metallic triple salt) for electroplating and giving directions for preparing electrolytic baths with trisalz. Recipes are given of copper baths, brass baths, bronze baths, zinc baths, gold baths and silver baths.

Graphite for the Boiler is the title of a booklet just published by the Joseph Dixon Crucible Co. This booklet deals with no new discovery, for graphite has been sold to remove scale from boilers for many years. It simply states in a few words as possible why Dixon's boiler graphite does its work.

Evolution of Methods of Handling Slime.—III

AUSTRALIAN PRACTICE.

By H. N. Spicer.

In the limited space of a short review, such as this series of articles is designed to present, it will be impossible to give more than a very general description of the varied methods of handling slime in such a large country as Australia. Diverse metallurgical processes are employed, and the slime problem varies in the different districts. In Broken Hill, New South Wales, for example, it is concerned with the concentration of lead-zinc ores, while in Kalgoorlie, Western Australia, it relates to the cyanidation of gold ores. In one respect, however,



FIG. 1.—BROKEN HILL PROPRIETARY CONCENTRATING MILL

the problem in all the districts has a common factor, viz., the conservation of water. Over practically the whole of Australia the water question is a serious one, and especially so in the mining districts, nearly all of which are situated at the edge of the great central desert. All sorts of schemes and devices, therefore, are adopted to recover water, this being quite as important as the saving of slime, or its proper preparation for treatment.

General Conditions at Broken Hill.

Broken Hill is the largest mining camp in Australia, and generally recognized as the home of flotation processes; but despite the acknowledged leadership of Broken Hill engineers in this branch of metallurgy, it is only recently that they have adopted methods of handling slime that are recognized as modern and efficient.



FIG. 2.—TYPICAL SLIME DAM AND SETTLING BASIN AT THE BROKEN HILL PROPRIETARY

The Broken Hill mines extend for a little more than three miles along the same line of lode, forming an isolated and compact mining camp. To the casual observer it might appear that all the mills are using the same metallurgical methods, but in reality no two mills are identical in process or operation, owing to the widely varying physical and chemical properties of the ores. Some mechanical features of the practice, however, are common to all mills. Gyratory rock breakers and

rolls are generally used for crushing. Trommels are used for coarse screening, and revolving screens of the King type for finer work. For fine grinding the Australian pan, and ball and Huntington mills are usually adopted. The tube mill has been tried in more than one plant, but does not seem to have found favor. Jigs of the Hancock and May types predominate, as does the Weir-Merideth concentrating table which was produced by two of the leading engineers of the camp. The prevailing type of slime concentrator resembles a Frue vanner, and is also of local production. Hydraulic classifiers of one sort or another are in general use.

Necessity of Sand-Slime Separation.

One of the recognized requirements for successful flotation is a pulp containing little slime; and as flotation processes are widely used in Australia, it is imperative that a good separation of sand and slime be made, and equally necessary that the water be recovered from the latter. Formerly classification was done entirely by the use of spitzkasten, the slimy overflow being settled in pointed boxes, tanks or pits. Not only is it almost impossible to obtain a clear overflow by this method, but the recovery of water is incomplete, for the settled slime must



FIG. 3.—SLIME SETTLING PITS AT BRITISH BROKEN HILL

retain sufficient water to permit pumping or flowing by gravitation to the slime dams, where a further recovery of water is made.

Slime Settlement and Water Recovery.

A general view of the Broken Hill Proprietary mill is shown in Fig. 1. The first experimental plant for the treatment of slime by the Potter-Delprat acid flotation process occupies the white building in the center. At the extreme left is the lead concentrator from which slime and water are pumped to settling tanks placed on a high platform at the right near the stack. These tanks are emptied periodically, the pulp being sluiced to slime dams, one of which is seen in the left foreground. The walls of these dams, or basins, are gradually built up of slime and stand nearly vertical, as will be seen in some of the views. The Broken Hill Proprietary has many acres of such settling basins, a portion of which is shown in Fig. 2. As the area exposed is great, and the rate of evaporation high, the loss from this cause is appreciable.

At the British Broken Hill, where the Elmore vacuum flotation process is installed, large concrete pits are used as settlers. Two of these are shown in Fig. 3. As the slime flows into the pits, the supernatant liquor overflows and is returned to the mill. When a pit no longer gives a clear overflow, the slime is sluiced to a sump whence it is pumped to slime dams similar to that shown in Fig. 2. Further settlement is obtained at the dams, and clear water is pumped back to the mill.

In Fig. 4 is shown a view of the concentration and minerals separation flotation plant of the Zinc Corporation, with sand and slime dumps at the right. In the foreground can be seen ridges of slime that have been thrown up to form new settling basins for the recovery of water. An interesting feature of this

mill is the use of troughed conveyor belts for dewatering the sand tailing (contains also some slime) from the Minerals Separation boxes. The discharge end of three of these belts is shown in Fig. 5. The one at the left is used for concentrates, and the two at the right for tailings, the latter being discharged with from 16 per cent to 20 per cent moisture. The belts travel slowly, and gradually elevate the material to the discharge point, while the slime and water flow in the opposite

vators, three filter-bottom, circular sand tanks and four V-shaped slime collectors. The pulp was elevated and discharged into the circular tanks where the sand settled, the slime overflowing to the V-shaped thickeners. These thickening boxes were built of lumber imported from the Pacific Coast of the United States and were erected at a cost of about \$2,800. They were designed to handle the entire slime product of the concentrator. The overflow, which was expected to be reasonably free from slime, passed to the circular steel tank shown in the center, whence it was pumped to the storage tank shown on an elevation in the background. The method was not satisfactory, however, as the overflow from the V-shaped boxes carried considerable slime, with the consequence that the water storage tanks soon became half filled with solids.



FIG. 4.—ZINC CORPORATION BROKEN HILL MINERALS SEPARATION PLANT, SAND AND SLIME DUMPS

direction to circular settling tanks, 60 ft. diameter and 8 ft. deep. One of these is shown in Fig. 6. Here some water is recovered, while the thick slime is sluiced to sumps from which it is pumped to the settling dams shown in the foreground of Fig. 4.

As stated before, these slime dams are typical of Broken Hill. Those shown in Fig. 7 are especially interesting as showing how nearly vertical the walls can be built. The sand dumps at the right contain 1,250,000 tons of zinc tailings from the old Broken Hill South mill, and have been bought by the Zinc Corporation for treatment by the Minerals Separation process.

The huge sand dump shown in Fig. 8 is that of the Sulphide Corporation, where the Minerals Separation process is used for the recovery of zinc from the tailings of the lead concentration mill. The slime settling basins at the foot of the dump show only a part of the extensive arrangement provided for this purpose, there being also a number of large, circular collecting

The scheme was finally abandoned, and Dorr continuous thickeners were substituted for the V-shaped boxes. The first of these thickeners to be erected in Broken Hill is shown at the left in Fig. 10, with a closer view in Fig. 11. Five such continuous thickeners have been built at this mill, and today they handle the entire slime product, giving a perfectly clear over-

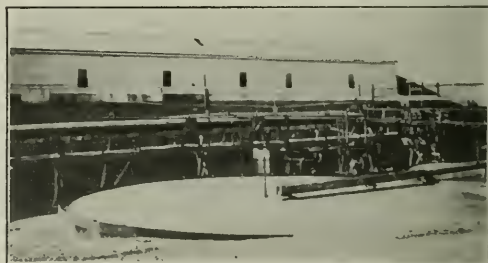


FIG. 6.—SLIME SETTLING TANKS, 60 FT. DIAMETER AND 8 FT. DEEP, ZINC CORPORATION, LTD.

flow with practically no loss of head, and a thickened underflow carrying less than 50 per cent moisture.

In Fig. 12 is shown the mill of the Amalgamated Zinc Company, using the De Bavay flotation process. This company owns no mines, but has bought a large tonnage of old zinc tailings and has contracts to treat current zinc tailings from several companies which have only lead concentrating mills.

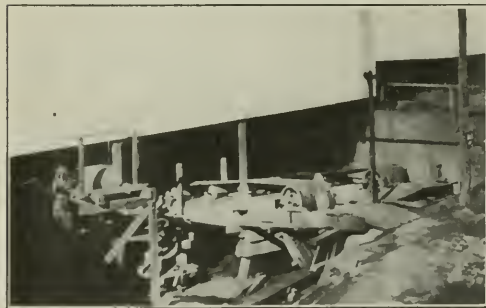


FIG. 5.—DISCHARGE END OF SAND-DEWATERING BELTS, ZINC CORPORATION, LTD.

tanks. The settled solids are dug by hand from the pits and tanks, and returned to the Minerals Separation plant for recovery of the zinc content.

Modern Methods First Adopted at Broken Hill Proprietary Block 10.

The last mill to be built in this camp was that of the Broken Hill Proprietary Block 10, a general view of which is shown in Fig. 9. It was the first to employ modern methods of dewatering slime, and commenced operations about one year ago.

That part of the plant formerly devoted to the separation of sand and slime, and the dewatering of the latter, prior to treatment by flotation, is at the extreme left in Fig. 9, but is shown more in detail at the right in Fig. 10. It consisted of six ele-



FIG. 7.—TYPICAL SLIME DUMPS, BROKEN HILL

A very elaborate system of flat-bottom, circular tanks has been installed in which to settle the slime, which is then pumped to slime dams. Close separation of sand and slime is necessary as the De Bavay process does not handle the latter material.

In concluding the review of slime handling in Broken Hill it may be stated that all forms of dewatering devices have been tried, including the Willey revolving filter, the Hunt horizontal filter and others, but until the introduction of the con-

tinuous slime thickener the problem was not successfully solved. These thickeners are now being widely adopted, with a consequent saving of water and cost of pumping.

The Kalgoorlie Goldfield, Western Australia.

In gold metallurgy the slime problem takes on a different aspect from that which it presents in the treatment of base

the plant of the Great Boulder with its sand dump and slime dams.

Corrugated Galvanized-Iron Agitator Tanks.

The settler-agitator tanks shown in Fig. 14 are at the Brown Hill mill, and are of special interest from the viewpoint of construction. They are built of corrugated galvanized iron, and were erected about 14 years ago to meet an urgent demand for an increased agitating capacity.

They were designed only for temporary use until steel tanks could be built, but have remained in service all these years, much to the surprise of all concerned. Each tank is provided with a derrick for raising and lowering the agitating mechanism. The latter being raised, the dilute slime pulp is run into the tank as long as a reasonably clear overflow is obtained.

The slime is then allowed to settle, and the clear solution is siphoned off, after which the agitators are started and gradually lowered into the thickened slime. The same type of agitator is shown in the foreground



FIG. 8.—SAND DUMP, WITH SLIME SETTLING DAMS AT BASE SULPHIDE CORPORATION, BROKEN HILL.

metal ores. The ore slime is much more valuable, particularly when, as at Kalgoorlie, the fine portion of the crushed ore is almost invariably richer than the coarse. Slime thus becomes the bugbear of the metallurgist unless his plant is properly equipped to handle it.

Until recently it was the practice in Kalgoorlie to provide



FIG. 9.—BROKEN HILL PROPRIETARY, BLOCK 10, OLD SLIME DAM IN RIGHT FOREGROUND

large settling areas in pointed boxes, for dewatering slime prior to filtration either in plate-and-frame presses or in the various forms of leaf filters. During the past year, however, the continuous thickener first adopted in Broken Hill has found

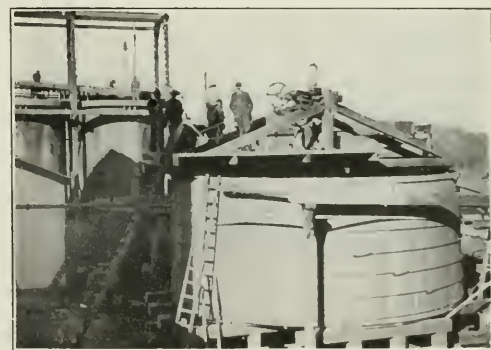


FIG. 11.—BLOCK 10, FIRST DORR CONTINUOUS THICKENER ERECTED AT BROKEN HILL. SAND TANKS AT LEFT

of Fig. 13, which is a small plant operated by the Lake View Company, for the retreatment of slime.

Retreatment of Old Slime.

It is significant of the progress of metallurgy in Kalgoorlie, particularly in the treatment of slime that old accumulations of this material which formerly could not be handled, are now being treated profitably. Two methods of reclaiming these old slime dumps are shown in Figs 15 and 16.

At the Lake View, Fig. 15, a train of cars is run onto a track laid along the edge of the dump. Chutes of galvanized iron are laid from the cars to the dump,

so that the solid slime dislodged from the top will find its way into the cars. When the angle of inclination becomes too small for the slime to run well, the dump is undercut and the track moved into a new position.



FIG. 10.—BLOCK 10, OLD V-SHAPED SLIME SETTLERS AT RIGHT AND FORMER ELMORE PLANT AT LEFT

considerable favor with Kalgoorlie metallurgists, and is rapidly being substituted for older and less efficient forms of thickening devices. At some of the mills, however, the older methods are still in use, as will be seen in Fig. 13, which shows part of

A different method is adopted at the Golden Horseshoe, Fig. 16, the slime being mixed with cyanide solution at the dump and pumped to the mill. The machinery consists of a portable truck on which is mounted a triplex pump, vortex mixer, motor and belt conveyor. The dry slime is barred down onto the conveyor which delivers it to the mixer beneath the pump.



FIG. 12.—AMALGAMATED ZINC COMPANY, LTD., USING DE BAVAY FLOTATION PROCESS

Here it is mixed with cyanide solution piped from the mill, and then pumped back into the agitators. The position of the truck is changed from time to time as the dump is removed.

Spitzkasten with Inclined Baffles.

About two years ago an interesting device for slime thick-



FIG. 13.—GREAT BOULDER SAND DUMP AND SLIME DAMS IN BACKGROUND. LAKE VIEW SLIME RETREATMENT PLANT IN FOREGROUND

ening was built at one of the large gold mines in Western Australia.¹ Experiment had apparently shown that a marked increase in the rate of settlement and density of pulp could be obtained in spitzkasten containing a number of inclined baffles placed near the surface of the slime. Accordingly several



FIG. 14.—CORRUGATED GALVANIZED AGITATOR TANKS AT THE BROWNHILL MILL

large spitzkasten were built and thus equipped; but evidently the net result was not satisfactory, for the device was abandoned in favor of continuous mechanical thickeners.

The Perseverance mill, Fig. 17, is the latest to be built at Kalgoorlie, having been rebuilt following the destruction by fire of the old plant about 2½ years ago. The principal steps in the process are dry crushing, roasting and cyaniding. Separation of sand and slime is made in pointed boxes, the slime being dewatered in cones.

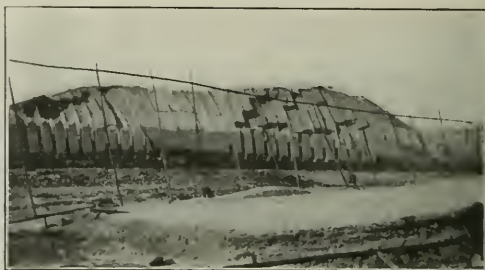


FIG. 15.—RECLAIMING OLD SLIME DUMP AT THE LAKE VIEW, KALGOORLIE

Gold Milling in New South Wales.

About 500 miles west of Sydney is the Cobar goldfield where metallurgical operations are conducted on a large scale. The Great Cobar copper mine is located here, and copper smelting has been carried on for a number of years. The Cobar company owns also several other properties, at one of which, the Cobar Chesney, copper-gold ore is produced. A 500-ton con-



FIG. 16.—RECLAIMING OLD SLIME DUMP AT THE GOLDEN HORSESHOE, KALGOORLIE

centrator has been built, equipped with rolls, Hancock jigs and Chilean mills for regrinding.

The ore is ground to pass 30-mesh screen, and dewatered in Dorr continuous thickeners to a consistency of 3¼ parts liquid



FIG. 17.—PERSEVERANCE MILL, KALGOORLIE

and 1 part solids, in which condition it is treated for the recovery of copper by the Minerals Separation process. The tailings contain practically all the gold, and today it is planned to build a cyanide plant to treat them.

The Cobar Gold 100-stamp mill, another property of the

¹See this Journal, September, 1911, p. 467.

Cobar company, treats a gold ore containing a little copper. A view of the plant, which was built a few years ago, is given in Fig. 18. It was designed to employ the Elmore vacuum flotation process for the removal of copper, with subsequent cyanidation of the tailings. The process was unsuccessful and



FIG. 18.—COBAR GOLD MILL, NEW SOUTH WALES

the plant was closed. Today it is proposed to install Minerals Separation flotation, which has been found to yield a tailing amenable to cyanidation.

Fig. 19 gives a view of the stamp mill of the Mt. Boppy mine, located about 50 miles east of Cobar. The Mt. Boppy is today



FIG. 19.—MT. BOPPY GOLD MILL, OLD SHALLOW LEACHING TANKS IN FOREGROUND

the premier gold mine of New South Wales, and the mill is the most modern in Australia. Reconstruction of the plant was in progress at the time the photograph was taken. The only feature of the ore treatment which may not seem to be in accord with modern cyanide practice is that the ore is crushed in water and not in cyanide solution. The explanation of this



FIG. 20.—OCCIDENTAL MINE AND MILL, COBAR, NEW SOUTH WALES

step is found in the fact that the ore is sufficiently acid to require a preliminary wash, which is best accomplished by crushing in water.

Some of the shallow leaching tanks in the foreground of Fig. 19 have been dismantled and converted into Dorr thickeners of twice the depth. A preliminary dewatering is accomplished in these machines, following which the final dewatering is done in a Moore filter. The discharged cake is then mixed with cyanide solution and agitated in concrete Pachuca tanks, after which it is finally filtered in another Moore filter. In this connection it is interesting to note that the Pachuca, or Brown, agitator is rarely seen in Australia, various forms of paddle agitators being generally used.

Still another important mill in the Cobar district, which has been reconstructed during the past year to conform to modern practice, is that of the Occidental company, shown in Fig. 20. The present equipment consists of stamps, card tables, tube mills, Dorr classifiers and continuous thickeners, Pachuca agitators and Moore filter.

Denver, Colo.

The Chemistry of Tungsten*

By Chas. Baskerville, Ph.D.

The classification and mechanical concentration of tungsten ores are relatively simple because of the high gravity of all tungsten compounds. However, the hydro-mechanical processes are insufficient to ensure complete separation of cassiterite, it being necessary, when this mineral is associated with the ore, to resort to magnetic separators.

There are a number of forms of apparatus used for the concentration of tungsten ores, such, for instance, as the separators of Mechernich, Johnson, Wetherill, Rowand, Knowles and King. A notable feature in tungsten ore milling is the large table area required per ton of ore treated, which reduces the tonnage capacity, according to experience in Colorado. Screen sizing has not been practiced in this country on a large scale, with a view of eliminating coarse material which might prove worthless; and it has been suggested that this procedure as well as classification be more widely considered.

The Chemical Treatment of the Concentrates.

The chemical treatment of the mechanical concentrates and the production of tungstic acid are based upon the following general principles:

(1) The alkaline carbonates attack in the dry way insoluble tungstates to form soluble alkaline tungstates.

(2) Mineral acids precipitate tungstic acid from tungsten in solution in a form insoluble in the solution.

(3) Ammonium tungstate is decomposed at redness in an oxidizing atmosphere into tungstic acid.

(4) Tungstic acid and tungstates are attacked by hydrochloric acid gas, forming the oxy-tetrachloride, easily volatile and decomposed by water into tungstic acid and hydrochloric acid.

(5) Solutions of alkaline tungstates are transformed into sulphosalts by the action of hydrogen sulphide. On decomposing the sulpho-tungstate with a dilute acid, the only slightly soluble tungsten trisulphide is formed.

Wolframite may be attacked by aqua regia, alkaline carbonates, or bisulphates. Decomposition by alkaline carbonates is the one generally used in the tungsten industry. The wolframite is fused, after pulverization, with sodium carbonate with the addition of nitrate. The fusion is done in a reverberatory furnace usually on a hearth of dolomite agglomerated with tar and burned in place. After the fusion the mass is broken up, digested with water in filter tanks with methodic washing; nearly boiling water is employed, and the alkaline tungstate dissolves along with silica and phosphoric acid as silico-tungstate and phospho-tungstate of sodium. The metallic oxides remain in the residue.

Hydrochloric acid is then added to the solution in a boiling state to aid reaction, when the neutral tungstate passes to the form of para-tungstate, $5\text{Na}_2\text{O} \cdot 12\text{WO}_3$, soluble in hot solution, but precipitated in the cold. The silico-tungstate and phospho-tungstate of sodium are more soluble and remain in the mother liquid. The para-tungstate is filtered off, washed and transformed back again to the normal tungstate by the addition of sodium hydroxide. Tungstic acid is then obtained from this solution by the addition to it, when boiling, of hydrochloric acid.

The objections to this process are that large quantities of reagents are necessary and that there are considerable losses on

*Extracts from a lecture delivered before the New York Electrical Society in the Chemistry Bldg. of the College of the City of New York. The lecture was profusely illustrated. The introductory chapters on the history of tungsten metallurgy, the production of tungsten ores, and geochemical considerations are here omitted.

account of the slight solubility of the precipitates. Therefore, Jean suggested heating with calcium carbonate and sodium chloride—a process applicable only to a very pure wolframite, since chloride silica is not eliminated; Gin heats to 1200-1300 deg. C. with magnesium chloride or fusion with potassium bisulphate (insoluble potassium acid tungstate is thus formed); while the Kempen Electrochemical Company heats sodium bisulphate to fusion with mono-hydrated sulphuric acid, then introduces the pulverized mineral. These processes are cited in passing only as examples.

Tungstic acid may be prepared from scheelite by decomposition with concentrated hydrochloric or nitric acid, forming chloride or nitrate of calcium in solution, and an insoluble residue containing tungstic acid; by fusion with potassium fluoride, producing soluble alkaline tungstate and insoluble calcium fluoride; or by conversion of tungstic acid into oxy-tetrachloride, $WOCl_4$, which boils at 227 deg. C., by means of ferric chloride, and then decomposition of this compound by means of water to form tungstic acid and hydrochloric acid.

The Preparation of Tungsten

Metallic (elementary) tungsten may be prepared by a variety of methods; as, for example:

- (1) By reduction of tungsten trioxide (tungstic acid) by means of hydrogen.
 - (2) By heating the hexachloride in hydrogen.
 - (3) By heating the trioxide with carbon, in, say, an electric furnace.
 - (4) By heating the nitride.
 - (5) By heating to redness a mixture of ammonia tungstate of the trioxide with metallic zinc.
 - (6) By the passage of an electric current in vacuo through filaments of an amalgam of tungsten, cadmium and mercury; cadmium and mercury are expelled, while the tungsten is left.
- Still other processes are the aluminothermic method, reduction with magnesium, reduction with phosphorus, electrolysis of the double chloride of tungsten and sodium, and by heating tungsten silicide with lime.

Cast tungsten may be prepared in the electric furnace by direct treatment of tungsten trioxide with carbon; reduction with calcium carbide in the presence of silica (not used industrially because the sulphur and phosphorus present are retained by tungsten carbide); or by reduction with ferrosilicon.

Sometime ago I made, during the course of an investigation of tungsten filaments for electric incandescent lamps, a study of the reduction of tungsten trioxide by means of carbon. Lack of time will only permit me to present the conclusions:

- (1) When metallic tungsten, obtained by the reduction of tungsten trioxide with carbon, is heated with carbon, the exterior portions of the metal contain carbon.
- (2) When metallic tungsten, obtained by the reduction of tungsten trioxide with carbon, is heated with an excess of carbon, the metal readily forms a definite carbide, W_2C , which possesses much the same properties as the metal, but is more easily attacked by reagents.
- (3) When a metallic compound serving as a solvent is present, the chemical action is moderated by dilution and the combination of carbon and tungsten occurs at lower temperatures, W_2C in all events resulting on heating to a higher temperature.

(4) When pure tungsten trioxide is mixed with carbon and then heated, to effect reduction to the metallic state, a carbide-free tungsten can only be obtained when the quantity of carbon used is insufficient for complete reduction and the temperature employed is below the melting-point of the metal. When tungsten trioxide is heated with an excess of carbon, or in a carbon crucible, the carbide W_2C results; the carbon in excess of that required by the formula is given up as graphite on cooling; and this graphite carbon may be burned off.

These conclusions, based upon the work of Moissan, Williams, Carnot and Goutal, Lebeau, Greenwood, Porchers, and Defacqz, as well as upon observations of my own, are of considerable practical importance to illuminating engineers, for they

naturally lead to the inquiry, "Do not the commercial preparations, obtained by the reduction of tungsten trioxide with carbon, and especially tungsten filaments for incandescent electric lamps, formed by heating tungsten with a carbonizable agglutinant, contain tungsten carbide, W_2C ?"

The physical properties of hard, brittle tungsten—the variety found in commercial lamps before the advent of the new Mazda lamp, and which were completely lacking in ductility and pliability, and were brittle and difficult to obtain with a permanent set—and those of pure ductile tungsten remind one of the differences between the tantalum of Berzelius and Moissan and that of von Bolton, since the admixture of carbon with tantalum was ascertained to explain the differences between the observations of the latter and those of his predecessors; and the comparison of non-ductile tungsten with hard and brittle tantalum becomes more relative when it is indicated that their respective atomic weights are 184 and 181.

The Manufacture of Metallic Tungsten and Ferro-Tungsten

For the manufacture of tungsten and its alloys on an industrial scale, especially for employment in the steel industry, the raw material is usually a concentrate containing from 60 to 70 per cent of tungsten trioxide. This is mixed with sodium carbonate (soda-ash) and is then roasted for about four hours; it is now treated in a dissolver, filtered, concentrated, and allowed to crystallize. Upon filter-pressing, the sodium tungstate remains in the press, and the mother-liquor is conducted into a vat for further treatment. The proper amount of hydrochloric acid is added to precipitate the yellow oxide, which, after drying, is reduced to powder metal of 99.25 to 99.50 per cent, purity, with a carbon content of 0.50 to 0.15 per cent and a density of from 19 to 19.25.

In the production of ferro-tungsten, the concentrates are often mixed with the proper proportion of steel, rod graphite, and a secret compound to assist reduction and fluxing, and the crucibles are placed in a gas-fired crucible furnace and smelted for several hours at a high heat. While, however, an 80 to 85 per cent tungsten alloy may be made by heating the powdered metal with low-carbon steel scrap in the crucible furnace, yet the higher-grade alloys, such as 85 per cent, are manufactured in the electric furnace directly from the concentrates and steel.[†]

The Uses of Some Tungsten Compounds

Tungstic oxide has been used as an oil and water color.

Large quantities of sodium tungstate are manufactured, much of which is used in fireproofing cloth for curtains and draperies, and as mordant in dyeing.

Other tungsten salts are used in weighting silks. When silks are dyed, tungsten salts, owing to their high specific gravity, are introduced to give more apparent weight to the fabric.

Sodium tungstate has almost exactly the same rate of expansion for moderate temperatures as platinum, so that it is used for sealing platinum apparatus.

Calcium tungstate, on account of its fluorescence, is used as a screen to make X-rays visible.

Tungsten compounds have also been used in glass and porcelain coloring, and in making certain bronze powders. Tungsten bronze is made by dissolving tungsten trioxide in melted sodium tungstate; the goldlike leaflets are used as a pigment. There is also a violet tungsten bronze; $K_2W_3O_{10}$ mixed with W_2O_6 .

Sodium phosphotungstate is used as a reagent for alkaloids. Cadmium borotungstate (sp. gr. 3.28 is used for the mechanical separation of minerals). Lead tungstate has been substituted for white lead in paints.

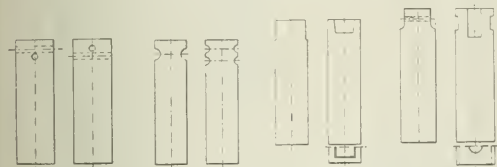
College of the City of New York.

[†]In Dr. Baskerville's lecture then followed chapters on the properties of tungsten, the compounds of tungsten, and the uses of tungsten for making special steels and tungsten alloys, and a very extensive chapter on the manufacture of tungsten filaments for electric incandescent lamps. An interesting account was given of some analyses of tungsten filaments carried out by Dr. Baskerville in which the composition of the filaments was to be determined and enough to be saved to serve as corpse details. The total weight of one of these filaments was a little over 2 milligrams. The filaments proved to be tungsten. Then followed a chapter on the uses of tungsten for crucibles, electric furnace resistors, gauge, Roentgen tube targets, thermocouples, etc.

Electrode Holder Construction for Electric Furnaces

A Description of the Principal Types of Electrode Holders for Furnaces, with Additional Notes on Joining and Dimensioning Electrodes

The issues No. 12 and 14 of March 20 and April 3, 1913, of *Stahl und Eisen* contain a very thorough and useful digest of the problem of proper construction of electrode holders, with additional notes on the joining of electrodes and the dimensioning of electrodes, from European electric-furnace practice. We herewith give a translation of this article practically in full.



FIGS. 1 TO 8.—USUAL ELECTRODE CONSTRUCTIONS

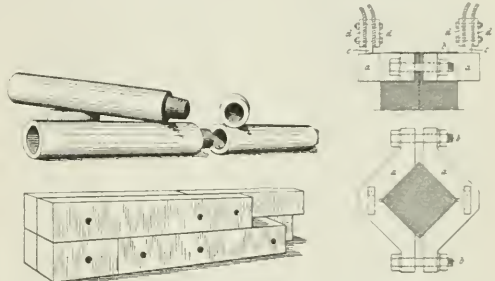
The holders in use are either topholders (Kopfassungen) or sideholders, pressing against the sides of the electrodes and allowing a feeding of the carbons, while the holders themselves remain in place.

In topholders the same contact surface is maintained between holder and electrode until the latter is used up. Topholders are almost exclusively used in the manufacture of carbide, ferrosilicon and aluminium, while in electric steel furnaces sideholders are usually employed.

Good electrodes are generally too hard for machining. Special shapes, therefore, which are often called for by the user are made by forming the material before it is baked. Figs. 1 to 10 show a number of the ordinarily used shapes of electrode ends. The recesses provide for either a good contact with the cable or for joining the ends of electrodes together so as to avoid waste.

That the proper construction of the holder and the contact is of great importance as it affects the electrode consumption and the cost of maintenance has been recognized by most manufacturers. This has led to quite a variety of constructions. The current-conducting parts are mostly made of copper or bronze, while those parts which press the contact surfaces against the electrodes are made of wrought iron, steel or steel

from coal or flue dust before putting the holder on. Elastic layers of copper netting or thin iron sheets are useful in improving the contact. Unfitted holders often cause trouble as the electrode may become red hot inside or below the holder causing it sometimes to break or resulting at least in an excessive amount by burning. Since the ohmic resistance of car-



FIGS. 18 AND 19.—JOINING OF ELECTRODES. FIG. 20.—HOLDER FORMERLY USED WITH CARBIDE FURNACES

bon decreases with increasing temperature, the current will pass preferably through the hot parts of the electrode, and thereby make the evil worse.

The following figures from Louis, *Les Electrodes en Carbone et Leur Emploi dans les Fours Electriques* (*Bull. Techn. de la Société des Anciens Elèves Des Ecoles Nationales d'Arts et Metiers*, 1910, No. 2, p. 244) illustrate the influence of the holder on electrode consumption and maintenance cost. In a large plant producing 4000 tons of ferroalloys per year the cost of maintenance of the holders was marks 6.40 (\$1.60) per ton of metal produced or marks 25,600 (\$6,400) per year. By



FIGS. 9 TO 14.—USUAL ELECTRODE FORMS. FIGS. 15 TO 17.—JOINING ELECTRODES

castings. A current density of 3 to 5 amperes per sq. cm. (10 to 32 amp. per sq. in.) is usually adopted for these contacts, the lower limit being preferred.

The surface condition of the electrode is of particular importance where a good contact is wanted. The surface of the carbon should be as flat and hard as possible and be cleared

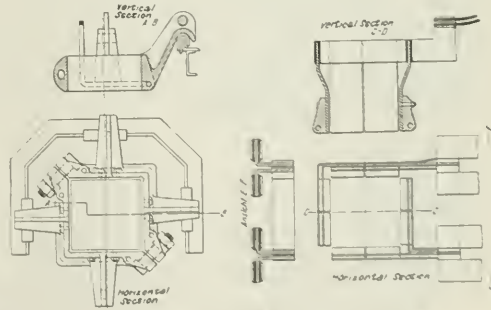


FIG. 21.—HOLDER WITH COOLING ARRANGEMENT

using an improved holder this cost was reduced to marks 1.60 (\$0.40) per ton, saving marks 10,200 (\$1,800) annually. The consumption of electrodes was also reduced resulting in an additional saving of marks 12,800 (\$3,200). The total saving due to the use of an improved holder was, therefore, marks 32,000 (\$8,000) per year, or marks 8 (\$2.00) per ton of product.

Topholders

In carbide plants the connection of Fig. 20 was formerly often used consisting of two solid V-shaped cast-iron clamps *a* tightened by bolts *b* and connected to the cables by means of necks *c* and pressure plates *d*. This solid electrode holder never proved satisfactory. When the electrode became short the holder became hot and the electrode was hot too and the cast-iron pieces often melted. Later on a cooling device was combined with this construction shown in Figs. 21 and 22.

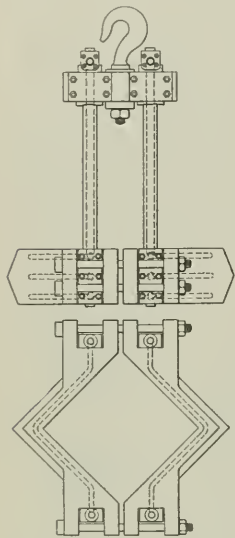


FIG. 22.—HOLDER WITH COOLING ARRANGEMENT

The holder, shown in Fig. 24, has been successful in the manufacture of ferroalloys. The electrode consists of four carbon blocks embedded in a tamped carbon mixture. Two opposite surfaces of the same block have dove-tailed recesses in order to clamp the copper plates *a* by means of the screwbolts *b* and the plates *c*. This holder is simply connected to the plates *d*

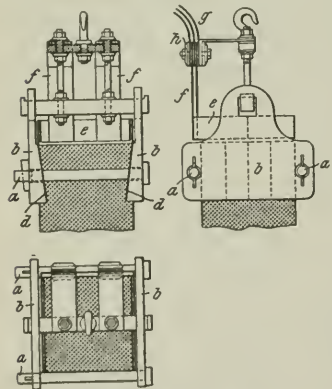


FIG. 23.—DETAILS OF HOLDER SATISFACTORILY USED IN CARBIDE FURNACE

of the current leads *e*. The plates *a* preferably consist of copper, the plate *c* of steel casting, the pressure screws of iron with square thread, and the plates *c* of bronze or of cast iron containing manganese.

If the surface of the dovetails is not smooth enough to secure a good contact, the heating at the contact will be avoided by putting a cushion of very fine strands of a copper cable

between the carbon and the copper plate. Under the pressure of the screws, this cushion of about 0.5 cm. (3/16 in.) thickness is pressed into the rough spots of the carbon and secures the passage of the electric current under favorable conditions.

When the electrode butts are to be utilized down to a length of 50 cm. (20 in.) it becomes necessary to cool the parts artificially to prevent the holder from being destroyed by the heat

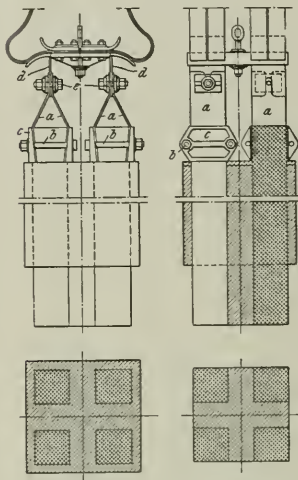


FIG. 24.—ELECTRODE HOLDER SATISFACTORILY USED ON FERROALLOY FURNACE

of the furnace. Fig. 25 shows the copper plates of the current lead being wedged to the carbon by means of water-cooled channels *a* open at the top. This design has given good results especially in furnaces with an open hearth.

The holder shown in Fig. 26 is good for cylindrical electrodes. The flexible leads are fastened to a bronze piece *a* of special form. This piece is fastened to the carbon by an electrically insulated ring and has a contact surface machined on the lathe. The holder itself is a box *c* surrounding the head of the electrode and pressed to the piece *a* by the screws *b*. After loosening the screws *b* the pieces *a* and *c* can be taken apart by a quarter turn. The piece *c* which is protected by the iron sheet *d* is put on top of the electrode, and bronze *e* is poured into the free space between the box and the carbon and on top of the electrode.

The weight of the electrode produces an excellent contact between the bronze and the walls of *c*. The holder is cooled with water which enters through *f*, and drops through the central overflow pipe onto the top of the electrode; the admission of water is regulated to make up for evaporation.

Fig. 27 shows a holder cooled by water under pressure. The ring-shaped piece *a* is hollow in the center, the water circulating through *b*. Copper sheets *c* serving both as mechanical suspension of the electrode and as current leads, are pressed against the electrodes with cast-iron or bronze wedges *d*. The wedges have regulating screws and piece *a* is held by four pressure screws.

While an intimate contact is hard to make between a carbon electrode and the cable, it is easily obtained between the cable and a metallic electrode by screwing the poleshoe at the end of the cable to the electrode. The contact surfaces of the poleshoe and of the electrode can be ground into each other, which warrants a good contact on the entire surface. As it is hardly ever possible to get the surface of a non-metallic electrode perfectly plane there will always be a certain contact resistance and evolution of heat and it may be necessary to remove this obnoxious heat of the contacts by special contact-cooling devices.

The Westdeutsche Thomas Phosphatwerke have a German patent (No. 207,361) for the construction shown in Fig. 28. The square-shaped electrodes *a* have a recess on top and a copper cap *c* is dumped over this head. In order to make good

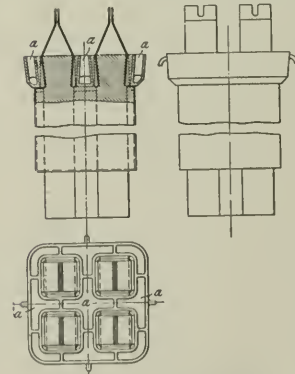


FIG. 25.—WATER-COOLED HOLDER

ways be a certain contact resistance and evolution of heat and it may be necessary to remove this obnoxious heat of the contacts by special contact-cooling devices.

The Westdeutsche Thomas Phosphatwerke have a German patent (No. 207,361) for the construction shown in Fig. 28. The square-shaped electrodes *a* have a recess on top and a copper cap *c* is dumped over this head. In order to make good

contact where the surfaces do not touch each other, the cap is filled with liquid aluminium or another metal of similarly high melting point and while the aluminium is still liquid a red hot wrought iron ring *e* is shrunk round the copper cap. During the simultaneous gradual cooling of the aluminium and the

iron ring a very close contact is produced between the copper cap, the aluminium layer and the top of the electrode.

Contact resistances cannot show up any more. The poleshoes *f*, the screws *g*, the hoops *h* and the cable ends *i* can be seen from the figure. As no soldering is required in this whole design, an important source of trouble has been eliminated.

With the holder constructions described so far, the remaining butt of the electrode, which is about 50 cm. (20 in.)

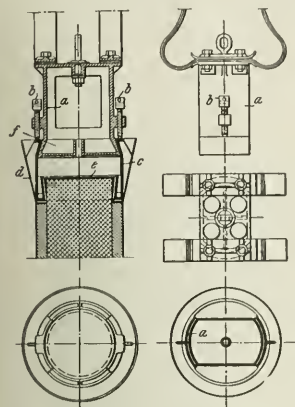


FIG. 26.—HOLDER FOR ROUND ELECTRODES

long in furnaces with an open hearth, is not used up. In furnaces with arches, as for instance in electric steel furnaces, the discard is larger, leaving a butt of about 90 cm. (36 in.) which means that only about one-half of the electrode can be used.

Movable Side Holders

In order to utilize completely the electrode material, movable holders are employed which allow the electrodes to slide through. The electrode is not held on one end, but somewhere between its two ends connection is made to the cable; when a piece of electrode is used up, another piece of carbon is added to the previous piece, and the holder raised accordingly.

Fig. 29 represents this type of connection in a design proposed by Louis. The electrode is held between two hollow pieces of metal *a* which form a pair of tongs. The tongs are pressed against the electrodes by the screw *b*. When the lower part of the electrode is consumed, it can be fed downward for the desired length. The cable is fastened at *c*; the best way of cooling the pieces *a* and the contact is by means of water under pressure.

Fig. 30 shows another tongs design for a combined electrode of several blocks. The carbon blocks are pressed by steel levers by means of the screws *a* against the central

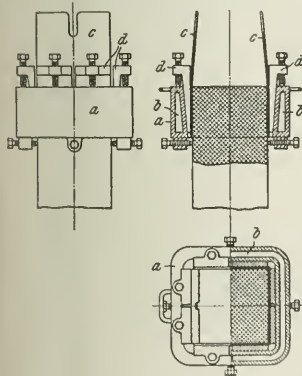


FIG. 27.—HOLDER COOLED WITH WATER UNDER PRESSURE

piece of bronze, which serves as suspension and current lead. Fig. 31 shows a similar design in which adjustable wedges are employed which are water-cooled.

The holders of most electric steel furnaces are built for complete consumption of the electrode material and have been repeatedly shown to our readers.

In the Heroult furnace each electrode is held by a bracket from a vertical rack, which is moved up and down by gear and pinion. Each carbon is surrounded by a band of metal, pieces of copper being inserted, which carry the current from the cables.

Figs. 31, 33 and 34 show various constructions used by Girod (for round electrodes), the Aktie bolaget Elektrometall Ludwika, and by Nathusius (side plates).

Joining of Electrodes

The joining of two electrodes is mostly accomplished by thread and screw, as shown in Fig. 15. Various other possibilities for making the joint are illustrated by Figs. 10, 16, 17 and 18.

The process of joining requires the use of specially shaped pieces, threadings, etc. All these cause a number of difficulties. These are overcome by the firm of Siemens Bros. & Company, by molding cylindrical, conical, or otherwise shaped recesses around the

end of the electrode, as represented in Figs. 35 to 37, and as known to our readers from the description of U. S. Patent 1,049,624 on page 217 of the April issue of METALLURGICAL AND CHEMICAL ENGINEERING.¹

The joint also causes a certain loss of voltage. This loss has been found to be 2 to 2.5 volts for square electrodes of 280 to 300 mm. (11 to 12 in.) side length with a current density of 5 amp per sq. cm. (32.25 amp per sq. in.). This would

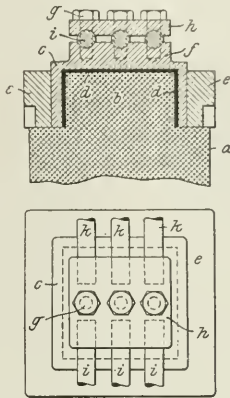


FIG. 28.—CONSTRUCTION OF WESTDEUTSCHE THOMA-PHOSPHATWERKE

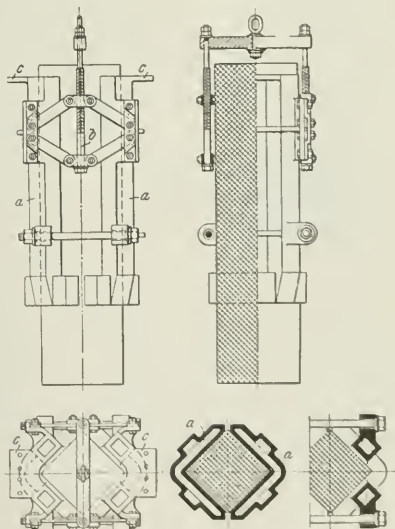


FIG. 29.—CONSTRUCTION WITH MOVABLE CONTACT

mean a drop of from 4 to 5 volts for an electric steel furnace with two arcs in series, or a loss of 16 to 20 kw. for a 500-kw. load, which represents 3.5 to 4 per cent of the total energy.

¹Concerning the method of joining employed by the National Carbon Company, of Cleveland, see the paper of Fitzgerald & Hinckley in our May issue, page 279.

securing it by a counter nut an excellent contact is obtained.

Keller, Fig. 42 (German Patent 218,054), allows the metallic conductor a certain play, the lower end being tapered thicker into the hollow top of the carbon. Liquid metal, copper or pig iron is then poured into the space left in between. This produces a mechanically and electrically reliable contact. No

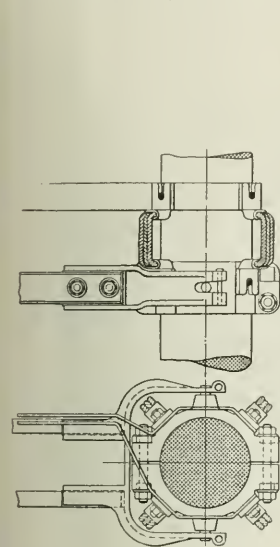


FIG. 33.—CONSTRUCTION OF AKTIE-BOLAGET ELEKTROMETALL LUDVIKA

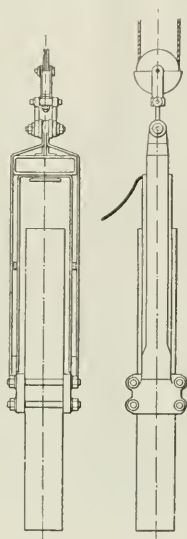
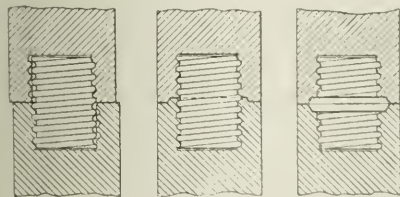


FIG. 34.—CONSTRUCTION OF NATHUSIUS

jaws, wedges or screws are required to make or improve the contact. The interior of the metallic conductor is intensely water-cooled in order to utilize as much length of the electrode as possible.

The metal has to be poured carefully into the open gap between conductor and carbon. The cavity in the electrode is first heated by a piece of red hot iron, the preheated current-conductor is then introduced and kept about 5 mm. (0.2 in.) distant from the bottom of the cavity by a small hard body. Very hot bronze or cast iron is then poured in.

Although the metal shrinks a little it makes an intimate contact. The effect of shrinkage can be entirely overcome by casting about 30 grams of tin around the contact metal after it has cooled. If the contact should not be perfect it heats up when the electrode is put to use, the tin melts and flows into the interstices and makes a perfect contact. The costs of this



FIGS. 35 TO 37.—JOINING OF ELECTRODE. FIG. 35 SHOWING THE WRONG WAY, FIGS. 36 AND 37 CORRECT METHODS

method are said to be low. The contact metal and the tin can be recovered after the electrode has been consumed.

This type of connection is mechanically strong and its electrical resistance is low. As the cooling effect reaches clear down into the bottom of the cavity in the electrode the contact will not soften even when it is heated by the proximity of the

arc. The electrode can, therefore, be introduced through the roof of the furnace and be used up to an extremely small rest piece. As no part of the metallic conductor is wider than the electrode it can pass through the same opening which fits closely around the carbon and can be packed in firebrick up to the diameter of the electrode. This method is satisfactory for single electrodes as well as for bundles of them.

The advantages of Keller's connection are:

1. It avoids all intermediary contacts between the current cable and the carbon.

2. It avoids all parts extending beyond the area of the electrode and has no brackets, screws, bolts, wedges, etc., which are liable to deteriorate and cause a high maintenance cost.

3. It requires no finishing of the electrode surfaces or of the metallic conductor. There is even a certain advantage in leaving the walls of the cavity roughly finished as it improves the contact surfaces and the mechanical strength of the joint.

4. The electrical contact is perfect as the liquid metal fills all the free spaces.

5. The mechanical connection is strong and fool-proof even when the carbon is almost used up.

6. The cost of maintenance is low

7. The electrode can be used down to below 25 mm. (less than 1 in.) length which dispenses with the cumbersome joining of electrodes.

A practical elastic connection between the electrode and the current leads is used by Keller with the above-mentioned electrodes (German patent 194,897). The successive up and down movement of the electrodes requires flexible leads between the conductor and the carbons. Sometimes an elastic cable of sufficient length serves this purpose. This, however, is subject to considerable wear and tear, takes much space, and is cumbersome for melting furnaces with vertical electrodes, especially so, when they lead the current in and out so that there is the danger of short-circuit.

Keller avoids these difficulties by using thin and very flexible leaf springs of copper, for example 0.5 mm. (20 mils) thick, bundled into two symmetrically arranged bundles and held to-

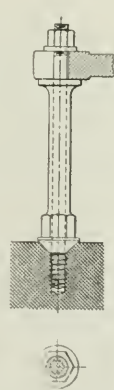


FIG. 38.—ELECTRODE WITH IRON CONNECTION IN TOP

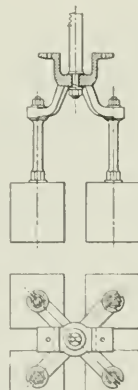


FIG. 40.—COMBINATION OF FOUR ELECTRODES

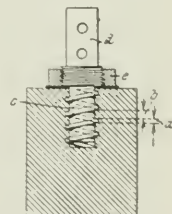


FIG. 41.—CONNECTION OF LEAVING

gether at several points by rings. Each side of the electric holder has a guiding rod which causes the leaf springs to be compressed or opened in vertical direction, when the electrodes move down and up. The shape of the leaves is determined by the distances of the rings which make knots when the springs are deflected or stretched.

If the lengths of the single lead springs are somewhat different from each other the elasticity of each spring is used with best efficiency. There is no danger of a short-circuit and the elastic connection is not exposed to the flames of the furnace. This construction has given excellent service and is used in electric furnaces of various types and employed for a variety of purposes.

Electrode Dimensions and Electrode Bundles

The best cross-section of electrodes is one of the most important questions for the consumer. Above all it is determined by the current density of the electrode. Experience is so far the only guide. A number of important papers on this subject have all been published or reviewed in this journal. No general formula which would be valid for all cases has yet been found for the calculation of this section. In general a good electrical and a poor thermal conductivity of the electrode are desirable to reduce the voltage drop and development of heat by the Joulean effect and so prevent any appreciable loss of heat through the electrodes. Much systematic work remains to be done to find the best conditions for improving the heat balance of furnaces in special cases. In view of the fact that the thermal efficiency of the electric arc furnace is 50 per cent or more, these energy losses through the electrodes are of minor importance. (The reviewer is of different opinion on this point.)

The current density of electrodes depends on the sectional area. It has been proven that round electrodes of from 70 to 100 mm. (2.8 to 4 in.) diameter, as used, for instance, in the Stassano furnace can carry 20 to 25 amp per sq. cm. (129 to 161 amp per sq. in.). (As the reviewer knows, graphite electrodes are used in these furnaces, which accounts for the remarkable difference, with the figures which follow.)

Larger sections are used in the electric steel furnaces of Heroult, Girod, Keller, etc. They cannot carry any such current density. Up to a section of 400 x 400 mm. (16 x 16 in.) 5 amp per sq. cm. (32 amp per sq. in.) are allowed with a temporary overload of 6 to 7.5 amp per sq. cm. (39 to 48 amp per sq. in.).

The large differences in the current-carrying capacity of electrodes of different section are partly due to the thermal and electrical conditions mentioned above. Most of the com-

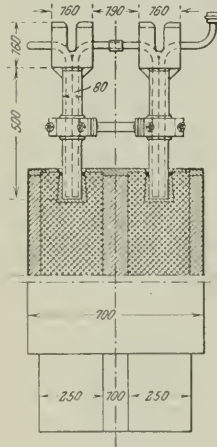


FIG. 42.—CONNECTION OF KELLER

mercial electrodes are made in presses. The capacity of the press is limited by its dimensions and design and it seems likely that the smaller sections of electrodes produced under a higher pressure, have a denser structure. From this it would seem that the current-carrying capacity of the electrode is determined by the pressing capacity of the machines.

The area of surface exposed to the cooling effect and the resistance of the electrode play, no doubt, an important part in these considerations.

It is noteworthy, however, how small the differences in the physical constants of small and large electrodes really are. For the identical carbon mixture we find the following values:

Size of Electrode	Specific resistance	Specific gravity
80 mm = 3 1/8 in. round	48 to 50 ohms	1.55
300 x 300 mm = 11 3/4 x 11 3/4 in. square	65 to 68 ohms	1.50 to 1.52

The consumer of electrodes is anxious to reduce the voltage drop by increasing the section of the electrode. The size of section is limited by the design of the furnace and by the facili-

ties of the electrode manufacturer. Recently much progress has been made so that square electrodes of 550 x 550 mm section (21.6 x 21.6 in.) and round ones of 625 mm (25 in.) diameter can now be bought from the leading factories.

These large sections naturally involve some disadvantages. The current-carrying capacity is reduced, the mechanical strength decreases with increasing section, and the weight of a single electrode reaches a considerable figure. In case of a broken electrode very heavy pieces might have to be removed

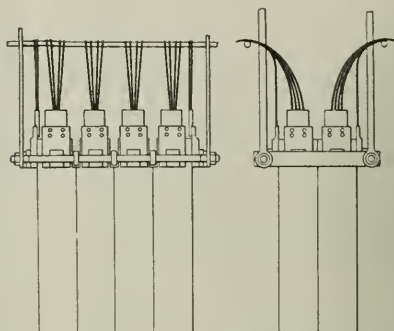


FIG. 43.—ELECTRODE BUNDLE

from a steel bath and an entire heat might be lost by carburization or by cooling off.

The carbide industry, in which large amounts of energy had to be handled at an early date, had previously chosen a different way to overcome the above difficulties. Six or eight or other numbers of smaller electrodes were combined into one package or bundle and lowered by means of cranes into the furnace. The size of the individual electrode is often chosen as 250 x 350 mm (10 x 14 in.). The electrode packages as illustrated in Figs. 24, 25, 42, and 43 leave an unused butt.

The amount of these butts is kept at a minimum by the use of proper water-cooled holders, and whatever remains is used in the carbide furnaces for patching the hearth.

The electric pig-iron furnace in Domnarfvet used three electrodes, each composed of two individual carbons: the over-all section was 660 x 330 mm (26 x 13 in.). In Trollhättan each electrode consists of four pieces of 2 mm (70 in.) length and 330 x 330 mm (13 x 13 in.) section, giving a section of 660 x 660 mm (26 x 26 in.) over all.

The combining of smaller pieces offers the additional advantage that the individual electrode has a higher current-carrying capacity and that considerable losses of energy are avoided.

Attempts have also been made to increase the conductivity of the electrode, for the sake of decreasing the section, by providing a metallic core. Heroult suggested as early as 1892 in U. S. Patent 473,117 (this journal, vol. II, p. 474) to bore holes through the electrodes and fill these with aluminium alloys or silicon alloys.

The "Planiawerke" in Ratibor, Germany, improve the conductivity and mechanical strength by metal fillets introduced before or after the baking of the electrodes (Norwegian Patent 21,366, 1910). The melting or volatilization point of the metal should be below the reaction temperature of the furnace. Measurements show that the resistance of such electrodes "reinforced" by a metallic core can be reduced to at least half of the original with a comparatively small cross section of the core.

The process is particularly intended for use with rather long not continuously fed electrodes. Certain alloys can be introduced into the steel by using same as the core. According to Perkins a tube of the carbon or iron can be filled with lime, iron oxide or other slag-forming materials and thus deliver the very liquid refining slag at the hottest point of the bath.

Copper Smelting Operations of the Santa Fé Gold and Copper Mining Co.

By Clarence T. Emrich

The property of the Santa Fé Gold & Copper Mining Company is situated about two miles above the town of San Pedro, in Santa Fé County, New Mexico, at an elevation of about 7600



FIG. 1.—COPPER SMELTER OF THE SANTA FÉ GOLD & COPPER MINING COMPANY, SAN PEDRO, N. MEX.

ft. Here are located the mine and smelter within 800 ft. of each other. A general view of the smelter is given in Fig. 1.

San Pedro is reached via Stanley on the New Mexico Central Railroad and via Los Cerrillos on the main line of the Santa Fé Railroad, a distance of about sixteen miles from the former and eighteen miles from the latter. Stanley is used as the shipping point; first, because of the nearer distance, and, second, because of a less mountainous wagon road.

Operations may be classed under four heads: Power Plant, Mining, Sorting, and Smelting.

Power Plant

The power plant consists of a boiler room, engine room, and machine shop. The boiler room contains three 150-hp return-tubular boilers and the necessary coal bunkers, feed-water pumps and heater. The engine room contains a Franklin cross-compound air compressor, size 16-27-24-14 x 18, capacity of 1100 cu. ft. per minute; one small Ingersoll-Sergeant and one Rand air compressor, both of which are seldom used; one 125-hp Nordberg Corliss driving a Root blower of 85 cu. ft. air per revolution, at an average speed of 98 r.p.m.; one General Electric Company marine generator, 110 volts, 91 amp., 450 r.p.m., used for lighting, and one Harrisburg Standard engine direct-connected to a Crocker-Wheeler generator, 75 kw, 250 volts, 300 amp. at 275 r.p.m. The machine shop contains the necessary lathes, drill presses, shavers, and, in fact, everything necessary to an isolated camp.

Coal is brought by wagons from the company coal mine, which is about sixteen miles north, at a cost of \$3.50 per ton for the haul. Water is obtained from the company well and pump house at San Pedro. This plant pumps the water about two miles through an elevation of 1000 ft. to two general supply tanks situated above the smelter site. The water contains

a great deal of lime and needs treatment with a soda-ash solution before it is used in the boilers. The corrosive effect is very heavy both in the boilers and furnace, which causes some trouble at times.

Mining is all overhead stoping and drifting. Very little timbering is done; pillars are left in the larger stopes. Hand and machine drills are used.

Sorting the Ore

The ore is a chalcopyrite occurring in a garnet formation and is very hard. The country rock is porphyry, garnet, quartzite and metamorphosed limestone and shale. The ore occurs in three horizons and all are badly faulted. The faults are both normal and reverse. They vary in width from a few inches to a hundred feet with a throw of the same dimensions.

The ore was formerly hoisted in cars on a cage, but this method was changed during the past summer to a 2-ton self-dumping skip, which gives much better results. All ore and part of the waste is delivered by tramping or gravity tramways to a 100-ton bin at the shaft bottom and from there hoisted 180 feet to the surface where it is automatically dumped into the feed bin for the picking-belt. This picking-belt was put in operation in September and has been the means of greatly improving the grade of ore received at the smelter. All ore passes over a grizzly, and the coarser ore to a Blake crusher set to 4 in. It is then elevated to the picking-belt by a belt conveyor, where eight or ten men pick out the waste. The waste is trammed to the dump while the ore drops from the picking belt to a cross conveyor belt which delivers it to the bin at the head of the gravity incline tramway. Two 2-ton skips on a three-rail track deliver the ore to the smelter bins at a cost of about three cents per ton. The smelter bins, three-rail track and shaft buildings are shown in Fig. 2.

Sampling and Analysis of the Ore

The analysis of the ore varies widely from different parts of the mine. A sample is taken by two revolving buckets which catch the ore as it falls from the picking belt to the conveyor belt, and delivers it to the sample bin, also at the head of the gravity tramway. This sample is allowed to accumulate until one of the 150-ton smelter bins is full; then the sample is lowered to the sample house near the bottom of the tramway.

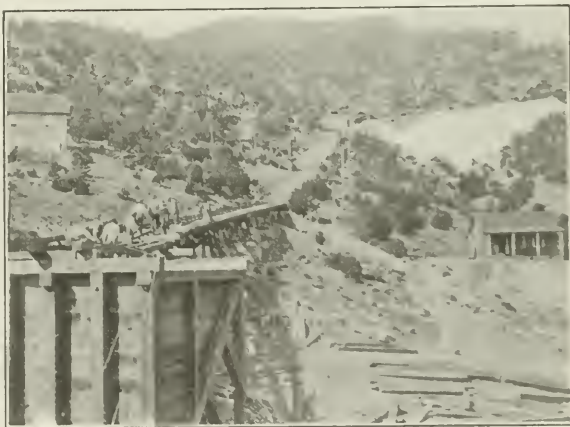


FIG. 2.—LOOKING FROM SMELTER BINS TO MINE SHAFT AND PICKING BELT

where it goes through the usual process of crushing, coning, quartering, etc., to pulp.

Analysis of the ore by treating with acids and fusion varies widely as is seen by the tables below. The treatment with acids gives an erroneous result, due to the garnet which is insoluble and holds up as much as 50 per cent of the SiO_2 , Fe

and CaO. The following analyses by acids and fusion were made on the same samples of ore:

Sample No.	WITH ACIDS			FUSION		
	SiO ₂	Fe	CaO	SiO ₂	Fe	CaO
1	65.7	8.8	10.5	36.2	14.0	22.9
2	59.3	13.2	8.8	35.9	20.6	19.6
3	60.5	9.2	10.8	36.8	18.4	21.3
4	60.7	11.7	9.1	43.5	15.1	16.3
5	54.8	6.8	13.8	41.2	10.0	20.1
6	58.1	8.1	11.4	40.8	11.9	18.5
7	65.0	6.9	8.7	34.4	11.5	23.5
Copper Average			4.2			
Sulphur Average			3.7			

The above results are representative of different bins on consecutive days, showing the variety of ore from the mine and the uselessness of the acid analysis so far as the metallurgist is concerned. The past records at this plant show very few analyses by fusion, the acid decomposition being generally used. This, together with the fact that furnace charges were never figured, partly accounts for the frequent "freeze ups" and small tonnage in the past. A comparison of results is given later.

Good Results from Obsolete Type of Furnace

The furnace used here is very much out of date, though good results can be obtained by watching it closely. It is composed of twelve jackets, six of which are steel side and end jackets, and six cast-iron bottom jackets. The side jackets are 9 ft. x 5 ft.; the end jackets are 9 ft. long by 4 ft. 5 in. on the bottom and 6 ft. 5 in. on top. The jackets are straight, with five tuyere holes in each side jacket the same diameter as the tuyeres themselves, which is 6 in. There is no bosh whatever, so when the jackets are in place all the tuyeres point upward to some extent, which helps the fire to the surface of the charge to a greater or less degree. The lower cast-iron jackets are 12 in. high and set on 2-in. bottom plates which are held in place by screw jacks. The intermittent tap is used on the side of the furnace. The two front lower jackets allow the spout to be bolted to the bottom plates. On top of the steel jackets rests an 18-in. fire-brick wall, 2 ft. high to the bottom of 2-in. cast-iron plates or slips which are on an angle of about 45 deg. This also cuts down the furnace shaft, as the furnace is seldom filled above the top of the jackets. This gives a shaft of about 7 ft.

Improved Water-Cooling for Jackets

During previous smelting operations the water always ran hot. There is no doubt about this when the jackets are seen for they are all very badly warped, due to improper cooling, which, by the system of piping that was in, could not be helped. A 4-in. supply pipe reached the furnace on one end from which point it served only three sides of the furnace. On both ends of the furnace (it is tapped on the side) two 2-in. pipes extended from each side of the 4-in. supply pipe to a branch tee from which six 1½-in. pipes supplied half the furnace. The overflow pipes were closed to within ½ in. with scale. These were all removed and the 4-in. supply pipe extended around the furnace, giving a direct flow to each jacket. There has been no trouble from boiling water since. Overflow pipes empty into a 4-in. pipe at the four corners of the furnace from whence it goes to a general sump tank where all overflow water is collected from the plant. From here it is pumped to a cooling tower, an elevation of about 100 ft. This tower has a receiving tank beneath which furnishes the pressure for the furnace where the water is again returned to the tower. Fresh water is supplied from the high pressure tanks.

The lining for the furnace bottom is composed of a layer of old broken brick to a depth of about 2 in. On this is tamped a mixture of damp flue-dust, coke-dust and adobe to a depth of about 6 in., sloping toward the tap hole where it is 3 in. thick. This bottom gives very good results, is cheap, quickly put in and takes no longer to dry out than a brick bottom. Matte occasionally eats through but is quickly stopped and has not caused any bad results to the furnace plates. The

furnace breast is bricked up with two courses of fire-brick placed on edge. This breast is found very convenient at times, when we have no sulphide to hold the matte, except the little sulphur in the ore, and the matte is partly reduced to metallic copper, thus building up the furnace bottom. Then we find ourselves breaking through the brick breast to tap as high as possible under the jackets.

Our settlers are 7½ ft. by 4 ft. by 4 ft. 4 in. outside. These we line with fire-brick on edge in bottom and on the sides. The sides are lined up about 2 ft. with the fire-brick, red brick and bats being used for the remainder. The settlers are tapped on the sides for matte, which drops from a short spout to a truck car on which are five moulds each holding about 150 lb. of matte. The slag runs off in a spout to 5-ton pots. These pots are drawn to the dump by a cable and small hoist which is located near the end of the dump. In summer the hoist is operated by compressed air; in winter by steam.

Results of Former Operations

The method of feeding under former management is seen at a glance from the following data which are taken from the best month's record during 1907:

Charges Ore	Charges Iron Ore	Charges Limestone	Charges Flue Dust	Charges Coke
43@ 2600#	21@ 400#	22@ 400#	7@ 1400#	50@ 400#
36@ 2600	18@ 400	18@ 400	11@ 1500	47@ 400
42@ 2600	42@ 200	42@ 400	21@ 1500	63@ 400
39@ 2600	5@ 500	34@ 500	34@ 1500	73@ 350
38@ 2600		38@ 500	19@ 1500	57@ 350
46@ 2600	4@ 500	42@ 500	23@ 1500	69@ 350

This gives a good idea of the method used. If the furnace froze it was cleaned out as quickly as possible and blown in again on the same charge. The records show very small deviations from the above. As no charges were figured at all the furnace was run by guess, with frequent freeze ups.

Some of the slags and mattes produced during the same period on consecutive days are as follows:

SiO ₂	FeO	CaO	Cu Matte
49.1	25.9	18.8	61.6
46.5	25.1	20.3	58.3
47.1	24.2	22.8	60.2
44.0	27.9	19.9	62.7
45.0	19.2	28.2	54.8
42.4	18.8	29.5	55.0
44.0	20.1	27.4	59.8
41.9	25.3	27.3	66.8

The correctness of these slag analyses is not known. More or less metallic copper was made.

Results of Present Operations

Following are some results we have obtained during the present operation under the same conditions with the same fluxes:

Charges Ore	Charges Iron Ore	Charges Limestone	Charges Flue Dust	Charges Coke
58@ 1800#	58@ 170#	58@ 180#	58@ 150#	58@ 285#
60@ 1800	60@ 170	60@ 180	60@ 150	60@ 285
76@ 1800	76@ 170	76@ 180	76@ 150	76@ 285
77@ 1800	77@ 150	77@ 200	77@ 150	77@ 285
85@ 1500	85@ 100	85@ 350		85@ 280
78@ 1650	78@ 100	78@ 350		78@ 280
75@ 2200		75@ 125	75@ 125	75@ 320
62@ 2200			62@ 200	62@ 320
72@ 2200		72@ 125	72@ 200	72@ 320

Slags under the same dates give the following results:

SiO ₂	FeO	CaO	Cu Matte
40.3	23.6	29.1	
41.1	23.6	29.2	
37.5	25.1	28.0	
38.8	23.2	28.4	70.4
46.5	21.2	31.9	74.4
37.1	20.8	33.5	78.8
39.4	17.3	33.2	62.0
38.0	17.8	35.2	62.9
38.7	17.8	34.4	70.0

Slag Average	Cu	S	SiO ₂	FeO	CaO	Al ₂ O ₃	MgO
September..	0.60	0.2	37.7	17.0	33.1	9.5	0.7
October..	0.69	0.15	38.7	20.6	29.5	8.2	0.7
November..	0.53	0.21	37.8	22.2	29.9	6.9	0.7
December..	0.53	0.31	37.5	21.3	32.5	6.2	0.9

Note should be taken of the copper slag loss due to small settler and intermittent tapping.

Analysis of Fluxes

	SiO ₂	Fe	CaO	S
Iron Oxide..	9.5	60.0		
Lime Rock..	3.0	1.0		
Leadville Sulphide	3.0	40.0		50.0

Leadville sulphide has been obtained only during the past few months which allows us to make a 50 per cent. matte and reduce the slag loss. This is quite an item of saving.

Feeding the furnace is all done by hand. The ore cars which carry the ore and fluxes from the individual bins are trammed by hand to the charge floor where they are dumped on plates, one on each side of the furnace. From here two men shovel it into the furnace. Before we obtained Leadville sulphide two charges were fed at a time, one on each side of the furnace. We tried this method at first with the sulphide, by mixing the sulphide to give about a 50 per cent matte. We found this gave us only from 18 per cent to 20 per cent of the sulphur, for our charge in the furnace is usually hot on top. Now the charge is allowed to go down about 2 ft. when enough sulphide for four charges is put in all at once and covered with four charges of ore and flux as quickly as possible. This method gives us about 30 per cent sulphur. Our flue-dust is drawn from the dust chamber, mixed with about 20 per cent water and fed before the ore. This is weighed to give 200 lb. net and is very satisfactory. Our blast pressure averages 14 oz. of mercury.

Our matte is weighed at the end of each shift and then piled away to be shipped.

Cost of Labor and Transportation

Shifts in the mine, picking belt and surface gang are 10 hours while those in the smelter and engine room are 12 hours. Mexican labor is used entirely.

Smelter		Mine and Other	
Feeders..	\$3.30	Machine men..	\$3.00
Tappers..	3.50	Machine helpers	2.50
Helpers..	2.70	Muckers..	1.75
Spout men	2.25	Blacksmith..	4.00
All others	2.10	All general } Labor	1.75

All hauling by wagon from Stanley costs \$4.00 per ton. We now have a Rumely oil traction engine drawing two 10-ton trucks and an American-La France Motor Truck. These have reduced the cost of hauling to \$1.50 per ton.

San Pedro, N. M.

The First Electric Steel Casting Plant on the Pacific Coast.—The Washington Iron Works, of Seattle, Wash., manufacturers of mill, mining, and marine machinery, are installing a three-ton Girod electric furnace in their new foundry, together with complete steel foundry equipment for the manufacture of high-grade castings. This company makes a specialty of logging and hoisting engines. This will be the first electric steel casting plant on the Pacific Coast.

The Colorado School of Mines senior class is taking the annual trip of mining and metallurgical inspection, having left Denver on April 21, and planning to return on May 21, two days prior to commencement exercises. The itinerary will include prominent mining and metallurgical districts in Colorado, Utah and Montana. Seven members of the faculty and seventy students comprise the party, which is traveling in a special train.

Note on Cold-Junction Corrections for Thermocouples

By Paul D. Foote

Ordinarily, in the laboratory use of thermocouples the cold junction is kept at a standard reference temperature such as the melting point of ice, while the hot junction is placed within the furnace or substance the temperature of which is desired. However, for technical purposes and general commercial use, it is not always convenient to have an ice box at hand. For this reason it is desirable to work with the cold junction at the temperature of the room. This necessitates a knowledge of the corrections to apply to the indicator readings, provided that the standardization of the pyrometer were carried out with the cold junction at some temperature other than that used in the technical works. The importance of these corrections is realized when one considers the large range of the cold-junction temperature, which in practice frequently varies from 10 to 100° C.

In order to obtain the temperature corresponding to any

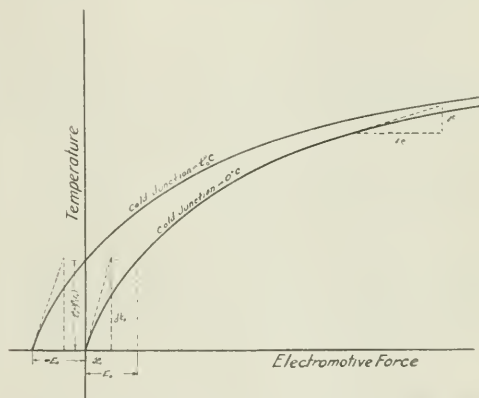


FIG. 1.—ILLUSTRATING THE SHIFT OF THERMOCOUPLE CURVE WITH A CHANGE OF THE COLD-JUNCTION TEMPERATURE

value of the emf observed when the cold junction is at a temperature of t_0 ° C. from the relation $e = f(t)$ computed on the basis that the cold junction is at 0° C., it is only necessary to add to the observed emf the value of the emf developed when the cold junction is at 0° C. and the hot junction is t_0 ° C., a number which is independent of the temperature of the furnace. However, many pyrometer galvanometers are calibrated to read temperature directly and have no scale of emf recorded. One must then express the correction in terms of degrees, and this quantity for any given value of the cold junction will depend, of course, upon the temperature of the hot junction, for the reason that in general the relation between temperature and emf is not linear. A table, accordingly, might be prepared stating the corrections to be applied to the temperature observations for all values of the hot-junction temperature and all variations in the temperature of the cold junction.

The usual method of obtaining the cold-junction correction consists in multiplying the value of the cold-junction temperature by certain empirical factors. R. Vogel¹ computed these factors for a Heraeus (Pt, 90Pt-10Rh) couple having a cold junction of 50° C. Offerhaus and Fischer² have calculated the factors for several types of couples.

This method of correction for cold-junction temperatures may be derived in the following manner:

¹R. Vogel, ZS f. Anorg. Chem., 1905, 45.

²Cornelis Offerhaus and F. H. Fischer, Electrochem. and Metall. Ind., Sept., 1908, pp. 361-364.

Derivation of the Slope Correction.—Let e and t be the observed emf and corresponding temperature when the cold junction = 0° C.

e_1 and t_1 = the observed emf and temperature when the cold junction = t_0° C.

For any thermocouple

$$e = f(t) \quad (1)$$

then for any value e_1

$$e_1 = f(t_1) \quad (2)$$

But $e = e_1 + e_0$ where $e_0 = f(t_0)$ = emf developed when the hot junction = t_0° and the cold junction = 0° C.

Substituting in (1)

$$e_1 + e_0 = f(t) \quad (3)$$

From (3)

$$t = \phi(e_1 + e_0)$$

From (2)

$$t_1 = \phi(e_1)$$

Denote by p the correction to add to the observed temperature when this value is read from the plot of a couple calibrated with its cold junction = 0° C.

$$p = t - t_1 = \phi(e_1 + e_0) - \phi(e_1)$$

$$= \phi(e_1) + e_0 \phi'(e_1) + \frac{e_0^2}{2} \phi''(e_1) + \dots - \phi(e_1)$$

= $e_0 \phi'(e_1)$ provided e_0 and t_0 are small in comparison with e_1 and t_1 .

$$\text{hence } p = \frac{e_0}{\phi(e_0)} \phi'(e_1) \phi(e_0) = \frac{e_0}{\phi(e_0)} \frac{dt_1}{de_1} t_0$$

$$= \frac{de_0}{dt_0} \frac{dt_1}{de_1} t_0 = \left(\frac{de}{dt} \right)_0 t_0 \text{ approximately.}$$

This equation, therefore, states that in order to obtain the true temperature (t) it is necessary to add to the observed temperature (t_1) the quantity obtained by multiplying the temperature of the cold junction (t_0) by the ratio of the slopes of the calibration curve at the origin and at the temperature (t_1).

The derivative $\left(\frac{de}{dt} \right)_0$ is but approximately equal to the

quantity $\frac{e_0}{\phi(e_0)}$ and becomes less so the farther t_0 departs

from zero. As seen from Fig. 1, $\frac{e_0}{\phi(e_0)}$ is more exactly the mean slope of the calibration curve from 0 to t_0° C. Using this value the single term slope correction becomes:

$$p = \left(\frac{de}{dt} \right)_0 - t_0 \cdot t_0.$$

The mean slope of 0 to t_0 may be determined accurately enough by assuming some value of t_0 in the neighborhood of the actual value of the cold-junction temperature, such as 30° C., and computing the mean slope from 0° to this point, i.e., the value of the emf observed when the cold junction is at 0° and the hot junction 30° divided by 30 . This form of the cold-junction correction requires but a single measurement of the emf at room temperature, whereas the usual form of correction practically necessitates a knowledge of the curve at several points in order to evaluate the slope at 0° C. The three forms of cold-junction temperature correction may be summarized thus:

$$\text{Form I } p = e_0 \phi'(e_1) + \frac{e_0^2}{2} \phi''(e_1) + \dots + \frac{e_0^n}{n} \phi^n(e_1) + \dots$$

$$\text{Form II } p = \left(\frac{de}{dt} \right)_0 t_0$$

$$\text{Form III } p = \left(\frac{de}{dt} \right)_{0-t_0} t_0$$

Form I is the correct expression of the cold-junction correction.

Form II is the approximation in general use.

Form III is the approximation which should be given preference over Form II.

Magnitude of the Cold-Junction Correction.—Fig. 2 illustrates the magnitude of the true cold-junction corrections for P_3 a (Pt, 90 Pt—10 Rh) Heraeus couple having for its temperature-emf relation the equations:

$$e \times 10^4 = 56.04t + 0.05434t^2 \text{ } 0 \text{ to } 100^\circ \text{ C.}$$

and

$$e \times 10^4 = -2820 + 80.71t + 0.01694t^2 \text{ } 300 \text{ to } 1500^\circ \text{ C.,}$$

where e is expressed in millivolts. These curves represent the true correction, Form I, for four observed hot-junction temperatures, and with the cold-junction temperature varying from 0 to 100° C. It is seen that the amount to be added to the observed temperature may reach a value as high as 60 or 65 Centigrade degrees.

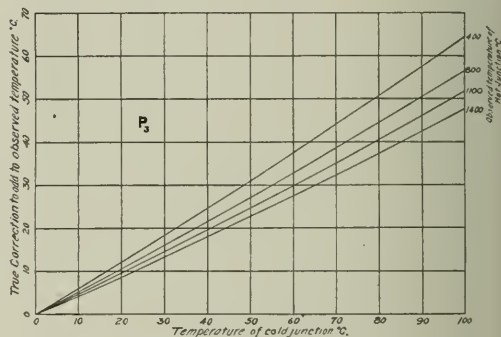


FIG. 2.—TRUE CORRECTION VS. COLD-JUNCTION TEMPERATURE

Errors in the Slope Corrections.—How closely Forms II and III express the true corrections for the cold-junction temperature depends upon the form of the function $e = f(t)$. Inasmuch as the series in Form I is frequently only slowly convergent a more convenient method than evaluating each term was used for determining the errors in the approximation corrections.

Several of the Bureau of Standards' thermocouples were calibrated (cold junction = 0° C.) from 0 to 1500° C. Assuming the temperature of the hot junction so regulated that any definite value of the observed emf developed by the couple remains constant whatever the temperature of the cold junction, the emf's for a cold junction 0° C. were obtained by adding to the observed value the emf readings between 0 and 100° C. The temperatures corresponding to these new emf's were computed and from them was subtracted the temperature corresponding to the observed emf. The differences are the true corrections to be applied to the observed temperature for a series of cold-junction temperatures. This operation was repeated for several different observed values of the emf. The slope corrections were then computed for the same points.

Fig. 3 presents the errors met with in the application of the slope corrections, Forms II and III, to P_3 (Pt, 90 Pt—10 Rh). The upper group of curves shows the amount to be added to the slope correction, Form II, to obtain the true correction. The largest error is 5° , with the hot junction at an observed temperature of 400° and the cold junction 100° C. The lower group of curves represents the corrections to apply when the slope correction is computed on the basis of the mean slope from 0 to 30° instead of 0° C., Form III. The errors are all

somewhat smaller in this case, the maximum being 3°, and hence, in general, the correction according to Form III is to be preferred.

Similar curves were obtained for Pt. a Johnson and Matthey (Pt, 90 Pt—10 Rh) couple having the equations:

$$e \times 10^4 = 63.86t + 0.07343t^2 \quad 0 \text{ to } 100^\circ \text{ C.}$$

$$e \times 10^4 = -4934 + 99.06t + 0.01594t^2 \quad 300 \text{ to } 1500^\circ \text{ C.}$$

Fig. 4 illustrates the errors found in the use of the slope

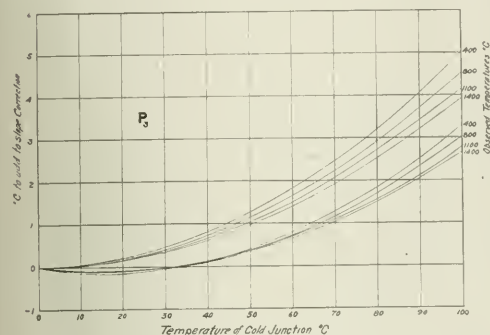


FIG. 3.—ERROR IN SLOPE CORRECTION

corrections with W_1 . This couple, which was obtained from Pellin, and supposed to have the composition (Pt, 90 Pt—10 Ir), had a temperature-emf relation expressed by the equations:

$$e \times 10^4 = 84.76t + 0.1603t^2 \quad 0 \text{ to } 100^\circ \text{ C.}$$

$$e \times 10^4 = -6260 + 130.2t + 0.03381t^2 \quad 300 \text{ to } 1500^\circ \text{ C.}$$

The maximum error found for Form II is about 10° with the hot junction at an apparent temperature of 400° and the cold junction 100° C. The lower group of curves is computed on the basis of the preferred form of the slope correction, Form III. The maximum error in this case is 6°.

For a copper-constantan couple of Adams and Johnston³ having the equation:

$$e \text{ (microvolts)} = 38.105t + 0.0442t^2 - 0.00002856t^3$$

applicable over a range 0 to 350° C., the ordinary slope correction, Form II, is an error from 0.1° to 0.5° when the cold junction varies from 0 to 20°. With a cold junction of 20 to 50° this slope correction is too small by amounts running from 0.5 to 2°. Form III applied to this couple shows corrections somewhat nearer those given by the true correction term, Form I.

Applicability of Slope Corrections.—If the couples examined are typical of all (Pt, 90 Pt—10 Rh), (Pt, 90 Pt—10 Ir), and copper-constantan couples, one may conclude that for values of the cold junction from 0 to 50° C. in the case of the platinum-platinum alloy couples, and from 0 to 20° C. in the case of the copper-constantan couple the slope correction, Form III, is fully applicable, and is not in error by amounts greater than the precision attainable with the ordinary pyrometric galvanometer. For all technical work it is sufficient to add to the galvanometer temperature reading the number obtained by multiplying the ratio of the mean slope from 0 to t_0 and the slope at the observed temperature (t_1) by the temperature (t_0) of the cold junction.

The ratio of the slopes are given in the form of numerical factors for thermocouples tested at the Bureau of Standards, and to all technical purposes are alike for similar couples. Their determination requires only a rough temperature-emf calibration made independently of the pyrometer indicator. Instead of a table of figures, the observer need remember but two or three numbers applicable over all ranges of the cold-junction temperature. Thus in the use of P^3 with any type of galvanometer indicator it is sufficient in ordinary pyrometric

work to correct for the cold-junction temperature, t_0 ° C., by adding to the observed temperature, t_1 ° C., the amount $0.6t_0$ from 400 to 700° C. and $0.5t_0$ from 700 to 1400° C., where t_0 is less than 50. If closer accuracy is desired, a number of slope values computed to another significant figure, as given in Table I, second column, or preferably the third column, must be used, and the corresponding corrections shown in Fig. 3 applied. For the highest attainable accuracy a potentiometer is required, and the emf corresponding to the temperature of the cold junction is added to the emf reading of the couple.

TABLE I.—TEMPERATURE CENTIGRADE VS. COLD JUNCTION FACTOR.
P₂ (Pt, 90 Pt—10 Rh)

Observed Temperature °C.	$\left(\frac{de}{dt}\right)_0 + \left(\frac{de}{dt}\right)_1$	$\left(\frac{de}{dt}\right)_{0-30} + \left(\frac{de}{dt}\right)_1$	$\left(\frac{de}{dt}\right)_{30-40} + \left(\frac{de}{dt}\right)_1$
400	0.59	0.61	0.63
500	.57	.59	.61
600	.55	.57	.59
700	.53	.55	.57
800	.52	.54	.56
900	.50	.52	.54
1000	.49	.50	.52
1100	.48	.49	.51
1200	.46	.47	.49
1300	.45	.46	.48
1400	.44	.45	.47

The second column shows the factor by which the value of the cold junction temperature must be multiplied when using the old type slope correction, Form II. The third column contains the factors computed from the preferred form of the slope correction, Form III. The fourth column shows the factors when the couple is calibrated with the cold junction at 30° C. and used with a cold-junction temperature not far from 40° C.

Slope Correction When the Couple Is Calibrated with the Cold Junction at a Temperature Other Than 0° C.—Many thermocouple pyrometers are so constructed that it is impracticable to calibrate them with a cold junction at 0° C. Suppose the standardization were carried out with a cold junction of $t_0 = 30$ ° C. and it is desired to use the couple near a furnace where the room temperature is $t_0 = 40$ ° C. Proceeding by a method similar to that shown in the derivation of the ordinary slope correction we now obtain the correction term:

$$p = \frac{e_0}{t_0 - t_1} \phi'(e_2) (t_0 - t_1)$$

where $\phi'(e_2)$ is the slope at the point corresponding to the

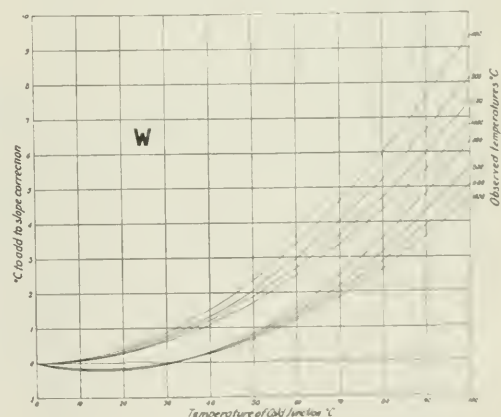


FIG. 4.—ERROR IN SLOPE CORRECTION

observed emf, and e_0 = the emf developed when the cold junction is at the temperature t_0 and the hot junction t_1 , or

$$p = \left(\frac{de}{dt} \right)_{30-40} (t_0 - 30)$$

where t_0 is not far from 40, t_1 being the observed temperature, corresponding to e_1 .

³L. H. Adams and J. Johnston Am. Jour. Sc., Vol. 33, June, 1912, p. 534.

This equation, therefore, states that in order to obtain the true temperature, t , it is necessary to add to the observed temperature, t_o , the quantity obtained by multiplying the difference in the temperature of the cold junction used in the standardization, $t_o = 30$, and during any observation, $t_o = 40$, by the ratio of the mean slope of the calibration curve from 30 to 40° and the slope at the observed temperature. The mean slope from 30 to 40° is obtained by dividing the emf observed when the cold junction is 30° and the hot junction is 40° by 10. The factors for P_2 used in this manner are given in Table I, column four.

Reduction to Fahrenheit Scale.—All the temperatures referred to above are given in degrees Centigrade. The conversion to the Fahrenheit scale occasions no difficulty. When the couple is originally calibrated with the cold junction at the temperature of melting ice, 32° Fahr., and used with the cold junction at the temperature, t_o ° Fahr., the actual temperature, t , of the hot junction is obtained by adding to the observed temperature, t_o , the value of the product $(t_o - 32)K$, instead of $t_o K$, where K is the correction factor at the particular observed hot-junction temperature. If the couple is calibrated with the cold-junction temperature of t_o ° Fahr. and used with the cold junction at a temperature of t_o ° Fahr., the correction term to be added to the observed temperature, t_o , has the same form as before,

$$p = (t_o - t_o)K'$$

where K' is the factor computed from the mean slope between t_o and t_o , and the slope at the observed temperature. The values of the ordinates for the curves in Figs. 2, 3, and 4 should be increased by the factor 9/5, while the abscissae are directly converted to the Fahrenheit scale. As an example, in

TABLE II.—TEMPERATURE FAHRENHEIT VS. COLD JUNCTION FACTOR
 P_2 (Pt, 90 Pt—10 Rh)

Observed Temperature °F	$(\frac{de}{dt})_2 \div (\frac{de}{dt})_1$	$(\frac{de}{dt})_2 \div (\frac{de}{dt})_1 = K$	$(\frac{de}{dt})_2 \div (\frac{de}{dt})_1 = K'$
800	0.59	0.61	0.63
1000	.56	.58	.60
1200	.54	.56	.58
1400	.53	.54	.56
1600	.51	.52	.54
1800	.49	.51	.53
2000	.48	.49	.51
2200	.46	.47	.49
2400	.44	.45	.48
2600	.43	.44	.46

the case of Fig. 4, the ordinates will lie between the 0 and 18, the abscissae between 32 and 212° Fahr., and the observed temperatures of the hot junction become 752, 1472, 2012, and 2552° Fahr. In Table II are given the correction factors for P_2 (Pt, 90 Pt—10 Rh) for a series of observed temperatures ° Fahr. These follow directly from Table I. The second column shows the factors computed on the basis of the old slope correction, Form II. The third column contains the factors computed from the preferred slope correction, Form III. In

TABLE III.—CORRECTION TO THERMOCOUPLE SCALE

Thermocouple Scale °C	Correction to Add Degrees Centigrade	Thermocouple Scale °F	Correction to Add Degrees Fahrenheit
300-1200	0	600-2200	0
1300	+2	2400	+5
1400	+6	2600	+14
1500	+14	2800	+35

the fourth column are given the correction factors when the couple is calibrated with a cold-junction temperature of 86° Fahr. and used with the cold junction not far from 104° Fahr.

Conversion of the Thermoelectric Scale to the Standard Temperature Scale.—When the ordinary parabolic equation $e = a + bt + ct^2$ is applied to a (Pt, 90 Pt—10 Rh) or (Pt, 90 Pt—10 Ir) thermocouple standardized at three temperatures, the melts of zinc, antimony, and copper, the curve

extrapolated to 1500° C. differs somewhat from the temperature shown by the gas thermometer. This is treated in full in "The Measurement of High Temperatures," pp. 112-116 *loc cit*, but may be briefly summarized in the following table, which holds approximately for the usual type of platinum, platinum-iridium, and platinum-rhodium couples.

The above discussion of couples P_3 , P_4 and W ; is on the basis of the thermoelectric scale. Hence temperatures higher than 1200° C. or 2200° Fahr. should be further corrected in accordance with Table III. The order of applying the cold-junction correction and the above correction is of little importance since both are small. Couples calibrated by the Bureau of Standards are corrected as shown in Table III so that the temperatures indicated by them accord with the established gas thermometer scale.

Devices for the Elimination of the Cold-Junction Correction.—Various methods have been proposed for the elimination of the cold-junction corrections. If a coil of copper wire were shunted across the terminals of the thermocouple in the head of the instrument, an increase in the temperature of the cold junction, ordinarily producing a decrease in the observed emf, would increase the resistance of the shunt and thus tend to compensate for the thermoelectric effect. However, for exact compensation at all temperature ranges, a coil of wire would be required of which the resistance is a function not only of its own temperature, but also of the hot-junction temperature. This, of course, is impossible to realize in practice, but the method might be applied as a partial correction.

Bristol¹ base metal couples are provided with extension wires which permit the location of the cold junction in a constant temperature room some distance from the furnace. Bristol has also devised an automatic compensator consisting of a bulb of mercury into which a loop of platinum wire dips. This is inserted in the circuit near the head of the couple so that the variations in the cold-junction temperature cause the mercury to expand or contract and short-circuit different lengths of the wire loop, thus altering the resistance with a result somewhat similar to that obtained by the use of a shunt coil.

Hartmann and Braun have the cold-junction head jacketed in order that a stream of water may be used to keep the head at an uniform temperature.

The Crompton Company use a multiple scale on their galvanometers so constructed that the pointer reads directly the temperature of the furnace, each scale corresponding to some definite value of the cold-junction temperature.

Several instrument makers have tried interposing a supplementary thermocouple circuit of inexpensive material, the hot junction being located at the cold end of the main thermocouple and the cold junction at the galvanometer, some distance away.

Possibly the most simple compensating device is that employed by Siemens & Halske, and one or two other firms. Here the galvanometer indicator has a large zero adjustment so constructed that the pointer may be set, on open circuit, at the temperature corresponding to that of the cold junction, the graduations having been previously empirically determined for the given thermocouple.

All of these methods possess certain advantages, but many of them are open to the objection that they are cumbersome and frequently liable to introduce new errors. Attention might well be given to the more careful construction of the indicating instruments. Many galvanometers have a temperature coefficient so large that small variations in the room temperature affect the readings to a marked degree. This may give rise to errors as large as those commonly found for the cold-junction temperature.

It is, moreover, needless to mention that the ordinary indicator should not be used in close proximity to other magnetic instruments or to iron unless originally calibrated in that position. A simple trial will readily convince one that this precaution is necessary.

¹The following and other methods of cold-junction compensation are discussed in "The Measurement of High Temperatures," Burgess and Le Châtelier, p. 156, 3rd Ed., 1912, Wiley, N. Y.

When due care is exercised to locate the galvanometer in such a position that its magnetic field is not disturbed by outside influences and where its temperature is reasonably constant, to occasionally recheck the thermocouple calibration, and to properly correct for the cold-junction temperatures, the

simple thermocouple without any compensating devices whatever will prove very satisfactory. Acknowledgment is due Mr. I. N. Kellberg for computing and drawing the curves shown and to Dr. G. K. Burgess for a number of suggestions.

Bureau of Standards, Washington, D. C.

Heat Transmission and Entrainment in Vacuum Evaporators

An Account of an Elaborate Series of Tests, by Prof. E. W. Kerr, in which the Effect of Various Elements on Heat Transmission and Entrainment was Analyzed.

A very elaborate series of tests of a vacuum evaporator with especial regard to heat transmission and, secondly, to entrainment has been carried out under the direction of Professor E. W. Kerr at the Agricultural Experiment Station of the

Louisiana State University and A. & M. College, at Baton Rouge, La. Professor Kerr was assisted in this experiment by Mr. A. J. Isaacks. Assistance was also given in the experiments by Messrs. H. A. Nadler, E. M. Copp, J. F. Gunther, A. Guell

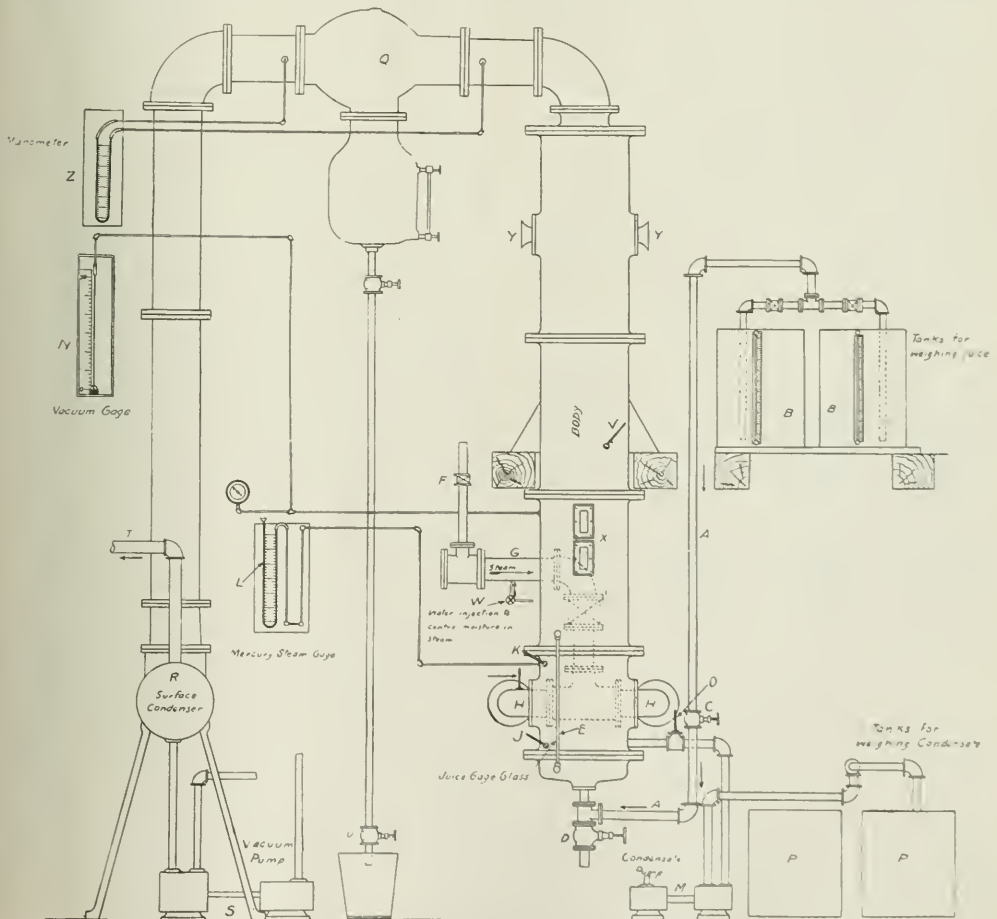


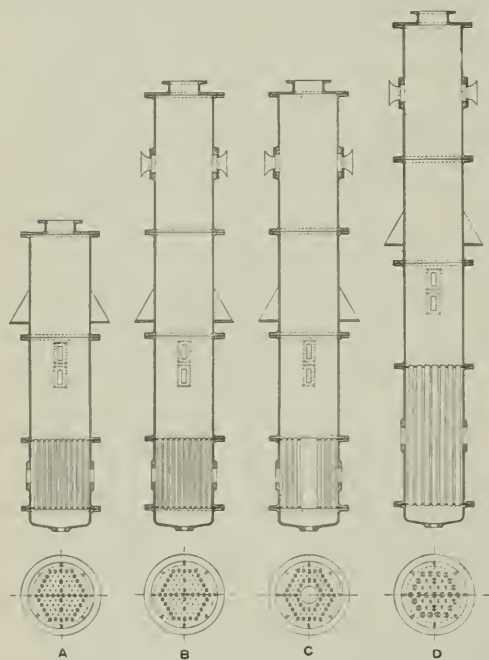
FIG. 1. GENERAL ARRANGEMENT OF EVAPORATOR TEST.

A—Entrance of Juice. B—Tanks for Weighing Juice. C—Valve for Regulating Juice Supply. D—Valve for Emptying Evaporator. E—Gauge Glass for Observing Level of Juice. The Steam Enters Through Valve F, Expands Into Pipe G and Divides and Enters Through Pipes H Into the Calandria. I J K—Thermometers. L—Manometer. N—Vacuum Gauge. M—Wet Vacuum Pump for Removing Steam Condensed in Calandria. P—Barrels on Platform Scales for Measuring Condensate. Q—Catchall or Separator for Catching Any Juice Carried Over by Entrainment. R—Surface Condenser. S—Wet Vacuum Pump. T—Connection to Dry Vacuum Pump (Not Shown). X and Y—Side Glasses for Observing Inside of Evaporator.

and W. P. Denson. The complete account of the experiments is given in Louisiana Bulletin, No. 138, of the Agricultural Experiment Station, Baton Rouge, La. In the following article we give a summary of the experiments and results.

Since a triple or double effect is more complicated and more difficult of control in operation than a single effect it was decided to use a single effect evaporator for the experiments, but to have it arranged so that both the pressure of the steam supply and the vacuum in the vapor space could be thoroughly controlled and made any desired and predetermined amount. In this way, the conditions as regards steam pressure, vacuum, density of juice, etc., of either body of a multiple effect could be reproduced.

Fig. 1 shows the arrangement of the test which, with the description in the caption, will be easily understood. Originally it was the intention to supply a pump with which to remove the syrup from the evaporator in the manner usually employed in sugar factories. It was finally decided, however, not to remove the syrup during operation, but to supply water instead of juice to the evaporator at a rate equal to that at which water is removed by evaporation. In this manner the density of the juice in the charge before the operation started



FIGS. 2 TO 5.—THE FOUR EVAPORATOR DESIGNS EMPLOYED IN THE TESTS

could be kept constant without the use of a syrup pump.

The condensing of the vapors was effected by means of the surface condenser R containing a total of 150 square feet of cooling surface. Cooling water was supplied to the condenser from the mains of the Baton Rouge Water Works. The condensed vapors were removed from the condenser by means of the wet vacuum pump S. In most of the tests this pump was sufficient for obtaining the vacuum necessary though where more than about 23 in. was required it was necessary to use a rotative drive vacuum pump, not shown in the figure, in addition to the wet pump. With both pumps it was possible to carry a vacuum of 27 in. The suction pump of the dry vacuum was connected to the condenser as shown at T. The $\frac{1}{2}$ -in. pipe attached to the bottom of the catchall Q was used for intercepted juice and condensation. By properly manipulating the

valves U this could be done while the evaporator was operating and under vacuum. The temperature in the vapor space of the evaporator was observed by means of the thermometer V inserted in a deep thermometer well. All other details will be understood from the descriptions in the caption below Fig. 1.

Four different arrangements of the evaporator itself were used in the tests as shown in Figs. 2 to 5. Fig. 2 shows the arrangement A of the evaporator body as first used containing 80 tubes, $1\frac{3}{8}$ in. in diameter and 24 in. long. The tubes were of copper and 18 gage B. W. G. The inside diameter of the body was 20 in. The height of the vapor space, upper tube plate to top, was 6 ft.

After the first experiments had been made it was decided that the height of the vapor space was insufficient and the design was changed to B as shown in Fig. 3, in which the height of the vapor space is 10 ft. Bell eye-glasses were added. The height of the vapor space is the same in the designs Fig. 4 and 5 as in Fig. 3. The arrangement C of Fig. 4 shows an evaporator body C with "down take." There are 66 copper tubes, $1\frac{3}{8}$ in. in diameter, 24 in. long, 18 B. W. G. The down take of 6 in. in diameter is made of copper 13 B. W. G. copper tube plates $\frac{1}{4}$ in. thick, height of vapor space 10 ft.

The arrangement D shown in Fig. 5 shows an evaporator body without down take. There are 39 copper plates, 2 in. x 48 in., 18 B. W. G. Copper tube plates $\frac{1}{4}$ in. thick, height of vapor space 10 ft.

The methods employed in making the experiments are described in the bulletin and extensive tables are given comprising the total results of all the tests.

In this article the "coefficient of heat transmission" is defined as the "B.t.u. transmitted per square foot of heating surface per hour for each degree difference of temperature of the two sides of the heating surface." This temperature difference is called the "temperature fall."

Effect of Varying the Head of Juice on Evaporation

Operating an evaporator with a high head of juice is detrimental to heat transmission, and therefore to capacity, because it reduces the temperature fall. This is due to the fact that the average temperature of the boiling juice is increased because of the increased pressure due to the weight of the liquid.

The theoretical average boiling pressures and temperatures as affected by hydrostatic head may be calculated, provided the head of the juice H and the density of the juice are known. Let us assume H to be 48 in. and the density of the juice to be 0.957, then a particle of water at the surface will be converted into steam under a pressure of the vapor space above the liquid, while a particle at the lower end of the tubes will be formed into vapor under the same pressure plus that due to the weight of the column of liquor above it. The average pressure of boiling will, therefore, be that in the vapor space plus the pressure due to half the column of liquor—that is 24 in. The weight of a cubic inch of liquor with density 0.957 is 0.035 lbs. and the pressure per sq. in. due to 24 in. head is $0.035 \times 24 = 0.83$ lb. per sq. in., or $0.83 \div 0.49 = 1.69$ in. mercury. Hence the average vapor pressure of the juice will be 1.69 in. of mercury higher than the pressure in the vapor space above the liquid, and the effective boiling temperature will be correspondingly increased, and consequently the effective temperature difference between the outside and inside in the evaporator will be reduced. If a calculation of this kind is carried out for multiple-effect evaporators it is found that this loss is greatest in the last effect where the pressure is lower than in the preceding vessels.

To determine experimentally the effect of the static pressure due to the weight of juice in reducing the effective temperature difference which causes evaporation, three series of tests were made. The results of these tests are shown in Fig. 6. Curve A refers to tests made with the evaporator of Fig. 2, the steam pressure being 3.75 in. mercury, the vacuum in the vapor space 3 in. Curve B refers to tests made with the evaporator of Fig. 5, the steam pressure being atmospheric and the vacuum in the vapor space being 8 in.

Curve C refers to tests made with the same evaporator (Fig. 5), but with a vacuum of 21 in. in the same calandria and a vacuum of 25 in. in the vapor space.

All three of these series were performed with water instead of juice for the reason that it would have been practically impossible to keep the density of the juice constant, with varying depths, and this would have introduced a second variable which would have made it impossible to isolate the effect of "head."

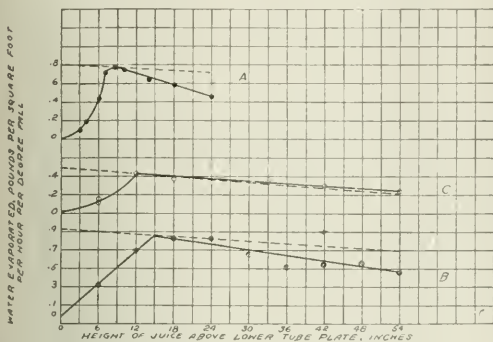


FIG. 6.—CURVES SHOWING THE INFLUENCE OF "HEIGHT OF BOILING" UPON HEAT TRANSMISSION. WATER BOILED IN EACH SERIES.

The broken straight lines in each of the three curves, Fig. 6, show the theoretical variation of heat transmission with change of head, calculated as outlined above, this line being drawn through the highest point of the experimental curve in each case. The ordinates in these curves represent pounds of water per square foot of heating surface per hour, per degree fall.

In series A (see curve A) it will be noted that the maximum capacity was attained with about 77 in. head—that is, with the surface of the liquid in the neighborhood of one-third the length of the tubes above the lower tube plate; in fact, the head for maximum capacity was better defined in this series than in the other two, for the reason that a greater number of tests were made with heads near that for maximum capacity. However, it is clear that the highest point in curves B and C are both with heads between 12 in. and 18 in. and thus showing that the maximum capacity was obtained with heads one-quarter to one-third the height of the tubes.

The curves show a very rapid decrease in capacity as the head is reduced below that giving the maximum capacity. This is due to the fact that with the low heads portions of the heating surface at the tops of the tubes are not effective, being uncovered by liquid. The unsubmerged portion of the heating surface will transmit only a small amount of heat from the steam on the one side to vapor on the other side, which will superheat the latter.

It will be noted that all three curves are straight lines for heads greater than that for maximum capacity, and are thus similar to the theoretical in this respect. The fact that curves A and B are lower than the theoretical indicates the actual loss due to high boiling to be greater than the theoretical. In A the maximum coefficient of heat transmission was 976 with a head of 7 in., whereas the minimum was 490 with a head of 24 in.—that is, a loss of 49.8 per cent. by increasing the head from 7 in. to 24 in. The actual coefficient was also 27.1 per cent. less than the theoretical at 24 in. head. Similarly, the lowest coefficient—that is, with 54 in. head—in B was 44.7 per cent. below the maximum and 31.8 per cent. below the theoretical lowest.

In C the minimum, at 54 in. head, was 42 per cent. below the maximum. The C series differed from the other two, however, in that the theoretical curve almost coincides with the actual. The reason for this is not apparent. In this series the average evaporation coefficient was little more than half that in the

other two series and perhaps this may have had something to do with it. The heads as recorded were measured in the juice gage glass. The level of the juice, as viewed through the sight glasses, generally appeared higher than shown in the gage glass, the difference depending on the rate of evaporation. It may be possible that the actual head was greater than that recorded with the higher rates of evaporation, which would account for the difference between the theoretical and actual curves in A and B; also for the coincidence of the two curves in C, where the rates of evaporation were relatively small.

Effect of Temperature Levels Upon Heat Transmission

In order to get experimental data showing the variation in heat transmission at different temperature levels, three series of tests were made with the temperature in calandria varying between limits that would include the temperatures usually existing in the different calandrias of multiple evaporators. These three series are designated A, B, and C, respectively.

The A series was made with the calandria having no down take and with tubes $1\frac{1}{2}$ in. x 24 in. (Fig. 3); the B series with the down take calandria having tubes $1\frac{1}{2}$ in. x 24 in. (Fig. 4); and the C series with the 4-ft. calandria (Fig. 5). The average temperature fall was practically the same in each series, the principal variable being temperature in calandria, or what is the same thing, pressure in calandria. All other conditions were kept as nearly constant during a series as was possible.

The effect of temperature level upon heat transmission is shown in Fig. 7. It will be noted that the direction and inclination of the three curves are quite similar, thus checking well. Assume that the temperature in the calandrias of the last bodies of doubles, triples, and quadruples are as marked 2, 3, and 4 respectively at the top of the figure—that is, 190 deg., 180 deg., and 175 deg.—also that 224 deg. is the temperature in the first body with either type.

From the three curves we find that the average coefficient of heat transmission in the second body of a double effect would be about 65 per cent. of that in the first body. In the same manner, the coefficient of heat transmission in the last body of a triple would be about 60 per cent. and in the last body of a quadruple about 58 per cent. of that in the first body.

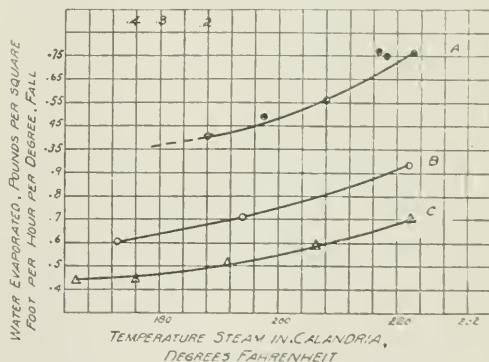


FIG. 7.—CURVES SHOWING THE INFLUENCE OF "TEMPERATURE LEVEL" UPON HEAT TRANSMISSION. WATER BOILED IN EACH SERIES.

Four factors—viz., density of juice, cleanness of heating surface, air in steam, and density of steam or vapor—are possible causes of low coefficient of heat transmission in the latter bodies of a vacuum evaporator, but two of these factors were eliminated in these tests—viz., density of juice and cleanness of heating surface. Water was used during all the tests so that the density (also the cleanness of heating surface) remained unchanged throughout the series.

Thus the variation in the coefficient must be due to the variation in the quantity of air in the calandria, or to the variation

in the density of the steam or to both of them combined. It will be noted that the height in the three curves varies considerably. This is probably due to the fact that different types of calandrias were used in the three series.

Effect of Air in the Heating Steam Upon Heat Transmission

As explained in the preceding chapter, decrease of heat transmission in the latter bodies of multiple evaporators may be due to low steam density, or the presence of air in the steam or to both combined.

In an attempt to get further data upon this question and to determine, if possible, the proportion of responsibility of each of these two factors, a series of tests was made to determine the effect of air in the steam supplied to the calandria upon the coefficient of heat transmission.

In all of these tests the vacuum in the calandria was constant at about 12.5 in. and the vacuum in the vapor space about 22 in.; in fact, all conditions were kept constant, as nearly as possible, except the quantity of air with the steam in the calandria. To a casual observer it would have appeared that all conditions were constant.

The amount of air in the calandria was varied mainly in two

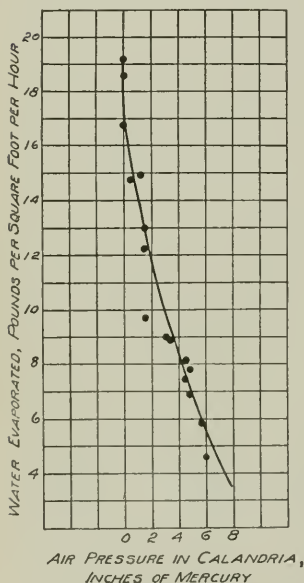


FIG. 8.—INFLUENCE OF AIR IN HEATING STEAM UPON EVAPORATING CAPACITY. WATER BOILED IN ALL TESTS

the presence of air had something to do with the reduced temperature in the calandria. It was also found that the temperature at the bottom of the calandria was often lower than at the top. The density of air is greater than that of steam, and the air naturally settles to the bottom, from whence it is removed by the condensate pump.

The method by which the quantity of air was measured was based upon Dalton's law, which states that "the pressure of a mixture of air or other gas and a vapor is the sum of the pressures each would have if it occupied the same space alone." In other words, the pressure or vacuum gage on the calandria shows the sum of the pressures due to steam and air. The temperature of the steam is that corresponding to its pressure, hence if the temperature of the mixture is known it is easy to find the partial pressure of the steam by referring to a steam table. By determining the pressure due to the

ways—viz., by varying the speed of the condensate pump M (Fig. 1), and, where it was desired to admit large quantities of air, by opening very slightly an air cock connected with the manometer L.

The amount of air in the steam was measured as follows: The temperature in the calandria was obtained by means of the two thermometers (J and K), one near the bottom and the other near the top. In preliminary experiments it was found that the average of the temperatures recorded by these thermometers was often lower than that of saturated steam corresponding to the pressure, as found in steam tables. It was also noted that this difference was greater when the condensate pump ran slow than when it ran at high speed, indicating that

steam in this manner and subtracting it from the sum of pressure as shown by the gage, the pressure due to the air is obtained. This method was used in the experiments, the average of the temperatures at the bottom and at the top of the calandria being taken as the temperature of the steam.

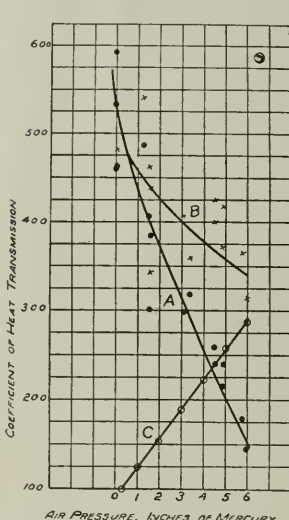


FIG. 9.—CURVE A, COMBINED EFFECT OF "AIR FILM" AND DECREASED TEMPERATURE FALL. CURVE B, EFFECT OF "AIR FILM," CURVE C, LOSS BY REDUCED TEMPERATURE FALL.

varying the quantity of air in the calandria the injection of water into the suction pipe of the condensate pump was also employed.

In order to get the most effective removal of air it was necessary to use injection in this manner.

Referring to the curve (Fig. 8) it is seen that the plotted points are very near the smooth curve, also that the decrease in capacity is very rapid with small additions of air, the evaporation varying from 4 to 19 pounds of water per square foot per hour, due to changing the air pressure from 8 to 0 in. of mercury. An air pressure equal to 1 in. of mercury caused the evaporation to be reduced from 19.2 to 14 pounds—that is, a reduction of about 27 per cent.

The effect of air in reducing the coefficient of heat transmission is shown graphically in the curves of Fig. 9.

Curve A is based upon the coefficient of heat transmission calculated by dividing the B.t.u. transmitted per sq. ft. per hour by the temperature fall. It is the curve of "actual coefficient of heat transmission."

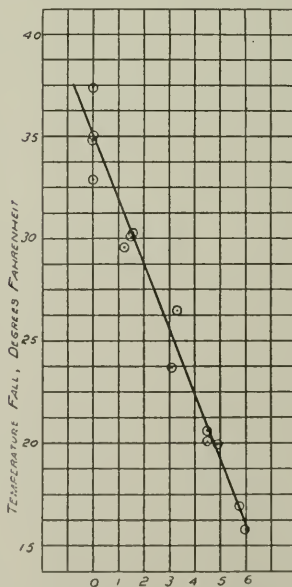


FIG. 10.—REDUCTION OF TEMPERATURE FALL AS AFFECTED BY AIR IN THE HEATING STEAM

Curve B is based upon a slightly different calculation in that the B.t.u. transmitted per sq. ft. per hour is divided in each case by the temperature fall obtained by subtracting the temperature in the vapor space from the temperature corresponding to the pressure in the calandria instead of from the actual observed temperature, and will be designated as the "apparent coefficient of heat transmission" curve.

As will be seen, the reduction in capacity due to the presence of air is composed of two factors; viz., (1) a reduction in the temperature fall, as shown by Fig. 10, and (2) a reduction in the heat transmission per degree fall—that is, a reduction of the coefficient of heat transmission due to the nonconducting film of air next to the tubes.

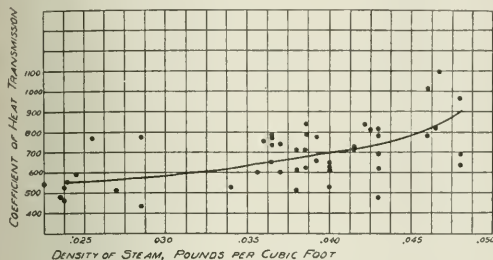


FIG. 11.—INFLUENCE OF STEAM DENSITY UPON THE COEFFICIENT OF HEAT TRANSMISSION

Curve B shows the latter quantity—that is, coefficient of heat transmission as affected by air pressure. Since curve A shows the total loss as affected by air pressure, it is evident that the loss due to reduced temperature fall is represented by the length of the ordinates between curves A and B. These values have been replotted and curve C drawn. This curve, therefore, shows the reduction in the number of B.t.u. transmitted per sq. ft. of heating surface per hour per degree difference due to reduced temperature, with varying air pressures.

Effect of Density of Steam

In order to analyze the effect of the variation of density of steam on evaporation some fifty-eight tests with heat transmission coefficients varying from 400 to 1100 and in which there was little or no air in the calandria were selected and coefficients of heat transmission plotted against density of steam. The curve in Fig. 11 is the result.

All of the tests selected were made with water instead of juice. In other respects, however, there was some variation of conditions, especially as regards head and type of calandria. For these reasons the points from which the curve was drawn are pretty well scattered though the general direction of the curve is rather definitely indicated.

The density of the steam is usually, say 0.024 in the calandria of the last body of a triple effect, and, say 0.045 in the first body. From Fig. 11 we find that the coefficients for these two densities are about 560 and 800 respectively. This indicates a great effect attributable to density of steam. The general correctness of the curve shown in Fig. 11 was checked in another manner given in the bulletin.

Effect of Density of Boiling Liquid

It is generally understood that the coefficient of heat transmission is greater with thin liquids and water than with denser liquids. This is, in all probability, due in part, at least, to the fact that with thicker liquids the circulation is not so good. Then, too, the resistance to the transfer of heat from metal wall to liquid is possibly increased by the greater amount of solid matter in the thicker liquids.

In order to isolate the effect of juice density upon heat transmission, two series of tests were made in which an attempt was made to keep all other conditions constant. The

density of the juice was varied within limits that would include densities likely to be used in evaporators, viz., 8.8 Brix to 67.3 Brix. Series A was made with evaporator B, Fig. 3, that is, with the calandria having no downtake and with tubes $1\frac{3}{8}$ in. x 24 in. Series B was made with evaporator C, Fig. 4, that is, with a calandria the same as the above except that it had a downtake in it. The results of these two series of tests are shown graphically in the two curves of Fig. 12.

It will be noticed that the two curves are parallel, thus checking well. That curve B is above curve A is doubtless due to the difference in the form of the calandrias used, the former having a downtake and the latter no downtake. With the downtake calandria—curve B, the coefficient of heat transmission for densities of 10 and 50 are 750 and 625 respectively, that is 1:0.83.

Since the lower curve is parallel to the upper one, the ratio is practically the same for it. It should be mentioned that the reduction in heat transmission shown in this series is due solely to the increased density of the liquid and not to any fouling of the heating surface. The heating surface was clean in all of the tests.

Effect of Superheat in Steam on Heat Transmission

It is held by some that with superheated steam in the calandria, the coefficient of heat transmission is reduced. The superheat usually results when live steam is used, due to the reduction of pressure by throttling.

In order to obtain data regarding the effect of superheat, two series of tests were made, but the results of both tests seemed to prove conclusively that the coefficient of heat transmission is not affected by varying the superheat in the steam. According to this, live steam should give as good results as exhaust steam, as far as capacity is concerned.

Value of the Downtake

"Most evaporators of the standard type are now built with downtakes—that is, with a large central tube, the diameter of which depends on the diameter of the body itself. Some few builders, however, claim that there is little merit in downtakes and do not use them. The advantage claimed for the downtake is that it adds in juice circulation—that is, it defines the path of circulation—and in this way increases the velocity, the movement being always downward through the downtake and upward through the tubes. Some few have claimed that the transverse area of the downtake should be equal to the combined transverse area of all the tubes. This, however, would result in a very large diameter of body and high first cost. In average practice the

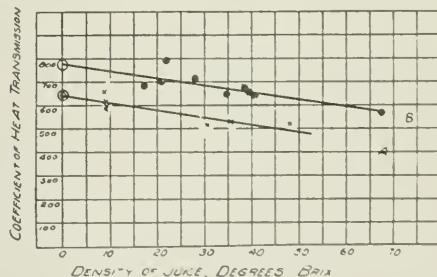


FIG. 12.—INFLUENCE OF DENSITY OF JUICE. CURVE A—EVAPORATOR WITHOUT DOWNTAKE. CURVE B—EVAPORATOR WITH DOWNTAKE

diameter of downtake for, say, an 11 ft. body would be 34 in. to 38 in. in diameter and this would give the downtake a transverse area of about 18 per cent of the total transverse area of the tubes."

In order to get data on this subject two calandrias (Figs. 3 and 4) were constructed, one with a downtake 6 in. in diameter and the other without a downtake. They were alike in every other respect. The calandria with the downtake had 60 tubes

of $1\frac{3}{8}$ in., the total transverse area of which was, therefore, 87.6 sq. in. The area of the downtake being 28.3 sq. in., was 32 per cent of the area of all the tubes.

Several series of tests were made with these two calandrias under conditions such as will make comparisons of the heat transmission in them possible. The different tests given in the bulletin show clearly the value of the downtake in increasing the heat transmission coefficient. Fig. 12 given before shows this in the two curves *B* and *A*, in which *A* refers to the evaporator without downtake and curve *B* refers to evaporator with downtake. The coefficients of heat transmission for the calandrias with and without downtakes are, therefore, in the ratio 100:83. Without going into any further details on this subject the final conclusions of the authors may be quoted.

"Tests on two calandrias, alike except that one had and the other had not a downtake, showed that the coefficient of heat transmission was 20 per cent greater in the downtake calandria.

"The net increase in evaporative capacity of the downtake calandria over that without a downtake, after accounting for the reduced heating surface in the downtake calandria, was 19 per cent.

"It is likely that this increase of the transmission coefficient is due to the improved circulation due to the downtake."

Dimensions of Tubes

Tests upon two calandrias, one with tubes $1\frac{3}{8} \times 24$ in. and the other with tubes 2×48 in., seem to show that the coefficient of heat transmission was about the same in both.

The 48-in. tube probably has the advantage of more rapid circulation than the 24-in. tube, while, on the other hand, it has the disadvantage of having to be operated with greater juice head; perhaps also with greater water flooding of the heating surface.

Entrainment

As already explained, any juice carried over by entrainment was caught in the separator or catchall *Q* (see Fig. 1). This separator used in the tests was the Swartwout separator of the Ohio Blower Company. It is shown in section in Fig. 13.

This separator is designed for use as an oil separator between steam engines and surface condensers. Thus it will be seen that the conditions for which it was designed are similar to those in vacuum sugar apparatus.

The separation of liquid from steam is effected by centrifugal force. The mixture of steam and liquid upon entering the separator is given a whirling motion by means of the spirals shown in the figure. This whirling motion causes the heavier liquid particles to be thrown to the circumference, from whence it drains down to the receptacle at the bottom. The liquid caught in the separator was removed from the separator during the operation of the evaporator, if necessary, by means of the two valves *U* and the long pipe.

The vapors passing on through the separator were condensed in a surface condenser, hence the condensed vapors were not diluted by direct contact with the cooling waters. This made possible more accurate determinations of the sugar lost in the same.

The object of the entrainment tests was to determine the effect upon entrainment due to varying the rates of boiling, the height of boiling, the length and diameter of tubes, the vacuum and velocity of vapors in the vapor space, and the density of the juice.

The results of these entrainment tests are summed up as follows:

Entrainment was found to vary mainly with the weight of water evaporated per square foot of heating surface per hour

(rate of evaporation) with the height of boiling and with the dimensions of the heating tubes.

With the downtake calandria the maximum entrainment was at the rate of 27.5 pounds of juice per hour, with an evaporation rate of 17.6 and a juice head of 26 in.

With low heads (12 in.) evaporation rates of 23 were maintained with practically no entrainment.

With the 24-in. calandria without a downtake the sugar caught in the separator varied from 0 to 44.7 lb. per hour. In one series, where the average rate of evaporation was 18, the average weight of sugar entrained was 0.46 lb. per hour, corresponding to 27.6 lb. per hour in one body of a 10-ft. standard, the head being 16 to 24 in.

With the same calandria another series of tests, with an average evaporation of 10 lb. and with heads of 16 to 24 in., gave no entrainment.

The maximum entrainment (average of four tests) with this calandria was 30 lb. per hour, with an average rate of evaporation of 25 and an average juice head of 19 in. In one test an evaporation rate of 27 was maintained without entrainment. This was due to the low head of 12 in.

The maximum evaporation rate possible with this calandria without entrainment is probably about 22, this being the average of five tests, in which there was no entrainment.

The maximum entrainment with the calandria having tubes 2×48 in. was 24.3 lb. of sugar per hour, with an average evaporation rate of 12.4 and an average head of 26 in., this being the average of three tests.

The average evaporation rate of a number of tests on this evaporator, all of which were near the entraining point, but in which there was actually no entrainment, was 7.8.

Thus it seems that the calandria with tubes 2×48 in. cannot be forced to more than half the rate that is possible with $1\frac{3}{8} \times 24$ in. tubes without entrainment.

By keeping the juice level low relatively large capacities are possible in either type of calandria without entrainment.

Neither the vacuum in the vapor space nor the velocity of vapors in the vapor space seem to affect entrainment to any measurable extent.

The tests do not furnish data regarding the effect of juice density upon entrainment. If entrainment is influenced by density at all it is probably in the fact that the greater the density of the boiling juice the greater the density of the juice caught in the separator and, therefore, the greater the amount of sugar.

The separator used was very efficient, a slight trace of sugar being shown in the condensed vapors in only one test.

In all other tests, even where large quantities of juice were caught in the separator, no trace of sugar was shown by the alpha-naphthal test.

The loss of pressure due to the friction of the vapors in passing through the separator was 0.5 in. water, with a velocity of 7940 F.P.M., an amount practically negligible.

While the above gives a concise summary of the contents of the bulletin anyone who is especially interested in problems of vacuum evaporation will find the whole bulletin interesting and profitable reading.

Professor Kerr and Mr. Isacks are to be heartily congratulated on the elaborateness of the tests and the success obtained and their future investigations will be awaited with interest.

A New Dry Mixer.—The thorough mixing of such materials as fertilizers, fuel ores, the ingredients for glass, dry plaster materials, etc., in fact, every material which is dry and dusty, is now being accomplished by many manufacturers by the use of specially designed dry mixers. For this purpose the Ransome Concrete Machinery Company, Dunellen, N. J., are building a dry mixer which is of the same principle as that of their concrete mixer, which has been so widely used all over the world, with the exception that the drum is sealed, thus preventing the escape of dust during the mixing process.

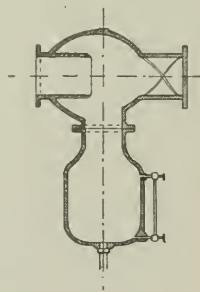


FIG. 13.—CATCHALL

Methods for Investigating and Recording Atmospheric Impurities, Including the Soot and Dust Suspended in the Atmosphere

By John B. C. Kershaw

One of the most important *Resolutions* passed at the International Smoke Abatement Exhibition and Conference, held in London in March, 1912, was that relating to the framing of a standard method of catching and measuring the more commonly occurring impurities of the atmosphere. This resolution was worded as follows:

Atmospheric Pollution.

"That in the view of this conference, it is desirable that immediate steps should be taken, to decide upon and secure the general adoption of a standard method for the measurement of atmospheric pollution by smoke, and the other products of combustion and dust, in order that the data now being collected may possess a comparative value, and that a committee be appointed to draw up details of a standard soot and dust measuring apparatus and methods of its use.

"That the committee do consist of Dr. H. A. Des Voeux, M. D. (Hon. Treasurer, Coal Smoke Abatement Society); Prof. J. B. Cohen, B.Sc., Ph.D., F.R.S. (Leeds University); Bailie W. Smith (Covener of the Air Purification Sub-Committee of the Glasgow Corporation); Dr. Hawksley (Assistant Medical Officer of Health, Liverpool); Mr. J. B. C. Kershaw, F. I. C. (Hamburg Smoke Abatement Society); Dr. John S. Owens, M. D., A. M. I. C. E. (Coal Smoke Abatement Society); Dr. W. N. Shaw, M. A., D. Sc., LL.D., F. R. S. (Director, Meteorological Office), and Dr. S. A. Vasey (*The Lancet*), with power to add to its number. Subsequently added: Dr. R. Lessing, Ph.D., F. C. S.; Mr. E. D. Simon (Smoke Abatement League of Great Britain), and Mr. J. G. Clark.

"That the conclusions and recommendations of such Committee be reported to the Delegates forming the Conference and to all Municipal Authorities in the United Kingdom."

The Committee, since its appointment, has held three meetings in London, and has given careful consideration to the various methods that have been suggested or tried for the collection and estimation of atmospheric impurity. The following is a list of the various methods which received consideration, with some notes upon the places where they have been tried and upon their relative advantages and disadvantages.

It was decided that the method selected as best must fulfil the following conditions:

(1) It should measure the actual amount of suspended matter in the air, not the amount relative to other places, i. e., it should give the pollution in, say, grams per cubic metre.

(2) The foreign matter or impurity should be collected in such a way as to permit of analysis.

(3) It should be as simple as possible, provided conditions (1) and (2) are fulfilled.

Methods that Have Been Suggested or Tried

I. Filtration of a measured volume of air through a cotton-wool or collodion-wool filter and examination of the deposit.

This method has been used by Russell in London and by Hahn in Germany and can be made to give good results. With care condition (1) can be fulfilled and when collodion-wool is used it can be made to satisfy condition (2). It requires, however, costly apparatus and considerable technical skill, and it possesses inherent defects if a color scale be employed as measure or gauge of the amount of deposit.

II. Collection of rainfall; evaporation and weighing of residue.

This method was used in the investigations started by Dr. Des Voeux and *The Lancet* in 1910 in order to obtain the soot-fall of London. It was also employed by Professor Cohen in Leeds about the same date. The method does not, strictly speaking, give the amount of air pollution, but the amount which falls on a given area (a) in rain, snow, etc.; (b) during

dry weather. (a) and (b) are not separated. The apparatus used for collection of the impurities may be very simple, and the technical work can be done in a central laboratory.

III. Aitken's dust counter.

This was devised by John Aitken for counting the number of dust particles in the air. The method and apparatus satisfy condition (1), but not (2).

IV. Exposure of glass plates to the air for a specified time followed by washing in water, and measurement of the adherent deposit that remains, by comparison with standard colors.

This method has been used by Professor Cohen in Leeds for catching the matter which will adhere to a glass plate, e. g., tarry soot. It would seem that the thickness and color of the deposit must be affected by the amount of tar present in the smoke of the city. The method aims only at comparative results, and does not fulfil either condition (1) or (2).

V. Exposure of glass plates coated with a drying oil, to a stream of air: or to the atmosphere, either (a) for a fixed time, followed by a measurement of the opacity by comparison with a calibrated scale, or (b) for such time as will produce a fixed opacity, and comparison of the time required with a calibrated time scale.

This method has been used by Cohen in Leeds and in Hamburg by Liefmann, and it might possibly be made to fulfil conditions (1) and (2). It requires, however, elaborate apparatus, and some technical skill in its use, and it is not suitable for application on a large scale and under the conditions which will obtain for the Committee's observations.

VI. The air is drawn at a measured rate through a filter-paper, and the degree of blackness produced in a given time is then measured.

This method has been used by Rübner in Germany and by Bailie Smith in Glasgow. It gives results which depend upon the color of the matter caught, as well as upon its amount. Condition (2), however, would be fulfilled.

VII. Measurement of the transparency of a column of air of given length, when illuminated with a standard light.

A modified form of this method has been used for observations of the smoke from chimneys in many towns. The method could probably be arranged to fulfil condition (1) by measuring the transparency of air holding in suspension known amounts of matter, but it would not fulfil condition (2).

VIII. The rainfall is caught, and its transparency is compared with a standard scale of values, made by adding definite quantities of soot to distilled water.

This method is open to the same objection as the second method; neither does it fulfil condition (2). The results would depend also on the nature of the impurity, as well as on its quantity.

IX. Dust-boxes are exposed, having a collecting surface of one square foot.

These boxes have been used by Chief Inspector Fyfe in Glasgow and Aberdeen, and are simply a less accurate means of operating the second method.

As already stated the respective advantages and disadvantages of the above nine methods were fully considered by the Committee before coming to a decision, and it was finally decided to adopt Method II, namely, that used by Professor Cohen and by Dr. Des Voeux and *The Lancet* for the investigations upon the soot-fall of Leeds and London, carried out by them in 1910.

A Specification for a *Standard Form of Apparatus and Rules* for the collection and examination of the deposit, have been drawn up.

The standard instrument consists of a circular open-topped gauge-vessel, supported in a galvanized iron frame. This frame is provided with a shelf for carrying a bottle to contain the rain-water which enters the gauge. The gauge-vessel is constructed of cast iron enamelled with a vitreous insoluble enamel. It is of importance that the superficial area of the top of the vessel (included inside the sharp edge) should be constant in all vessels, thus assuring the same catchment area in

each. To provide for this the top edge of the vessel must be machined before enamelling. Fig. 1 shows one of the soot gauges used for the London observations in 1910 with the gauge-vessel and bottle in position. Fig. 2 is a section of the standard gauge-vessel alone, removed from its frame, and Fig. 3 is a plan of the standard gauge. A well-fitted glass strainer is provided as shown.

The stand is formed of the four angle iron legs riveted at the top to two iron rings, the lower of which is to be of suitable size to receive the gauge-vessel and support it. To the upper ring are attached galvanized iron wires of 14 S. W. G. These are fixed at their lower ends to a ring of 1/16" galvanized iron riveted between the legs and the heavier ring which supports the gauge-vessel. A cage is thus formed surrounding the gauge-vessel, but opens at the top. A shelf of 1" pine is provided at the bottom, the boards being screwed to 1 1/2" x 3" battens which fit inside the iron supports riveted to the legs. All metal except the gauge-vessel is galvanized iron.

Each instrument is provided with four bottles of stock pattern and of a capacity of about two gallons each, and having

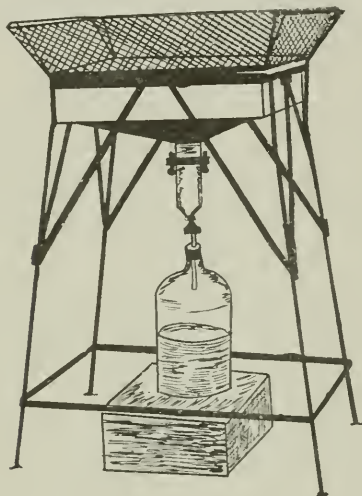


FIG. 1.—SOOT GAUGE

rubber bungs, one of which is fitted with holes for the glass tubes.

As regards the method of using this standard gauge, the following preliminary instructions have been drawn up by the committee:

1. All gauges are to be placed on the ground level, in an open space free from abnormal dust.
2. The bottles containing the water and deposit are to be removed on the last day of each month, and thoroughly cleaned empty bottles are to be substituted.
3. Before removing the bottles for analysis, the gauge-vessel is to be washed down with some of the water already collected, a brush of standard pattern being used to remove any adherent matter.
4. A chemical analysis of the water and deposits is to be made by a standard method, particular attention being given to obtaining the relation between the quantity of tarry oils present and the amount of free carbon.

The examination of the solid matter contained in the bottles may be confined simply to the determination of the total solids (which can be separated either by evaporation or by filtration), or it may cover detailed separation of the soluble and insoluble portions of the impurities collected from the atmosphere by the rain-fall. A sub-committee is at present dealing with the framing of a standard method of examination. This will be used for the observations which are to be carried out under the auspices of the Committee.

For the *Lancet* observations, carried out in London in 1910 at three stations in the London area, the following constituents were determined: Total solids, insoluble deposit, soluble deposit, soluble volatile solids, soluble fixed solids, sulphate am-

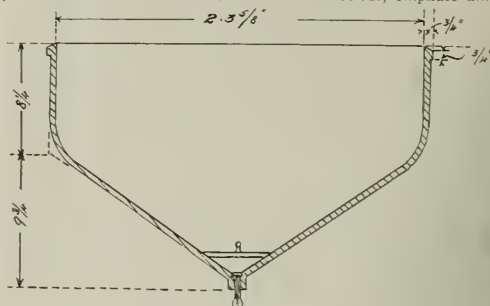


FIG. 2.—SECTION OF SOOT GAUGE

monia, chloride. The Table presented below summarizes the results obtained.

The *Lancet* inquiry was confined, as already stated, to

LONDON'S COAL SMOKE DEPOSIT

Observation Station	Litres of Rain	Total Deposit	Insoluble Matters	SOLUBLE MATTERS					Total Deposit per Square Mile
				Total	Fixed	Volatile	Sulphate SO_4	Ammonia NH_3	
A. Buckingham Gate, S.W.	201.1	71.985	46.335	25.650	13.311	12.339	6.017	5.013	569
B. Horseferry Road, S.W.	192.0	60.339	33.496	26.849	12.324	14.525	5.942	5.133	404
C. Old Street, E.	192.0	92.813	60.920	31.893	19.242	12.651	10.043	7.542	403
D. Sutton Surrey	210.5	27.894	8.365	19.529	8.723	10.806	0.892	0.347	163

NOTE. All results in columns 2 to 9 are expressed as grammes, and represent the totals obtained from the 4 square feet of catchment area, over the 12 months of observation.

observations made at three London stations and to one in the suburban districts of London (at Sutton in Surrey), and its



FIG. 3.—PLAN OF SOOT GAUGE

value for comparative purposes was, therefore, less than would have been the case if similar results had been available from

other cities and towns. The Committee appointed at the International Smoke Abatement Conference of March, 1912, therefore decided to communicate with all the principal local authorities in the United Kingdom, and to ask for their co-operation in an organized attempt to collect reliable data by means of the apparatus and methods of examination approved by the committee for this purpose. It was believed that many of these authorities would welcome the introduction of a standard method of conducting such observations, and that the results would prove of much greater practical value than any number of independent inquiries based on the use of different apparatus and different methods of observation. A circular letter sent out by the Committee in January, 1913, proved that this opinion was correct, and it is now practically certain that the following important cities of the United Kingdom will commence these observations on or before October 1 of the present year, using the standard apparatus and method recommended by the Committee: London, Liverpool, Manchester, Glasgow, Birmingham, Newcastle, Bradford, Hull, Leicester, with others that are now considering the proposal.

The results of the observations in each city or district will be passed on regularly to the London Committee, and an annual report will be published containing all the figures and the conclusions which the Committee arrives at as to the practical progress of the smoke abatement movement in the various cities and towns from which the returns are sent.

Other Methods of Observation.

Turning now to a consideration of the other methods for collecting and examining the solid impurities of the atmosphere, those of Rübner, Renk, Hahn, Liefmann, Cohen and Tyfe will be briefly described.

Rübner's Method.

The Rübner method of soot and dust determination is based upon the aspiration of a large volume of the air which is to be tested (200-300 cubic feet) through a filter paper placed in a special holder, Fig. 4. Upon the surface of this paper the soot and dust are collected. The aspiration is effected by a water-jet pump, the volume of air being measured by an ordinary dry meter. The determination of the amount of soot and dust collected on the paper in 12 or 24 hours is carried out by comparison with a standard scale, since the amount is too small even with a large volume of air used to be determined by direct weighing. Renk has employed for the comparison a mixture of a known amount of soot and oil, which when used in a wedge-shaped glass vessel, gives a color scale of gradually increasing density. This method of estimating the soot and dust in the atmosphere was employed in 1900-1910 by the officials of the Hygienic Institute in the city of Hamburg, the apparatus being installed at five different points in the city. The results of the investigation will be found summarized in the report for the year 1910 of the the Hamburg Smoke Abatement Society.

Hahn's Method.

The apparatus devised by Hahn consisted of a filtering tub and of a double-cylinder pump worked by an electric motor provided with current from accumulators. The volume of air in this case was ascertained by multiplying the cubical capacity of the two cylinders of the pump by the number of revolutions of the crank shaft as recorded by a counter attached to the same. The soot and dust were retained by drawing the air

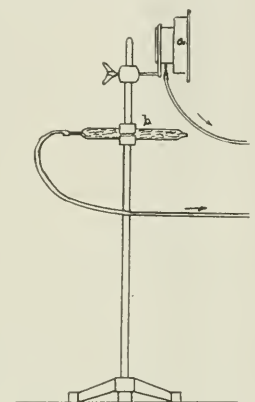


FIG. 4—THE RÜBNER & HAHN APPARATUS

through a tube containing collodion-wool, shown in Fig. 4.

The weight of solid matter was estimated by dissolving the collodion-wool with its deposit of soot and dust, in a mixture of two parts of ether and one part of alcohol. The final step was to compare the turbid fluid thus obtained with various standard solutions containing collodion and known amounts of impurity.

Liefmann's Method.

Liefmann's method differs essentially in principle from the two methods described above since it is based upon "sedimentation" of the soot and dust particles, upon prepared plates, and not upon filtration. The method is described in considerable detail in the thesis published by Liefmann, on taking his doctor's degree at Halle University in 1907. Considerable space is devoted in this pamphlet to the theory of sedimentation as a means of measuring the soot and dust particles suspended in the air; and to the various reasons why two surfaces are necessary (a vertical one and a horizontal one) in order to obtain correct results. The practical operation of the method only can be described in this article. The apparatus required (Fig. 5), is simpler than in the case of the Rübner and Hahn method, and it is therefore more easily moved from place to place, or erected in any desired point of observation.

A firm three-footed base of cast iron, provided with a vertical rod or spindle carries a horizontal plate, upon the edges of which are fixed four thin uprights, carrying a light roof or cover. Under this cover is fixed the vertical disc of glass, covered with a thin film of oil, for catching the drifting particles of soot and dust. These drifting particles would not be caught by the horizontally placed disc, and in this respect Liefmann's apparatus and method, with a vertical as well as a horizontal disc, differs from that used by Cohen for the observations carried out at Leeds, and reported in the Proceedings of the Smoke Abatement Conference held in Manchester in November, 1910.

The results obtained by Liefmann at Hamburg cannot therefore, be compared with those obtained by Cohen at Leeds, although obtained by a method depending upon the same principle of sedimentation. This defect in the two sets of observations shows how necessary it is that some standard method

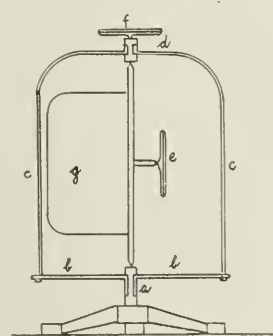


FIG. 5—LIEFMANN'S APPARATUS

and apparatus should be used, in order to obtain the greatest value from observations upon the soot and dust falls in the various centers of population and industry.

Liefmann's observations were carried out in Hamburg in the winter of 1903-1904 and the total deposit varied between 10 and 200 milligrams per square metre of exposed surface per 24 hours according to the direction of the wind and humidity of

the atmosphere. These totals are equivalent to 15 tons and 283 tons per square mile per annum.

Cohen and Ruston's Apparatus and Methods.

The observations upon the pollution of the atmosphere in Leeds (referred to above) were carried out by three methods which are similar in principle to those that have been described. In one set of observations the rain-fall was collected, and the insoluble and soluble impurities were separated and examined.

This is the method that has been selected by the London Committee for its investigations, and the only improvement they have made in Cohen's apparatus is that of increased size, the funnel or collector having an area of 4 square feet, instead of 0.7854 of a square foot.

In another set of observations snow was collected after a

heavy downfall, and the amount of solid impurities it contained was estimated. The third set of observations were made by the sedimentation method, clean glass plates one foot square being exposed horizontally, and the sticky, tarry matter which adhered to them being weighed. Cohen and Ruston's figures for the soot and dust fall of Leeds and the surrounding suburbs varied from 25 to 539 tons per square mile per annum, the lowest figures being that of Roundhay (a country district to the southwest of the city), and the highest figure that of 539 tons for Leeds Forge, in the heart of the manufacturing district of the city.

Fyfe's Method.

During the winter of 1910-1911, Chief Inspector Fyfe, of the Glasgow City Sanitary Department, carried out a series of observations upon the soot and dust fall in Glasgow and other Scotch towns, by means of "atmospheric dust-boxes." These boxes were shallow wooden boxes, one foot square in superficial area and one foot in depth. They were exposed on the roofs of buildings in suitable situations, at a height of 30 to 35 feet above the ground. After two months' exposure the boxes were collected and the contents were carefully weighed and analyzed by the Glasgow City Corporation Chemist. The figures obtained by this very simple and crude method varied from 2.27 cwt. per area per annum at Bo'ness on the Firth of Forth, to 19.6 cwt. per acre at Falkirk, equivalent to 72 and 627 tons per square mile, respectively.

Benner's Method.

For the smoke investigation which is now being carried out by Benner and others in U. S. A. under the auspices of the Department of Industrial Research of the University of Pittsburgh, Benner has used glass cylinders 4 inches in diameter and to inches deep, which are exposed on the roofs of buildings four stories high, in various parts of the city. The water and dust which are collected in this way are evaporated to dryness; the insoluble residue is then weighed and extracted with ether. From the percentage of tar obtained in this manner that part of the total solids due to soot can be calculated. No figures have yet been published for the results of the Pittsburgh observations.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

Institution of Mining and Metallurgy.

At the last meeting of the Institution of Mining and Metallurgy Mr. Herbert K. Scott read a paper on "Bulgarian Mineral Deposits," and said hitherto mining in Bulgaria could not be accounted very successful, although a favorable mining code and a sympathetic government had greatly assisted it. But when the war, the fear of which had for many years deterred commercial enterprises, shall have been concluded, it is only reasonable to suppose that the development of the mining resources of Bulgaria would be rapidly and successfully achieved. In ancient times the mines of Bulgaria appeared to have played an important part, if the extent of the old workings and the accumulation of slag at various points were any criterion. But until 1878 they were almost neglected.

The State in that year began operations on a lignite formation near the village of Mochino, about eighteen miles from Sofia, and the output was entirely taken up locally. Another lignite deposit in the same district was started on in 1891. The mining law of 1892 made generous provisions and considerable activity resulted; and from that year up to 1907 a great number of prospecting licenses was given, and some thirty-one mining concessions were granted by the government during the same period. The State reserved to itself the lignite formations Pernik-Mochino-Vladja and Bobov-dol, south of Sofia, as well as the Belnorvzh coal deposits in the central Balkans, and the deposit of lead in the Rhodope

mountains near Lacavitzza. Although a large number of licenses and concessions was granted, operations were being carried on in only three mines in 1909; namely, at Pernik, near Sofia, for lignite; at Plakalnitza, near Vratza, for copper ore, and at Blagodat, near Kustendil, for lead, copper and zinc.

Mr. F. White discussed "Errors in Sampling and Assaying Ores Due to the Presence of Coarse Gold." He contended that the discrepancies which frequently existed between the quantity of gold actually recovered and the quantity which should have been recovered by direct cyanide treatment of low grade gold ore demanded a complete investigation of the circumstances affecting the estimate of the gold contained in the charges and also of that remaining in the residues. The author decided that the cause of the variations shown in triplicate assays must be looked for in the final sample.

After careful examination it was found that the variations were mostly due to the presence in some of the samples either of coarse grains of gold or of flaky re-precipitate gold; but these particles were not distributed through the whole of the charge of ore in the same proportion as in the ore used in the fusion. This was more noticeable in the charge than in the residue, because the large grains of gold would mostly be dissolved by the cyanide treatment.

Tests were made to ascertain whether the grain or two of coarse gold which might have given a high figure in one of the triplicate assays existed in the same proportion in the whole of the charge, or whether those grains were infrequent. From these experiments the author concluded that where there were only a small number of gold grains of unusually large size, the assay sample should be passed through a 30 or 40 mesh sieve only. Triplicate fusions should be made; and where any serious variations occurred they should be repeated. Abnormally high results should be rejected in taking the averages. In such cases it was reasonable to conclude that there would be a tendency for the "average" to exceed the real value.

Institution of Mechanical Engineers.

In his recent presidential address Sir H. F. Donaldson dealt largely with the conditions necessary for the advancement of the Institution and the necessity for practical training. He said that the recent action of the Institution in deciding to ascertain by examination the extent of the scientific knowledge possessed by candidates for admission as graduates or associate members would in the future have a very wide effect. A system of examination was at present the only means of ascertaining the candidate's knowledge; but for a practical profession like theirs it was always open to suspicion that the knowledge shown by examination was not of that stable character which was desirable, but might have been obtained by the pernicious process of cramming. It rested, therefore, with the Council and its collaborators to arrange the examinations so as to prevent as far as possible the system of cramming, and to make sure that the knowledge displayed by the candidates was sound.

He was inclined to take teachers to task for some of the advice they gave explicitly or tacitly to students. Many young men had the idea when they had gone through a course of technical study and had possibly displayed great ability, that that, and that alone, entitled them to look for a good position in engineering works. That idea was quite erroneous unless they had had practical instruction as well. It would probably be found that employers looked on such half-trained men with suspicion, and might have actually suffered from engaging such men, and might have been led to say that although they wished to have men trained both theoretically and practically they could not employ a man who had theoretical knowledge only, and that if they could only get men trained on one line they would take the practical man and not the theoretical.

The President personally desired the services of men who were thoroughly trained, because he had had experience of their value; but he was sorry he could also speak from experi-

ence in saying that the practical workshop was too often neglected, and he was certain that the omission of that was disastrous to the student himself as well as to his profession. Some of the blame must attach to the teachers, and he hoped that any who were responsible, directly or indirectly, would recognize that if they desired, as, of course, he thought they did, the co-operation of employers in advancing the interests of their students, they must avoid the inculcation of such wrong ideas and impress on their students that to be qualified for a practical profession they must go through a thoroughly practical as well as a thoroughly theoretical training.

It would seem that those who were responsible were teachers who had not themselves gone through the workshops, or if they had taken up with teaching before gaining adequate practical experience. Such men would be thoroughly conversant with theory and fully competent to teach on this side of the subject, but when they lacked practical training it was inevitable that they lacked also the power of imparting knowledge of the application of theory to practice. He felt strongly that a man with only technical and no practical training was a long way off from being a practical engineer; and also that a man with practical and no technical training was not sufficiently equipped to carry out the engineering work of today.

The Institution should not concern itself solely with its internal affairs. Just as each man had duties to the State, to his profession and to his institution, so the institution had similar duties, and its work should include not only its own interests but also the throwing open to the general advantage the results of its work. Of course some *kudos* attached to successful work when its useful results were fully published, but the institution should be willing to forgo that *kudos* if by combination with other institutions better co-ordination could be obtained. The institution spent a certain amount of money on research, and it was possible that part of the money which they devoted to individual research might be better spent in supporting the work of a central engineering research committee, and that other institutions might also be willing to make grants for a similar object. The advantages which would result from this course would be the prevention of a considerable amount of over-lapping work, individual research would be carried out to definite results, such results would be published and such publications would include accounts of private researches conducted by those who associated themselves with the engineers' research committee.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the three months ended on March 31 show very satisfactory advances, except in imports of electrical goods and new ships, as compared with the corresponding period of last year. Iron and steel, including manufactures, were imported to the value of £3,067,780, an increase of £1,057,803, and were exported to the value of £13,321,611, an increase of £682,203. Imports of other metals, including manufactures, amounted to £8,220,004, an increase of £547,752, and exports totalled £3,429,356, an increase of £576,751. In electrical goods the imports were £374,330, a drop of £36,022, but exports rose to £1,510,256, an increase of £349,518. Imports of machinery figure at £1,833,892, a rise of £201,400, and exports went up to £8,500,063, an increase of £777,593. Imports of new ships are put at £2,000, a decrease of £7,967, while exports are valued at £2,030,400, an improvement of £863,761.

Taking the returns for March by themselves, imports of iron and steel rose by £306,106 and exports by £23,071; imports of other metals increased by £103,862 and exports by £32,035; imports of electrical goods fell £10,080 but exports rose by £153,015; imports of machinery were £31,075, but exports dropped £34,212; imports of new ships declined £0,472, while exports showed an improvement of £452,054.

Market Prices, April, 1913.

Copper was a rising market until the 14th, when it touched £60; has since been weaker. Opened at £66, closes £67.10.0.

Tin has been on the whole a rising market, fluctuating up to £231½ (18th); it closes £227½.

Lead rising the first half of the month from £17 to £19, it has afterwards been easier, closing slightly lower at £18.10.0.

Haematite has been very steady at or about 79/-, at which price it closes.

Scotch Pig showed depression from 3rd to the 14th, then recovered on 18th, and since stronger closes 73/3.

Cleveland has been through the same fluctuations as Scotch Pig. It closes 67/3.

	£ s. d.
Aluminium, ton (ingots).....	95. 0. 0
Alum, lump, loose, per ton.....	6. 0. 0
Antimony, ton.....	20. 0. 0
Borax, British refined, crystal, ton.....	17.10. 0
Copper ore, 10 to 25%, per unit.....	12/4½ to 12.10½
Copper sulphate, per ton.....	23. 7. 6
Carbolic acid, liquid, per gal.....	1. 4
Creosote, ordinary, good, liquid, per gal.....	2½
Caustic soda, 70%, per ton.....	9.12. 6
Ebonite rod, per lb.....	4. 6
Hydrochloric acid, cwt.....	5. 0
India rubber, Para, fine, lb.....	3. 4½
Mica, in original cases, medium.....	3/6 to 6. 0
Naphtha solvent, 90% at 160° C.....	10½
Petroleum, Russian spot.....	9½
Quicksilver, per bottle.....	7.10. 0
Sal-Ammoniac, lump, 1st delivered U. K. per ton.....	44. 0. 0
Sulphate of ammonia, f. o. b. Liverpool, per ton.....	13.17. 6
Sulphur, recovered, ton.....	5. 5. 0
Shellac, per cwt.....	4. 5. 0
Platinum, oz.....	9. 5. 0
Tin ore, 70%, per ton.....	148 to 150. 0. 0
Zinc, Vieille Montagne, f. o. b. Antwerp, per ton.....	28.12. 6

Higher.

Lower

	£ s. d.		£ s. d.
Aluminium.....	6. 0. 0	Caustic soda.....	7. 6
Alum.....	10. 0. 0	Ebonite rod.....	0
Copper ore.....	9	India rubber.....	1½
Copper sulphate.....	7. 6	Quicksilver.....	5. 0
Petroleum.....	1	Sulphur.....	5. 0
Sulphate of ammonia.....	7. 6		
Tin ore.....	5. 0. 0		
Zinc.....	5. 0		
Copper.....	2.12. 6		
Tin.....	9.10. 0		
Lead.....	1.18. 0		
Cleveland.....	6. 0		

Recent Chemical and Metallurgical Patents

Iron and Steel

Electric Steel.—Another patent has been granted to Mr WILLIAM R. WALKER, of the United States Steel Corporation, for the combined Bessemer converter and electric furnace steel process. The particular feature of this patent is the re-carburizing of the metal decarbonized in the Bessemer converter, the re-carburizing taking place in the ladle before the metal is transferred to the electric furnace. The process is in detail as follows:

Molten pig metal is introduced into a Bessemer converter and blown until the silicon has been eliminated and the carbon reduced to a definite percentage, preferably to about 0.05 per cent, which percentage can be gaged with accuracy because it is about the percentage contained in the metal at the time when the flame "drops," a phenomenon which is well known and readily recognized by the skilled blower. The disiliconized and substantially decarbonized metal a proportion of carbon is then added sufficient to raise its carbon content to a definite percentage, preferably to about 3 per cent, though this percentage may be varied. The carbon addition is made by adding to the Bessemer metal in the ladle, pig iron either in the form of cold pig or molten pig, and as the percentage of carbon of

the Bessemer metal is substantially constant, the amount of pig metal to be added in the ladle can be readily determined with substantial precision. The carbon may be introduced by adding anthracite coal or powdered coke to the metal instead of adding carbon in the form of pig iron. If pig iron is added, it is preferably in the molten condition. The metal thus enriched in carbon is then charged into an electric furnace where it is treated with basic additions for removal of sulphur and phosphorus, and for increase of its heat to bring it into refined and heated condition proper for casting into the ingot molds.

The importance of adding carbon to the blown metal before introducing it into the electric furnace is that in this way the loss by skulking is lessened, and by bringing the metal to a definite carbon content the heats in the electric furnace are made uniform in character and duration, the product rendered

efficient handling of such product, form the basis of a patent granted to Mr. DAVID T. CROXTON, of Cleveland, Ohio. The inventor provides shallow pits near the furnace, of such capacity as to hold the slag produced in 24 hours. Arranged on the bottom of the pit is a parallel series of flexible members, such as chains, attached to pins at each side of the pit. The slag is allowed to run uniformly over the pit, each successive charge of slag forming a layer over the previous one. When the pit is full, each chain is successively detached from one of its retaining pins, and lifted or ripped upward through the bed of slag. The latter is disintegrated and then removed from the pit by a grab bucket or other suitable device. Slag produced by this method is claimed to be in good condition for use as road ballast or an aggregate for concrete. (1,056,632, March 18, 1913.)

Another invention having a similar object in view is that of Mr. GEORGE L. DANFORTH, Jr., of Chicago, Ill. He provides a slag receptacle consisting of a long metal trough, having a slot-like opening running lengthwise along the curved bottom. This slot is normally closed by strips of metal, designated agitators, attached to vertical arms by means of which they may be raised or lowered. Slag is run into the receptacle, forming a layer, which is then sprayed with water to cause shrinkage and cracking. When the first layer is cool another charge is run on, and this action continued until the receptacle is full. The agitators are then lifted by a crane, disintegrating the slag, and allowing it to run out through the slot in the bottom of the receptacle. (1,058,158, April 8, 1913.)

Manganese Steel.—Mr. ALBERT E. GREENE, of Chicago, Ill. produces manganese steel or other alloy steels by charging the solid ferro-manganese into a suitable vessel, then pouring enough molten soft steel of proper composition on top of it and completing the melting and heating in a reducing atmosphere by means of electric heating. (1,060,078, April 29, 1913.)

Steel for Electrical Purposes.—Sir ROBERT A. HADFIELD, of Sheffield, England, the well known pioneer in the manufacture of alloy steels, has covered in an older U. S. patent 745,829 of Dec. 1, 1903, the heat treatment of an iron-silicon alloy for the purpose of obtaining a magnetic material of high permeability, high electric resistance and low hysteresis. He now claims a process of making this alloy of improved solidity and homogeneity of structure. It will be especially adapted

for use in ballast coils, transformer plates and like electrical apparatus, wherein the total magnetic and electric losses have to be reduced to the lowest possible degree. The essential feature of the process is that 50 per cent ferro-silicon is placed in a ladle or other vessel and raised to a red heat. The pure molten iron is poured upon the red-hot ferro-silicon and fuses the same. Inasmuch as the latter substance fuses about 300 deg. to 400 deg. C. below the iron, the additional heat units required to dissolve the silicon-alloy are small and readily communicated to it from the iron itself. Just before casting in ingots a suitable proportion of aluminium, not exceeding 1 per cent, or other equivalent deoxidizer, is added to the mixture. (1,061,373, May 13, 1913.)

Charging Apparatus for Blast Furnaces.

—Messrs. E. J. WINDSOR-RICHARDS and T. LEWIS, of Glenarnock, Scotland, patent charging apparatus of the hopper type and inclined railway for charging buckets. Characteristic features are the small amount of headgear and parts projecting from the top of the furnace roof plate. (1,061,349, May 13, 1913.)

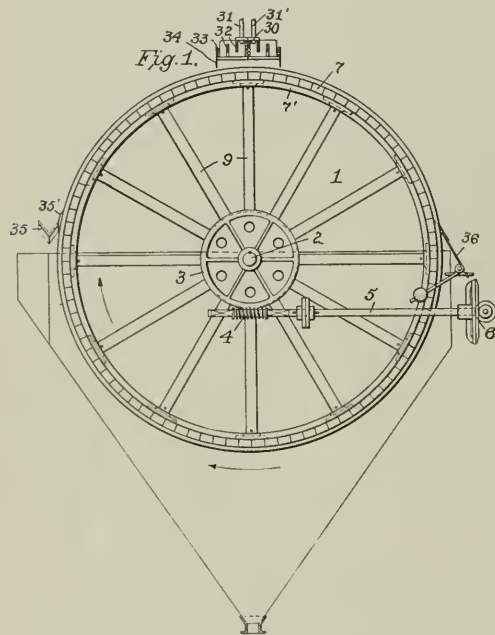


FIG. 1.—ROTARY FILTER FOR ORE SLIME

more uniform and the manufacture thereby greatly facilitated.

As the decarbonized metal is low in carbon when taken from the converter and is often transferred a considerable distance to the electric furnace, a considerable amount of this metal will solidify in the ladle and form skulls therein. This skulking action will reduce the amount of metal poured into the electric furnace from the ladle, and as the skulking action will vary, the amount poured in will vary. By adding carbon to the metal before its being carried to the electric furnace, its carbon content is raised and its melting point thus lowered, and the amount of skulking during the transfer to the electric furnace is prevented or greatly lessened. This, therefore, makes the charges more uniform in amount as poured into the electric furnace. The obtaining of a uniform carbon content is rendered easy by adding carbon to the blown metal rather than by attempting to stop the blow at the desired point, for to determine the point of stopping before the flame drops is very difficult, while to add the necessary percentage of carbon to the bath depleted of carbon is easy. (1,135,538, February 25, 1913.)

Methods of Handling Slag.—The production of blast or open-hearth furnace slag in a granular condition, and the

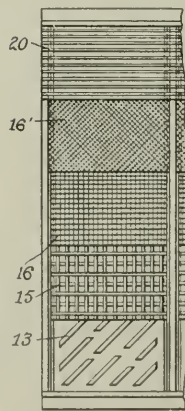


FIG. 2.—DETAIL OF FILTER SECTION

Gold and Silver

Rotary-Drum Vacuum Filter.—Figs. 1, 2, 3 and 4, show details of a revolvable filter for use in the cyanidation of gold ores, designed by Mr. RANDALL P. AKINS, of Denver, Colo., and assigned by him to the Colorado Iron Works Co., of the same place. Fig. 1 gives a side view, showing a drum 1

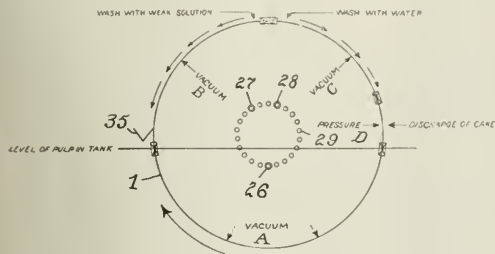


FIG. 3.—CYCLE OF FILTRATION

mounted on an axle 2 which is driven by a worm wheel 3 and worm 4. The face of the drum is composed of filter sections, as shown in Fig. 2, wherein 13 indicates strips of wood fixed to the face of the drum, supporting coarse wire screen 15, on which are placed successive layers of burlap 16 and canvas 16'. Outside of these layers of filtering material is wound a series of wires 20 to hold the filtering medium in place. Each of the filter sections has separate pipe connections to the valve in the hub of the drum.

The valve is designed to permit the separate withdrawal of gold and wash solutions, and to conduct compressed air to one section at a time for the purpose of dislodging the cake from the filter. The cycle of the various steps is shown in Fig. 3. That portion of the filter below the line of pulp in the tank is under vacuum, which results in the withdrawal of valuable solution and the formation of a slime cake on the surface of the filter. As the drum revolves, the rising side is washed or sprayed with weak cyanide solution from a box or pipe placed at the top of the filter, as shown in Fig. 1. This wash solution is removed by vacuum and delivered separately from the first solution. A similar wash, with a separate removal, can be applied from the box or pipe to the face of the filter as it travels downward into the tank again.

The controlling valve situated at one end of the hub consists of a movable plate and a stationary plate or member. The former is attached to the drum and has a circular line of openings connecting with the pipes leading to each separate

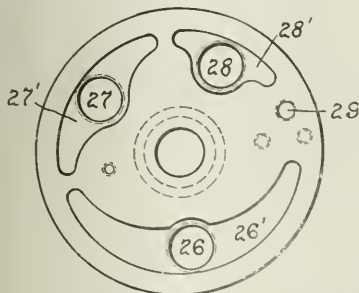


FIG. 4.—STATIONARY MEMBER OF FILTER VALVE

section of the filter. The stationary member of the valve, shown in Fig. 4, has three slots, 27, 27', and 28', and one circular opening 29. As the filter revolves, the openings in the movable member of the valve are continually registering with the different slots in the stationary member. Those registering with slot 26', as at 26, are under vacuum and are delivering

gold solution to a suitable place, those registering with slots 27' and 28', as at 27 and 28, are also under vacuum and are removing the different wash solutions, while that opening in the movable member which registers with the opening 29 of the stationary member receives compressed air for the purpose of dislodging the cake on that particular section of the filter, after it has been washed. (1,059,327, April 15, 1913.)

Another form of filter, patented by Mr. HARRY E. KIER, of Colorado Springs, Colo., is shown in Fig. 5. It is designed for use in the cyanidation of gold and silver ores and is adapted to leaching and filtering. It consists of a tank with a spherical bottom of cellular construction, forming a filter. Within the tank is a revolving member composed of perforated pipes, having bearings outside the tank, and being driven by power applied to a pulley. Air can be delivered into the perforated pipes through a connection in the right-hand bearing and thus caused to mingle with the slime pulp in the tank.

In operation, slime pulp is delivered into the tank, aerated and kept in suspension until all valuable metal is dissolved. Filtration can proceed during this operation, the revolving arms

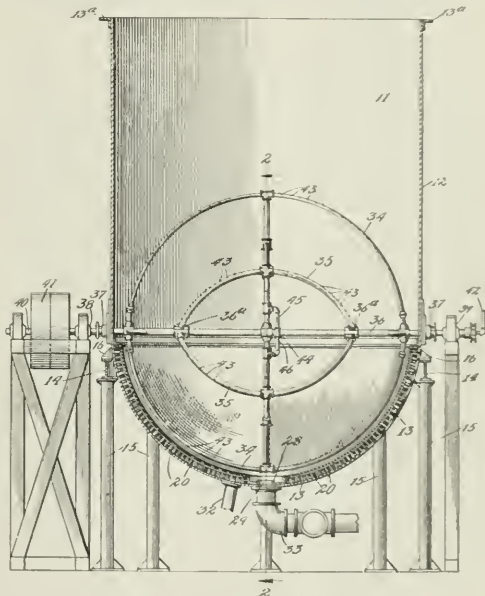


FIG. 5.—LEACHING FILTER TANK

keeping the filter surface free from impervious accumulations. The pulp can be washed with successive solutions and discharged through a pipe in the bottom of the tank. (1,058,860, April 15, 1913.)

Lead, Copper and Zinc

Production of Minium.—According to specifications of a patent recently granted to JAROSLAV MUEHLHAUER, of Prague, Austria-Hungary, it is possible to greatly increase the rate of production of minium, PbO_2 , by carrying on the customary process of oxidation under a pressure of several atmospheres instead of at ordinary pressure. Minium is ordinarily produced by roasting oxide or carbonate of lead. The process is slow and the material must be roasted in thin even layers for periods ranging from 24 to 48 hours. The reaction cannot be accelerated by increasing the temperature, but the inventor has found that by increasing the pressure under which the oxidation takes place, say to 12 atmospheres, he can accomplish within one hour the same effect produced at ordinary pressure in 15 hours. By using pure oxygen under pressure the reaction occurs in a few minutes. (1,059,195, April 15, 1913.)

Extracting Tin from Base Bullion.—An improved process for removing tin from lead or base bullion is due to BRIAN C. BESLEY, of Howell, N. S. W., Australia. A reverberatory furnace is charged with a certain quantity of rock salt. When the salt is molten, lead or base bullion is introduced, sinking beneath the salt cover. Oxide of lead, preferably litharge, is then added to the bath, in the ratio of two parts litharge to each part of tin in the bullion. After about half an hour

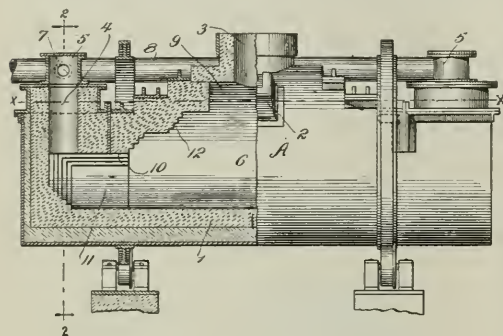


FIG. 6.—COPPER CONVERTER WITHOUT TUYERES

all the tin has become associated with the litharge, whereupon the refined bullion is run out of the furnace, followed by the salt, leaving the solid tin-lead oxide combination to be removed in lumps. The latter is then smelted for the recovery of a marketable tin-lead solder. The salt can be used repeatedly. (1,060,527, April 29, 1913.)

Copper Converter Without Tuyeres.—In Fig. 6 is shown an elevation, partly in section, of a form of copper converter patented by Mr. HOWARD B. JERNEE, of Perth Amboy, N. J. The principal object sought in this type of construction is the elimination of the tuyeres of the ordinary type and the troubles incident to their use. As seen in Fig. 6, the converter consists of a cylindrical body A, rotatably mounted, and having an opening 3 for the reception of ore and discharge of gases. Air boxes 5 are placed at each end of the shell, and are connected with an air pipe 8. Air inlets extend downward from the boxes into the ends of the converter, as at 4. The body of the converter is lined with refractory basic material 1, which is so arranged that the inner chamber is contracted at the ends. The bottom surfaces 10 of the outer ends of the top wall of the lining 1 are preferably straight or flat, and extend well down toward the bottom wall of the lining, forming baffles 12, which serve as deflecting surfaces to increase the efficiency of the air.

In operation, the converter chamber is filled with matte, and air is forced through the passages 4, flowing through the body of the metal and finally escaping through the opening 3. (1,059,367, April 22, 1913.)

Electrodeposition and Refining of Zinc.—A method of electrolyzing a strongly acid solution of a zinc salt with a current of high density is patented by URLYN C. TAINTON, of Manchester, and JOHN N. PRING, of Sandbach, England. The inventors prefer a sulphate solution of zinc, with from 10 per cent. to 30 per cent. of sulphuric acid, and 0.1 per cent. to 1 per cent. gum tragacanth. This is electrolyzed with a current density of 500 amperes per sq. ft. of cathode surface. Densities of 1/5 this amount can be used with success, but time is saved by the higher density.

The novel features of the process are: the high current density employed, the production of a smooth, lustrous and compact deposit, the use of a colloid agent, such as gum tragacanth, the wide limits of acidity, and its adaptability to plating on aluminium. The method is specially useful in galvanizing wire electrically. (1,059,233, April 15, 1913.)

Reduction of Zinc Oxide.—The usual practice in zinc smelting consists in making an intimate mixture of some form

of zinc oxide and carbon, and subjecting this mixture to a high temperature in small retorts. The method is wasteful of heat, does not effect complete reduction of the zinc, and entails considerable expense for retorts. A general feature of all proposals for zinc smelting, is the intimate mixture of the ore and reducing agent. In contrast to this feature is the proposal of Prof. CHARLES F. BURGESS, of Madison, Wis., whose process consists in keeping the zinc ore and carbon entirely separate from each other. In the preferred form of his process, the zinc ore is heated electrically, and the zinc oxide thereby volatilized. Simultaneously a mass of carbon is subjected to a current of electricity to heat it to a proper reducing temperature. The vapors of zinc oxide are then passed through the mass of incandescent carbon, and the reducing action is quickly and thoroughly accomplished.

A form of furnace in which the operation can be carried out is shown in Fig. 7. The zinc vaporizing chamber is shown at A, and the carbon container at B. Electrodes 4 in the ore chamber provide the necessary heat for volatilization of zinc oxide, and electrodes 9 and 11 in the carbon chamber are used to bring that body to the proper temperature. A passage 8 provides an exit for the volatilized oxide into the reducing chamber. Slag may be removed from the ore chamber by the tap 7; ashes can be removed from the carbon chamber through the door 18. Flues 16 and 17 provide for the exit of the reduced zinc vapor. The consumption of carbon in this form of furnace is very limited, as it is confined practically to that required in the reduction of ZnO to Zn.

The advantage of this kind of a furnace is very considerable, inasmuch as the temperatures for volatilization of zinc oxide and its reduction to zinc are not the same, the proper temperature for the carbon being considerably lower than that which is required to volatilize the oxide. Mr. Burgess has found by experiment that zinc oxide begins to volatilize from ore at a temperature of about 1100 deg. C. This, therefore, is the minimum temperature at which the ore chamber should be maintained. By heating at a higher temperature, the action is carried on more rapidly, but with volatilization of impurities

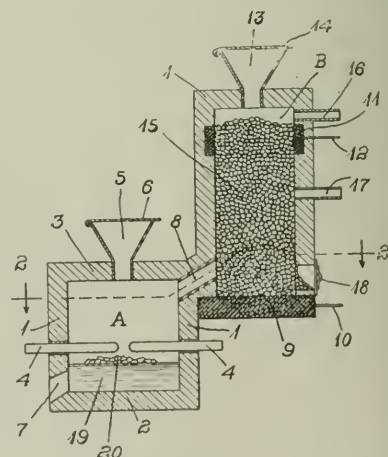


FIG. 7.—ELECTRIC FURNACE FOR VOLATILIZATION AND REDUCTION OF ZINC OXIDE

which are undesirable. Experiment has shown that zinc oxide can be completely volatilized from ordinary roasted zinc ore by the method just described. (1,059,342, April 22, 1913.)

Roasting Furnaces

Construction of Multiple Hearth Furnaces.—In a patent recently granted to REINHOLD SCHERFENBERG, of Berlin, Germany, the inventor presents certain improvements in furnaces of the McDougall type. In ordinary construction, the passages

for the downward movement of roasted ore are used also for the upward passage of air from one hearth to the next. In the present case, the inventor introduces air free from SO_2 to the several hearths through the central tube of the furnace, and separate conduits distributed at the circumference of each hearth to convey the gases enriched with SO_2 directly to the uppermost hearth. Means are provided for regulating the passage of the air and gases. (1,057,584, April 1, 1913.)

Another patent of similar intent is that granted to MR. UTLEY WEDGE, of Ardmore, Pa. His improvement consists in placing flue casings in the side walls of the furnace, to divert the draft from the passages for the downward flow of ore. The casings extend through the wall, so as to be accessible from the outside. (1,060,389, April 29, 1913.)

Dust Collection and Filtration

A dust collector suitable for use in sulphuric acid work, for the separation of dust from the gases coming from the roasters, is shown in Fig. 8. It is the invention of Mr. WILLIAM T. HARDER, of South Baltimore, Md. It consists of a series of frames containing a large number of shelves, placed in the flues leading from the roasters. The normal position of the shelves in the flue is horizontal, in which position they gather the dust from the gases passing between them. The frames are adapted to be rotated, so that their load of dust can be dumped into suitable hoppers in the bottom of the flue. In the figure, one of the frames is shown partly rotated to the dumping position, and one in the dumping position. At the

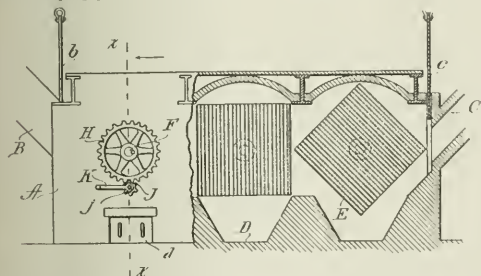


FIG. 8.—DUST COLLECTOR FOR SULPHURIC ACID WORKS

left is shown the mechanism for changing the position of the shelves. It is understood that the flues are in duplicate, so that one can be closed while dumping the shelves in the other.

A filter medium suitable for use in filtering acid gases coming from smelting or roasting furnaces, has been patented by Dr. FREDERICK G. COTTRELL, of Berkeley, Cal. Dr. Cottrell has found that the deterioration of cotton and other vegetable fibers in acid gases is due chiefly to the esterification and hydrolysis of the cellulose which is itself conditioned by the carbohydrate grouping of that body. By breaking down the groups of cellulose without seriously disturbing the mechanical strength of the fibers, a fabric can be obtained which has great resistance to the action of acids. This breaking down of the cellulose groups can be accomplished by heating the fabric, preferably out of contact with air, at a temperature above 200 deg. C.

As an example of the results obtained by this treatment, the following is cited: Cotton fabric weighing 350 g. per square meter and having a tensile strength of 2300 kilos per meter, were heated in a closed vessel to maximum temperatures of 300 to 340 deg. C., for periods ranging from 10 minutes to 3 hours. After cooling in the closed vessel they were found to be perfectly flexible and of a uniform dark brown to jet black color. The samples lost from 68 per cent to 76 per cent of their weight, and showed a tensile strength of 30 to 85 kilos per square meter. They withstood boiling in sulphuric acid without imparting any color to the latter. The inventor prefers to use this fabric in the form of bags, as for bag houses, and to treat the bags after they have been made. After the heat

treatment the fabric will no longer burn with a flame. (1,060,065, April 29, 1913.)

Electric Furnace Processes

Regulating electric furnaces.—Most of the regulators for arc furnaces, based on the application of an auxiliary current to raise and lower the electrodes, suffer from the inconvenience that the regulator cannot be stopped at exactly the desired posi-

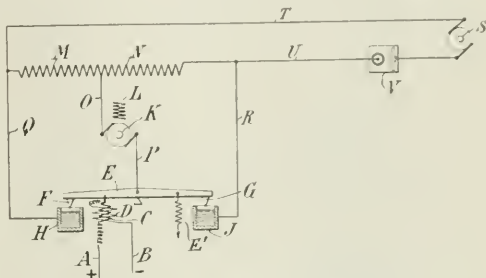


FIG. 9.—CONNECTIONS OF ELECTRIC FURNACE REGULATOR

tion. The momentum of the moving parts causes them to move too far until the opposite regulation takes place, thus resulting in a continual swinging of the parts, back and forth past the point of perfect regulation. A new invention of Dr. PAUL L. T. HÉROULT, of La Prar, France, overcomes this difficulty by an additional device inserted into the auxiliary circuit and allowing to break same a sufficient length of time—which can be regulated—in such a way that the regulation of the furnace is effected step by step and to enable the new position of the furnace to have its effect upon the regulating apparatus.

Fig. 9 is a diagrammatic view of the electrical connections. A and B represent the furnace circuit, C a movable, D a fixed coil. C is mechanically connected to the lever E, the ends of which F and G can dip into mercury cups H or T. K is the motor which actually moves the regulable parts of the furnace. M, N, are the two parts of its armature resistance. The current generated in S passes through M or N according to the position of the lever. So far the arrangement has been known before and its working does not need to be explained. The special feature of the invention is the apparatus V, of Fig. 1, inserted in the current leads T or U at any point common to both the forward and the backward regulating currents. It is especially illustrated in Fig. 10. A ring of copper or other con-

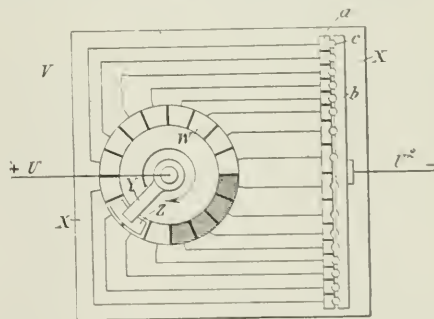


FIG. 10.—DETAILS OF REGULATOR

tacting segments W', insulated from each other is mounted on an insulating plate X; a rotating arm Y, carries a brush Z which passes over the segments W'. Y is connected to one branch U' of the circuit direct, the segments are connected to the other branch U'' individually over contact blocks a, plugs c and conducting bar b. The entire device is connected to a small motor I (Fig. 11). The speed of this motor rotating the

arm *Y* and the number of segments plugged can be varied and so adjusted as to allow the most efficient furnace regulation. Fig. 11, showing an assembled design without the auxiliary wiring is self-explanatory. (1,061,612, May 13, 1913.)

Manufacture of Silicon.—Messrs. THOMAS B. ALLEN and FRANK J. TONE, of Niagara Falls, N. Y., assignors to the Carborundum Company, have greatly improved the efficiency of the arc furnaces previously described by Mr. Tone in U. S. P., 921,183 of May 11, 1909, for the production of silicon. As the larger volume of carbon monoxide gas escapes along the electrodes carrying with it the vapors of silicon and its oxides, the

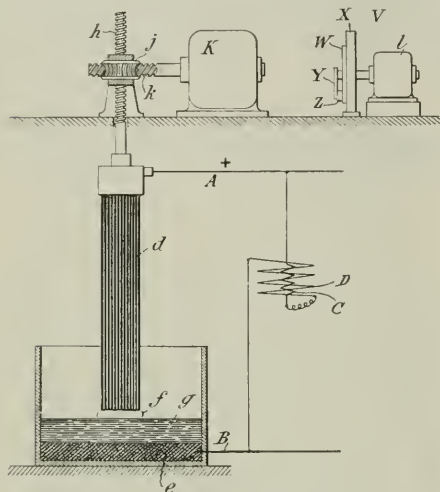


FIG. 11.—ELECTRODE WITH REGULATOR

furnace mixture is finely divided, for example, 12 mesh and finer, and formed into briquets of preferably spherical shape. The interstices then are so evenly distributed over the furnace section that the silicious vapors are condensed all over in the cooler parts of the charge and the carbon monoxide gas being discharged quite uniformly throughout the entire mass preheats the charge more efficiently. The briquets are made from 70 parts pure silica, 30 parts of petroleum coke mixed with 7 per cent hard pitch; 1½ in. has been found to be the best diameter. (1,061,256, of May 13, 1913.)

Electrolytic Processes

Manufacture of Persulphates.—It is generally known that the direct electrolysis of potassium-sulphate produces persulphate only with very poor yields and that therefore mostly ammonium-persulphate is produced from the sulphate by electrolysis in cooled acid solutions and reacted on by potassium-carbonate. From the resulting ammonium-carbonate the sulphate must be regenerated by the aid of sulphuric acid. Messrs. G. ANOLPH and A. PIETSCH, of München, Germany, succeeded in finding a process of producing potassium-persulphate directly from the sulphates, the neutral as well as the bisulphate by acting with the solid salts on the solution of ammonium persulphate. The ammonium sulphate thus obtained can be regenerated into persulphate by electrolysis. The expensive potassium carbonate and the addition of sulphuric acid for conversion of ammonium carbonate are thereby entirely eliminated. In going one step further the inventors have found that the potassium salts in solid form can be added to the ammonium sulphate in the electrolytic cell direct and the electrolysis carried on with a yield of from 80 per cent at the anode while the reduction does not exceed more than 1 to 2 per cent, no diaphragm being required. The sodium persulphate which on account of its great solubility, hitherto was very difficult to obtain, can be prepared with great ease by the analogous process, a

diaphragm, however, being employed in view of the ready solubility of the sodium-persulphate. (1,059,809, of April 22, 1913.)

Electrodeposition of Nickel, Iron, Cobalt.—Mr. CHARLES J. REED, of San Francisco, Cal., claims an improvement in the electrodeposition of highly magnetic metals, like nickel, iron and cobalt from sulphate solutions with an insoluble anode. He refers to Carl Hering who overcame the difficulty in the case of zinc. Iron has the same property as zinc that it is not easily or efficiently deposited in the presence of free sulphuric acid. It is further unreduced and unaffected in solution as sulphate by contact with metallic lead. The invention consists in electrolyzing the iron sulphate solution with a cathode of any suitable conducting substance, such as carbon, copper, aluminium, etc. As anode is chosen spongy or finally divided lead such as is usually employed in the negative plates of a storage battery. When electrolyzing, the chemical reactions taking place may be represented by the equation: $Pb + FeSO_4 = Fe + PbSO_4$. After precipitating the iron, the anode is removed, placed in a dilute solution of sulphuric acid, and reduced by the electric current to metallic spongy lead. It is then ready for another use. (1,055,652, March 11, 1913.)

The same process applied to cobalt instead of iron is covered by Mr. Reed's patent 1,055,653, March 11, 1913.

Process of Manufacturing Plated Goods.—Mr. DAVID G. REA, of Providence, R. I., makes a two-fold application of electrolysis in the process of protecting the surface of precious metal, during the act of forming an article from this metal. The various parts necessary for the construction of the article, are first shaped in the rough, then electroplated with a base metal, preferably nickel. The parts are then trimmed and assembled, soldered, or otherwise fitted together. Hereafter the base metal is removed or stripped from the surface of the precious metal, by subjecting the completed article to an electrobath. The surface of the precious metal is thus maintained in perfect condition. (1,060,361, April 29, 1913.)

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver.

Mexican Cyanide Practice.—The new 250-ton mill of the Cia. Beneficiadora de Pozos, at Pozos, Guanajuato, Mexico, is described by H. F. CARTER in the *Informes y Memorias* of the Mexican Institute of Mining & Metallurgy, Vol. 3, No. 2. Certain machinery existing at another mill had to be used in the Pozos mill, which otherwise would not have been selected. Symons gyratory crusher and Symons disc crushers are in use, and have given excellent results. The latter are used to reduce the ore to ½-in. size, and have shown a capacity of 250 tons each in 12 hours, with an expenditure of 28 hp. The crushed ore is sampled in a series of two Snyder disc samplers, delivering an ultimate sample of 1 per cent of the mill feed. Chilean mills are used following the Symons disc machines. They have runners of 7 ft. 2 in. diameter and 19 in. face. Each runner is fitted with a tappet arm terminating in a 5-in. roller which strikes an adjustable inclined frame on the operating lever of the feeder. A further adjustment of the feed is obtained by means of a turnbuckle on the feeder mechanism, and the whole arrangement works exceedingly well. Rakes carried by the cross heads direct the ore under the runners and also swirl it against the screens.

Dorr classifiers are used to separate sand for tube milling. Two of the tube mills have El Oro liners, and two have Komata liners which have been used so successfully in New Zealand. The latter consist of longitudinal angle-bars or ribs about 18 in. apart with plates between. The bars lift the pebbles to the proper height without any slip. It is claimed for the Komata liner that it gives increased capacity; perfect cascading of pebbles on the lining; small consumption of pebbles, power and metal; and that it can cast locally of semi-steel, white or grey iron, and is easily installed.

Close concentration is not desired, as it is necessary only to

remove ingredients which are noxious to cyanidation. Deister sand tables are used, with the shortest possible stroke, at 275 r.p.m. The slime pulp from the tables is thickened in Dorr thickeners to 1.5 to 1, elevated by Premier pumps working in tandem, each with a 15-ft. lift, to Pachuca tanks. There are six of the latter, arranged in a circle and adapted for continuous agitation. Each has an overflow discharge pipe, so that any one tank may be the receiver, and any one can be cut out for repair. The $\frac{1}{2}$ -in. connecting pipes between the tanks are inclined at 60 deg., and the upper end is provided with a valve or cover, readily removable. When it is desired to cut out a tank, the pipe cover is placed on the pipe of the tank preceding it, as is also done in the case of the last tank in the cycle. The overflow pipe of this tank is left open while those of all the others are plugged. The overflow is received in a box, and thence flows to an intermediate tank outside the filter house. The Pachuca tanks are connected at the top by launders, and three of these have shoots inclined downwards to carry off scum which is sometimes troublesome.

The Moore system of slime filtration is used, and the gold solution is precipitated on zinc shavings. Melting of precipitate is done in a Steele-Harvey tilting crucible furnace. Electric power is used throughout, and the mill is effectively lighted by 100 32-cp tungsten lamps.

Errors in Sampling and Assaying Ores Containing Coarse Gold.—A discrepancy between the quantity of gold recovered and that which apparently should have been found in the cyanidation of a low-grade ore, caused Mr. FRANKLIN WHITE to investigate all possible sources of error in estimating the value of the ore treated and the residues discharged. Some of the results of his investigations, particularly on assaying, are contributed in *Bulletin No. 103, Inst. Min. & Met.*

It had been the custom to make three assays of 3 A.T. each, and weigh the combined buttons. On investigation it was found that the individual buttons did not check, and that the combined weight did not give an average result that could be relied upon. Variations of from 10 per cent to 30 per cent were noted in a long series of assays:

Examination of the ore showed that the gold existed chiefly in a finely divided condition, but that occasionally it occurred as flake or leaf gold, and on rare occasions, as very coarse gold in quartz. In crushing, the fine portion of the ore was always richer. Sampling and assaying, therefore, were difficult, and the errors finally were ascribed to the accidental presence of particles of coarse gold in the samples. Trials were then made to ascertain the best way of obtaining samples. The old coning and quartering method of sampling, as well as others, was subjected to critical tests, but without satisfactory results.

In order to show the effect of uneven distribution of large particles of gold in the ore, some ingenious tests were made, using ore containing small colored beads and lead shot, to which arbitrary values were assigned. Samples of this kind were quartered, and the number of beads and shot in each sample counted. The greatest variation imaginable occurred, and there was no uniformity in the number of beads appearing in the different samples at successive steps in the process.

The investigations were continued on typical ore samples, which were quartered carefully, and the final sixteen products of the fourth subdivision were assayed. From the results obtained, it appeared to the author that when coarse gold is unevenly distributed in an ore, the odds are against obtaining a correct sample by any means. Neither is it probable that the average of the assays of the whole final quartering will be correct. The probabilities are in favor of the average being too high rather than too low, as the smaller and more evenly distributed particles of gold will be in much the same numbers in every case, whereas the larger pieces, less in number, will be distributed more irregularly.

The author is further of the opinion that fine grinding of the sample is not an adequate remedy, for this would simply result in evenly distributing the few coarse particles of gold throughout the mass, and while the results would be more even, they would still be too high.

The conclusion arrived at is that assays should be made in triplicate, and that repeats should be made at the least in notable variations. The abnormally high results should be rejected and not included in the average result.

The San Francisco Mill, Pachuca, Mexico.—This mill was the first to use the cyanide process in Pachuca, and was also the pioneer in the use of the Brown agitating tank, otherwise known as the Pachuca tank. The present practice is described by J. P. HOLCOMBE, in *Bulletin No. 103, Inst. Min. & Met.* The ore contains principally sulphide of silver, with some manganese, galena, pyrite and blende. The latter three minerals are removed by coarse concentration. The average value of the mill feed is about 20 oz. silver and 0.10 oz. gold per ton. The crushing machines are Blake crushers, Cornish rolls and stamps. The ore is stamped in cyanide solution through screens having six or eight holes per linear inch. The first operation is concentration on Wildfley, whereby about 14 per cent of the valuable mineral is removed. The tailings are classified and reground in tube mills.

The stamps have a much greater capacity than the tubes, hence the small duty of the former, 5.57 metric tons in 24 hours. The tube mill feed is kept as thick as possible and contains about 35 per cent moisture. All the tubes have El Oro linings. Mine rock is used instead of imported flint pebbles, and has been found quite as effective. The consumption is 67.5 lb. per ton of ore. This rock contains approximately 6.43 oz. silver per ton.

The classifier overflow, containing 8 parts solution to 1 of ore, is thickened to a consistency of 1.2 to 1, and agitated in Pachuca tanks where sodium cyanide is added to bring up the strength of solution to 0.4 per cent KCN. Cyanide consumption is 1.83 lb. KCN per ton of ore. Protective alkalinity is kept at 2 lb. CaO by adding lime at the tube mill. Lime consumption is 9.7 lb. per metric ton. Agitation is continued for 36 hours, following which there is a rest of 10 to 12 hours and then further agitation before filtering.

A Moore filter of forty leaves treats 20 tons of ore per 24 hours, with the following cycle: Loading, 20 minutes; barren solution wash, 40 minutes; water wash, 15 minutes. The cake is $1\frac{1}{4}$ in. thick. Precipitation is done in boxes, and the precipitate is washed through a 40-mesh screen and pumped into a Shriver press. No acid treatment is given. Oil furnaces are used for melting, consuming about 0.328 gal. of oil per kilo of bullion. The flux is 7 per cent borax glass, 5 per cent soda ash, 1 per cent lime and 2 per cent sand. The bullion assays from 910 to 940 fine in silver, with four to five parts gold.

The average cost per metric ton milled is as follows:

Rock breakers	\$0.11
Batteries	0.24
Concentration	0.04
Tube mills and classifiers.....	0.20
Agitation	1.05
Filtration	0.10
Pumping	0.00
Precipitation and melting	0.41
Sampling and assaying.....	0.13
Surface expenses	0.05
General repairs	0.17
Power	0.81
General expenses	0.31

\$3.86

The extraction during the period to which the above costs apply was 92.4 per cent of the silver and 94.5 per cent of the gold.

Slime Settlement

A series of interesting experiments on slime settlement is recorded by Mr. W. SHELLSHEAR, in the *Proceedings of the Australasian Inst. Min. Eng., Dec., 1912*. The author used various minerals which were selected for their purity. The list includes zinc blende, galena, rhodonite, garnet, feldspar, quartz, calcite, iron pyrite, copper pyrite, hornite, magnetite

wolframite, scheelite, tin oxide, fluorspar, mica schist, steatite, kaolin, stibnite and molybdenite.

Each mineral was crushed extremely fine in an agate mortar. Distilled water was used to make the suspensions, and some reagents were added to determine their effect on the rate of settlement. An examination of the settling rates of the different minerals showed that they could be divided into three classes, according to the time required for the settlement. Those requiring from 7 to 8 hours to one week were classified as slow settling; those settling in 4 to 7 hours, intermediate settling; and those requiring only 1½ to 3 hours, fast settling. In all cases 2 g. of mineral were suspended in 100 cc. distilled water, and allowed to settle until the water was clear.

Class A, Slow	Class B, Intermediate	Class C, Fast
Tin oxide 4-7 da.	Copper pyrite 4-5 hr.	Zinc blende 3-4 hr.
Wolframite 4-7 da.	Bornite 4-5 hr.	Calcite 2-3 hr.
Felspar 3-4 da.	Molybdenite 4-5 hr.	Garnet 2-3 hr.
Steatite 20-30 hr.	Quartz 7-9 hr.	Iron pyrite 3-4 hr.
Kaolin		Magnetite 2-3 hr.
Mica Schist 30-50 hr.		Stibnite 3½ hr.
Rhodonite 2-4 da.		Galena 3 hr.
Scheelite 3-4 da.		Fluorspar 2-3 hr.

Investigation was made of the effect of different minerals on each other, with the following general deductions: Those of Class A have no effect on the settling of each other; those of Class B have no effect on the settling of each other; those of Class C will settle those of Class A if present in sufficient quantity; those of Class C increase the time of settlement of those of Class B; Class B will not settle Class A, but will either be held up or settle, leaving Class A in suspension.

It will be observed in the table that the sulphides are in Classes B and C; also that three of the minerals of the slow-settling Class A are of high specific gravity, tin oxide, wolframite and scheelite. Again, calcite, a mineral of low specific gravity, is one of the most rapid-settling minerals in Class C. Curves are given which show that for mixtures of minerals of the different classes, there is a definite ratio at which the settlement is most rapid; also that the effect of dilution on settlement varies with the different classes. Thus, with Class A, the greater the dilution, the greater the settlement, as shown by tin oxide. With Classes B and C, and with mixtures of C and A (provided the Class A mineral is not present in sufficient excess to change the mixture into a slow-settling one), the dilution does not affect the rate of settlement.

In testing the action of electrolytes, the author found that 2 cc. of N/10 sulphuric acid per 100 cc. water hastened the settlement in all classes. Calcium chloride has a similar effect, but is specially rapid on suspensions of silicious matter. Lime has an action similar to that of calcium chloride. Caustic soda in small quantities (1cc. N/10 solution per 100 cc. water) causes silicious slime to remain in suspension; in large quantities, however, it acts similarly to the electrolytes. Alum and sodium carbonate settle all classes of slime by forming precipitates. Potassium permanganate settles slime with great rapidity; while tannic acid kept all suspensions in a dense mass, depending on the strength of the solution. After continued agitation, however, the suspension suddenly settles rapidly.

Test for Gold in Precipitated Solutions.—The following test was devised by Mr. Dowsett, reduction officer at the Brakpan mill, South Africa, for use in connection with the Merrill system of zinc-dust precipitation. To 1000 c.c. of cyanide solution add three drops saturated solution of lead acetate and shake. Add 0.75 g. zinc-dust and shake again. Add 20 c.c. saturated cyanide solution and shake for 30 seconds. Pour into an evaporating dish and allow to settle. Pour off solution and dissolve zinc-dust with 10 c.c. aqua regia, boiling until the solution is reduced to about 4 c.c. Decant the solution from the chloride of lead and pour a few drops into a test tube. Cool and add two drops of tin chloride solution. If gold is present to the extent of 0.02 dwt. per ton of the original cyanide solution a very slight coloration will be observed; 0.03 dwt. will show a slight yellow; 0.04 slight pinkish-yellow; 0.06 strong pink, and 0.08 purple of Cassius. The test can be made in a few minutes, and indicates the efficiency of precipitation.

Refractories Manufacturers' Association

Closely following steps taken a number of years ago by industrial and manufacturing interests of all types in this country, about forty of the leading manufacturers of refractories in the central, western, and eastern states have perfected an organization known as the Refractories Manufacturers' Association. The last steps toward organizing the new association were taken at a meeting held April 22, at the Fort Pitt Hotel, Pittsburgh, which was attended by about two score of the most prominent officials connected with the refractories trade.

Manufacturers of refractory products have long since felt the need of an organization that would bring them into closer personal and social relation with their competitors. Although such an organization had been discussed for a number of years, the project remained in an embryonic state until the early part of the present year, when renewed interest was taken in the movement.

The new organization is similar in many ways to those maintained by other business interests throughout the United States. At the April 22 meeting, a constitution was drawn up and accepted, and officers were elected. Any individual or company, engaged in the manufacture of refractories, is eligible to membership and it is probable that the membership roll eventually will contain the names of every prominent official identified with the trade.

The purpose of the organization is "to promote closer relations between manufacturers, dealers and consumers of all kinds of refractories; to improve the products of its members; to standardize, as far as possible, the various designs and shapes of products manufactured; to endeavor to secure the most equitable freight rates for manufacturers and consumers; and to promote social intercourse among its members." It is the intention of the organization to secure some of the most prominent ceramic experts, engineers, and manufacturers in the country to make addresses at the regular meetings.

Mr. H. D. Savage, of the Ashland Fire Brick Company, Ashland, Ky., was elected president and Mr. John H. Cavender, of the American Refractories Company, Chicago, was elected secretary and treasurer of the Refractories Manufacturers' Association, and it is the ultimate intention of the Association to have their own testing laboratory, etc.

The Executive Committee, composed of nine members, is as follows:

Missouri.—J. L. Green, Laclede-Christy Clay Products Company, St. Louis, Mo.

Illinois.—C. S. Reed, Chicago Retort & Fire Brick Company, Chicago, Ill.

Kentucky.—E. S. Hitchens, General Refractories Company, Olive Hill, Ky.

Southern Ohio.—D. D. Davis, Davis Fire Brick Company, Oak Hill, Ohio.

Northern Ohio.—T. E. Thomas, Niles Fire Brick Company, Niles, Ohio.

Western Pennsylvania and West Virginia.—J. H. McFeely, McFeely Brick Company, Pittsburgh, Pa.

Central Pennsylvania.—Geo. H. Diack, Queens Run Fire Brick Company, Lock Haven, Pa.

Maryland.—C. H. Claiborne, Union Mining Company, Mt. Savage, Md.

New Jersey & New England.—Cyrus Borgner, Cyrus Borgner Company.

The members of the association will meet in Pittsburgh, May 20, at their next meeting and an interesting program is being arranged.

It is the general feeling of all parties concerned that the association cannot help but be of great benefit, both to the manufacturer and consumer, not only in such matters as the improvement of quality and character of products of its members, but also in working out and putting into effect efficient methods of manufacture, reduction of costs, standardization and all other problems of common interest to this important industry.

The Production of Sound Steel Ingots

Some seven years ago experiments were commenced by the Simonds Manufacturing Company, of Fitchburg, Mass., at their Chicago plant, aiming to develop a method for the production of sound ingots especially for the manufacture of saws, and the method and apparatus briefly described is the outcome of this long series of experiments. When it was first suggested that the ingots commonly used in crucible steel mills be made sound by fluid compression, it was felt that it would not be practical to develop a method and apparatus that would handle the very small ingots which it would be necessary to operate on, and it was found by investigation that practically no plants were in operation working on ingots of much less than a ton weight.

Having this in mind, the first experiments along the lines of method and pressing apparatus were on small experimental ingots weighing but 15 lb. each, and it was found that there was no difficulty whatever in producing these absolutely sound and without blowholes or pipes. The next step was on slightly heavier ingots, weighing around 110 lb., and of rectangular cross section $3\frac{1}{2}$ in. x $5\frac{1}{2}$ in. A pressing unit was then built capable of handling one 400-lb. ingot or two 200-lb. ingots at each heat and, with modifications from time to time, this arrangement has been found very satisfactory and at the present time at the steel plant of the Simonds Manufacturing Company, Lockport, N. Y., eight presses are in continuous operation, making ingots ranging from 180 lb. up to and including 600 lb. in weight.

It was found early in the series of experiments above referred to that it was quite out of the question to compress ingots in the same molds in which they had been poured, chiefly for the reason that the mold cost and up-keep was too great to make the proposition a commercial one. It was found that an ingot of approximately square cross section could be operated on more satisfactorily than any other cross section (excepting round or octagon), although ingots have been made

only one ingot in the series that could possibly receive proper treatment. Aside from this, most of the apparatus of this nature was rather cumbersome and expensive to operate and the press had to be practically dismantled to get the ingots out after they had been operated on. It was, therefore, considered desirable to set to work on the lines of producing an apparatus possessing the following general features:

First, have the units small and extremely simple so that if

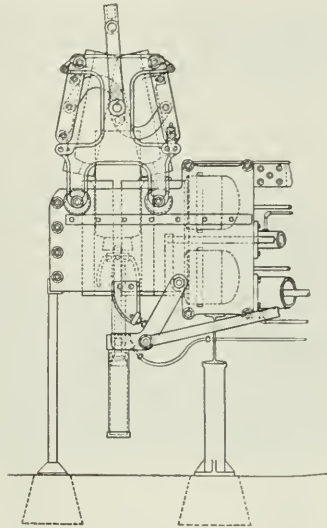


FIG. 2.—POSITION WITH INGOT PARTLY DESCENDED INTO THE DIES

one unit was out of order for any reason it would not in any way affect the rest of the plant.

Second, design these units so that the plant could be added to and the system extended at will and, at the same time, have each unit entirely separate and independent of the others so that it could readily be lifted out of its place with a crane if out of order or damaged and a spare unit put in its place. So far it has never been found necessary to do this, but it was considered desirable to have this in mind in developing this method.

Third, be able to operate on the ingots in such a way that every individual ingot receives precisely the amount of pressure and time necessary to get the best results without reference to any of the other ingots being cast at the same time.

Fourth, develop a mechanism that would be so extremely simple and rugged that it would not require a different type of labor to operate than is commonly found in melting shops.

In a general way, the method consists of casting the ingots in molds made of special cast iron and, while the ingots are still internally fluid but sufficiently set so that they will not burst by careful handling, they are transferred mechanically to steel compression dies where they receive lateral or "side" compression of from $1\frac{1}{2}$ to 3 tons per square inch of greatest area of cross section operated on, this pressure depending largely on the composition of the steel, temperature of the steel when poured, etc.

Fig. 1 shows the general arrangement of presses for handling ingots up to $1\frac{1}{2}$ or 2 tons and as small as may be desired, and as many of these presses as are necessary for a given output are arranged side by side, supported in any suitable way, dependent on local conditions. An extension of the back end of the press serves as a support for a working platform which is made continuous from press to press, this working platform on the smaller size presses being approximately 3 ft. wide and the length of course varying with the number of presses making up the installation.

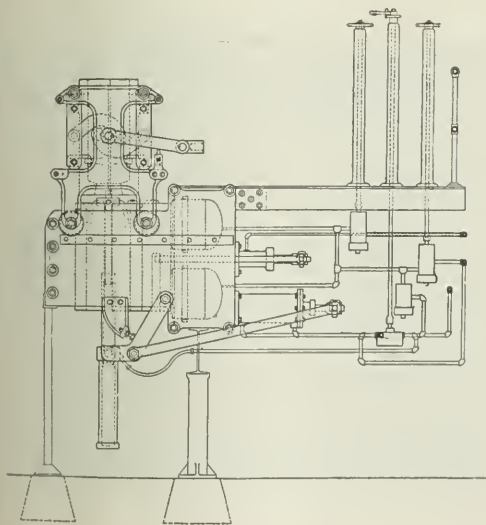


FIG. 1.—GENERAL ARRANGEMENT OF PRESSES FOR HANDLING INGOTS

satisfactorily by the process with a cross section 4 in. x 8 in. x 12 in., and weighing respectively 350 and 600 lb.

Another feature which our first investigations revealed was the fact that practically all attempts to compress ingots by lateral pressure previous to the method outlined here had been along the lines of placing a large number of molds "tandem" style in one pressing unit, and investigation as to the effect of this arrangement showed conclusively that there is practically

This is shown quite plainly in Fig. 3, which also gives a good idea of the valve control stands and the ingot racks. It will be noticed in this view that press No. 1 in the foreground is equipped with two molds whereas the rest of the presses are each equipped with one large mold. A full equipment of both molds and dies are provided, however, for working all of the presses simultaneously on either size of ingot. Other odd sizes are provided for one or two presses where the small tonnage

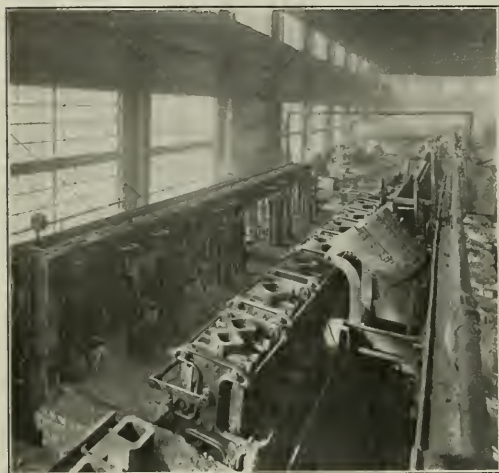


FIG. 3.—WORKING PLATFORM WITH PRESSES

does not warrant equipping eight presses with the special molds and dies.

In ingots up to the sizes mentioned above and also up to and including 3000 lb., split molds are found satisfactory and desirable, as they greatly facilitate the stripping operation and on account of the very short time which the ingots are in the molds there is an entire absence of warping and distorting of the molds commonly met with in split molds where the ingot is allowed to entirely cool in the molds to a point where it usually is stripped. Molds are in operation in Lockport at the present time which have been running eight months night and day and are just beginning to show slight cracks in the side walls of the molds and which are good for another six weeks or two months.

It will be noted by referring to Fig. 1 that the press proper is of a box-like form or design, this being found desirable on account of the great strength for the total weight of the completed press, and for the low cost of machining, aside from making it very rugged and not easily damaged by rough handling. The hydraulic compressing cylinders form one end of the "box" and a heavy steel casting forms the other end, these two members being tied together by cast steel side stress members, interlocking joints being provided, so that the greater the strain between the back head and the rams the more closely do the side members pull in on the cylinders and back head. The tie rods holding all of the press members together have, therefore, only to keep the members in place when there is no load on the rams or when handling the units.

A simple retracting arrangement for the cross head and hydraulic rams is provided which, in the case of the smaller size units, is usually a spiral spring and which, in the larger sizes is a hydraulic cylinder of very small cross section relative to the cross section of the pressing cylinders and which is piped to the high pressure supply without any provision being made for valves or control.

The mechanism for lowering the partly cooled ingots from the molds into the compressing dies consists of a hydraulic cylinder and ram carried on two swinging arms (one on each side

of the press), so that it may be brought out of vertical alignment with the dies when desired and leave the pit under the dies absolutely clear and unobstructed. The mechanism for accomplishing this result consists of a small low pressure cylinder usually mounted on the back end of the press and provided with a simple cross arm and connecting rod arrangement controlling the movement of the swinging arms carrying the lowering ram cylinder noted above. The controlling valves and mechanism are extremely simple, all of the movements of the ingot after it leaves the mold being accomplished by one low pressure controlling valve, the handle of which being moved from notch to notch causes each of the parts to function properly and in their proper order. The high-pressure cylinders are controlled by a simple form of stop valve and a simple one-way valve for discharge.

Mounted above the compressing dies is a mold-carrying rack which needs very little explanation, as the drawing Fig. 1 shows quite clearly the principle on which this works. The molds shown here are split molds and in the smaller sizes this mold-carrying rack is operated by hand and in the larger sizes by a small hydraulic cylinder mounted on the side of the press frame and which is not shown in this drawing. Ingot of various cross sections have been operated on by the regular equipment, but the present molds and dies are designed and built for pressing the ingots corner-wise; that is, with one of their cross-section diagonals at right angles and the other normal to the direction of compressing, and ingots of square cross section have been found to work out better not only from the compression standpoint but also for subsequent operations in the mills.

The bottom of the split mold referred to is formed partly by a removable round taper block carried on the lowering ram and so arranged that when the molds are "set up" this lowering ram is in its uppermost position and the taper block forms a "bottom closure" to prevent steel from running through the bottom of the mold. It is found that these blocks or mold bottoms take practically all the wear of the mold, such as the cutting effect of the stream of molten steel when pouring, and



FIG. 4.—VIEW OF POURING

they are practically the only parts that have to be renewed under seven or eight months, and these ordinarily last when running night and day on one 30-pot furnace from two to three months. The regular procedure or operation on ingots with this apparatus is as follows:

The molds (having been previously well smoked without removing from the racks) are locked together by the toggle arrangement shown in the drawings and the lowering plunger

brought to its uppermost position so that the bottom of the mold is closed by the taper block referred to. The controlling valve on the high pressure cylinder is, of course, closed and the discharge valve opened and the dies are thus opened to a predetermined point.

The molds are now poured by any approved pouring device, but at the plant described by a top-poured hydraulic operated ladle, shown in the half-tone, Fig. 4, and after pouring the ingots are allowed to set a pre-determined length of time, depending on the size of the ingot, composition of steel, etc., but in any event just as short a time as is necessary to form a light skin which will not burst open when the ingots are stripped. This time varies from one to five minutes on ingots from 150 to 600 lb. in weight and of various composition. The stripping mechanism is now operated so that the molds occupy the position shown in Fig. 2 (which shows the ingot partly descended into the dies), leaving the ingot supported at the top end by the molds (which are not opened much, if any, at their top ends) and by the tapered block at the bottom end which forms the bottom closure of the mold during the pouring.

The low-pressure controlling valve handle is now moved one notch, causing the ingot to descend into the compressing dies, the speed of this descent being controlled by a stop cock in the discharge line. The lowering cylinder is provided at its lower end with a spring or in some cases with a hydraulic stop so that the ingots are brought to rest very gently and in a position suitable for the pressing dies to operate on them. The high-pressure release valve is now closed and the low-pressure controlling valve handle is moved to the next notch and low-pressure water is thus caused to flow from the low-pressure supply through a high-pressure check valve into the high-pressure compressing cylinders and the compressing dies are thus rapidly moved up to and in contact with the ingots and the cylinders filled with water under low pressure, usually 100 to 125 lb. The high-pressure controlling valve is then opened very slightly and compression is commenced. At this point, the ingots are still fluid internally and have not begun to show the slightest sign of pipe or "sinking" in the top end.

The compressing cylinders are allowed to come up very slowly, indeed, so as to just keep the ingots from forming pipe and this is continued until by experience it is known that the ingot is very near the point of setting or becoming solid throughout, when the pressure is increased slightly and a small amount of the last portion of the ingot to cool is squeezed out of the top, this, of course, being the liquated portion mentioned by A. Capron in his very able paper before the British Iron & Steel Institute of 1906 and, as pointed out by Mr. Capron and others, this liquated portion contains practically all the segregation and is decidedly higher in phosphorus, sulphur and other impurities. This liquated portion forms a rough ball-like knob on the top end of the ingot and is practically the only portion of the ingot not suitable for finishing and in weight varies from 2½ to 4 per cent. of the total weight of the ingot.

As soon as the compressing dies have a good grip on the ingots so that there is no possible danger of their dropping through into the pit, the low-pressure controlling valve handle is moved to the fourth notch, causing the low-pressure cylinder to move the ingot-lowering cylinder out of vertical alignment with the compressing dies and the controlling valve is then left in this position until compression is completed.

The pressure is kept on the ingot for a considerable time after they have ceased to "rise"; in fact, after they have reached the point where they are considered solid the pressure is quite materially increased and the ingots then receive what is virtually a hydraulic forging treatment which tends to break up the coarse crystals formed in cooling into a fine crystalline structure much resembling an ingot worked under a hammer or forging press.

When the operation of compressing is completed, the high-pressure control valve is closed and the discharge valve opened, when the dies immediately retract and the ingots drop through

into the pit, and this may be provided either with a bottom of sand so that the ingots may be given a partial annealing before going to the stock piles, or the ingots may be dropped on to cars or a conveyor. The resulting ingots have the appearance shown in Fig. 5, this being from a 400-lb. ingot split longitudinally, while Fig. 6 is an ingot poured at the same time weighing slightly less on account of its decreased length and not compressed.

There is no dismantling of the presses to discharge the ingots and the operation of setting up ready for another heat takes but a few minutes. In fact, the first of the presses to be poured are quite often set up, that is the molds locked together and the low-pressure valve handle moved to the first notch of its quadrant, thus causing the lowering ram to rise to its uppermost position and the taper block which it carries to close the bottom of the mold, before the last of the series have dropped their ingots and the total operation of setting up eight presses occupies but 10 or 12 minutes, the greater part of which is used in smoking the molds.

One man easily operates eight presses and the presses are so designed that they may take care of from two to three furnaces (in the plant being described these being 30-pot



FIGS. 5 AND 6.—INGOTS WITH AND WITHOUT COMPRESSION

crucible furnaces), providing all the furnaces are running on the same size of ingots so that it is not necessary to change the molds and dies between the heats, as the presses are amply provided with cooling facilities so that the parts do not become overheated.

It is quite obvious that with the hot ingots in contact with the mold walls only about 1/20th of the time usual where they are allowed to cool in the molds, the latter do not get overheated and no artificial cooling of any kind has been found necessary on the molds. The cross head and, on large presses, the compression dies are provided with water cooling, and the high-pressure cylinders are also piped so that when they are not under actual load low-pressure cooling water may be circulated through them, thus keeping the rams and packing leathers at a reasonable temperature. There has been no trouble whatever experienced with packing leathers, all of these having on an average run ten months, night and day, without renewal.

The pressing dies, as mentioned early in these notes, are of cast steel and in the equipment described there are two grades:

that is, one-half of the dies are of .40 carbon and the remainder .70 carbon steel castings, but up to the present time there is no appreciable difference in their performance, as all of the dies show practically no signs of wear and should last at least one year without replanning, and as the design has made allowance for at least seven or eight replannings, they should last four or five years without renewal.

One point of special advantage in changing from one size ingot to another is the fact that there are absolutely no fastenings of any kind holding the molds or dies to the other mechanism, excepting that the molds are provided with loose links which drop over one of the cross bars in the mold rack and can be readily lifted out by hand or by using a pair of tongs.

The molds are usually lifted out of the rack with the crane in pairs, special hooks being provided for this, and it is no more trouble to lift them out or in the mold racks than lifting them up from one position on the mill floor and setting them down in another. The compression dies are simply lifted in or out of the press frames and gravity alone keeps them in place. The retraction of the movable dies is accomplished by a loose fitting link, one end of which engages a pin in the top end of the movable die, the other end a pin in the top end of the cross head, and this is readily lifted off and on with tongs or by hand.

The plant at Lockport is served by a single-acting triplex, high-pressure pump with 1¼-in. rams and 8-in. stroke and driven by a 15-hp. motor, and the low-pressure supply is obtained by a 3½-in. x 4-in. triplex, single acting pump running at 100 lb. pressure and driven by a 3-hp. motor, and as these pumps are only run about one-half hour at each heat, it is obvious that the power consumption is extremely low. Both pumps have a common supply tank and, of course, the discharge from the cylinders is piped back to this tank.

This portion of the plant was so designed that, if found desirable, during the extreme cold weather incurred at Lockport, a solution of alcohol and water could be used to guard against freezing, but so far this has not been found necessary as the radiated heat from the adjacent furnaces has been sufficient to obviate any trouble from freezing.

It may be said in passing that all of the hydraulic valves on the high-pressure side of this system are designed on the lines shown in Fig. 7, so that the valve and valve seats can be removed very readily for repairs without breaking any pipe connections. The time for making a change on any of these valves practically never exceeds 30 seconds, and on a test has been done in 9 seconds, and as the parts are extremely simple to machine, this is one of the least of the troubles in the every day operation of this plant. It has been found desirable to also equip the hydraulic pump which operates at 4500-lb. pressure with valves of this general type, so that there is never any shut-down or stoppage on account of leaky valves.

In a general way, the method is not intended to be a curative one, as tests made in our early experiments proved conclusively that if a pipe in an ingot had once developed and sulphides and oxides formed on the surface of the cavity, it was practically impossible to weld it up or cure it in any way, but the method is preventive and most effectively prevents a pipe forming. Various grades of steel have been operated on that may be made in a crucible or electric furnace, but practically all of the tonnage so far handled is tool steel used in the manufacture of saws and machine knives, this including straight

carbon steels of the usual carbon content for the purpose noted, as well as low alloy steels. It has been found, however, that in the case of strictly high-speed and semi-high-speed steels the apparatus must be handled very carefully and the ingots at once annealed after pressing, this owing to the increased density of the already very dense material, owing to compression. Great care must also be taken in re-heating ingots of this type that the heat is brought up slowly and uniformly to prevent the ingots bursting.

The plant as described above has been working night and day for something over a year and at the present time all of the band saw and circular saw steel, excepting the very small circulars, are made from compressed material and equipment is now in process of construction which will enable the Simonds Mfg. Co. to compress every pound of steel made in Lockport.

It has been found that the total cost of operating this plant is less than 3 per cent. of the value of ingots operated on and this will be reduced somewhat when the new equipment is completed, as a greater tonnage of ingots will be handled with no increase in labor cost and a decrease in capital charge, for the reason that interest and depreciation is now taken at 15 per cent. on not only the total cost of the plant as a going proposition, but also on all of the experimental work and expense up to date.

Experiments are also under way for a modification of the device to be used, especially on large ingots in which the split molds are replaced by the usual form of solid tapered mold handled by an ingot stripper of the usual design. The process and apparatus is covered by patents Nos. 922,587 and 926,489 of 1909 and 1,056,101 of 1913, as well as patents in foreign countries, all granted to Mr. L. E. Howard and assigned to the Simonds Mfg. Co.

Iron and Steel Committee of the American Institute of Mining Engineers

The following Iron and Steel Committee has been appointed by the American Institute of Mining Engineers to serve until the next annual meeting: Charles Kirchhoff, chairman; Albert Sauveur, vice-chairman; Herbert M. Boylston, secretary, Abbot Building, Harvard Square, Cambridge, Mass.; John Birkbine, William H. Blauvelt, James Gayley, Henry D. Hibbard, Henry M. Howe, Robert W. Hunt, J. Esrey Johnson, Jr., William Kelly, Richard Noldenke, Joseph W. Richards, A. A. Stevenson, Felix A. Vogel, Leonard Waldo, William R. Walker, William R. Webster, Frederick W. Wood.

The various interests of the iron and steel industry are authoritatively represented on this committee by producers of raw materials (iron ore, coke and other fuels, refractories, etc.) and of finished materials, by consumers, consulting metallurgists and educators.

The primary duties of the committee are to secure important papers on iron and steel, to promote their discussion, to organize lively and fruitful meetings and otherwise to foster the interests of the large number of members of the Institute connected with the iron and steel industry in its many phases. Its success during the past year has been noteworthy and was highly conspicuous at the meetings in Cleveland and New York.

Since the next meeting of the Institute is to be held in the mining district of Rutte, far remote from iron and steel centers, plans are now being made for a meeting in a more accessible place in the latter part of October under the auspices of the Iron and Steel Committee to be devoted to the presentation and discussion of papers relating to iron and steel.

Bureau of Standards' Analyzed Samples of Acid Open-Hearth Steel

The Bureau of Standards, Washington, D. C., has now ready for distribution samples of acid open-hearth steel with 0.8 per cent and 1 per cent carbon, the fee being \$2 per bottle holding about 150 grams.

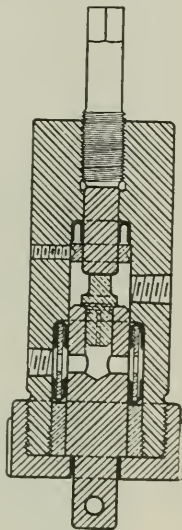


FIG. 7. — HYDRAULIC VALVE

A New Recorder Pen Arm to Facilitate the Changing of Charts

Every user of recorders has found, without doubt, the changing of charts to be a continual source of annoyance as well as time-consuming operation, owing to the obstruction caused by any practically rigid pen arm. The lifting of the pen arm from the chart holder, retaining it in a strained position with one hand, while the chart is changed with the other, without spilling the ink or soiling the chart, requires considerable practice and dexterity.

The Schaeffer & Budenberg Manufacturing Company, of Brooklyn, N. Y., have applied for patents on a new recording pen arm to be used exclusively on their Columbia recording gauges, Columbia recording thermometers and Columbia tachometers.

The three elements of the new Columbia pen arm and their respective functions are clearly apparent from Fig. 1, in which the upper part is the pen arm proper, the lower part is the pedestal, so to speak, and between these two parts there is the adjusting screw.

In Fig. 2 the pedestal is directly connected to the recording element of the instrument, the pen arm being placed over this and retained there by its slight resiliency and with the aid of the pin on the adjusting screw.

When the chart is changed, the pen arm is drawn towards the operator, lifted off the pin, and allowed to drop down to the right while the pedestal remains stationary (see Fig. 2). The obstructing pen arm being thus removed, and both hands being free, the chart can be changed and the pen arm placed in its proper position again in an instant. The adjusting screw represents an important feature. A slight manipulation of the thumb adjusts the pen arm whenever necessary.

The Schaeffer & Budenberg Manufacturing Company have just issued a handsome and comprehensive catalogue on their Columbia recording gauges, in which the Columbia pen arm is illustrated and described.

A New Ammeter and Voltmeter

The Keystone Electrical Instrument Company, of Philadelphia, Pa., has been perfecting for the past year a midget ammeter and voltmeter 3 in. in diameter of the highest grade for use on switchboards, automobile dashboards, and for portable service where a small instrument is particularly desirable.

In bringing out these instruments the Keystone Electrical Instrument Company has endeavored to reduce the number of parts re-

FIG. 1.—RE-
CORDER PEN
ARM

quired in the standard d'Arsonval system to a minimum. It is a well known fact that the d'Arsonval principle is the most accurate for direct-current measurements, and in order to reduce the cost of these small instruments, and at the same time produce a high-grade accurate instrument much study has been given to this feature.

The adjoining illustration shows the board-type Keystone ammeter, with cover removed, showing the simplicity of construction. A tungsten-steel magnet of unusually large size is used so that permanency can be secured, and to increase the dead-beat operation of the instrument.

This magnet has no special pole pieces, the poles being accurately ground. This does away with the magnetic joint between the magnet and the pole pieces, and also increases the magnetic strength. The bases are of moulded material to insulate the studs carrying the current through the switchboard,

or dashboard, and this material is also used so that the magnetic strength may be unaffected. To obtain the greatest strength from the magnets the covers are also formed of brass instead of iron. All holes in the covers and bases are square so there is no possibility of turning of any of the parts when tightened up.

The aluminium coil form, seven thousandths of an inch in

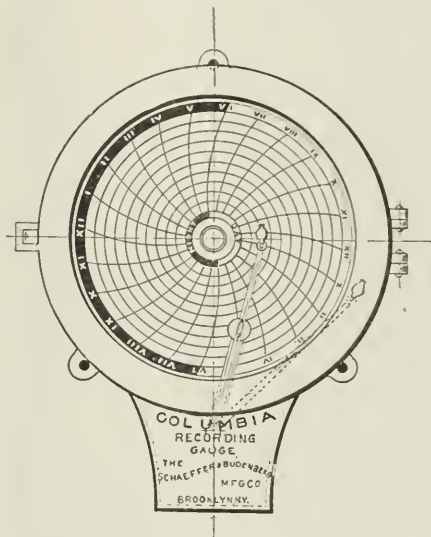
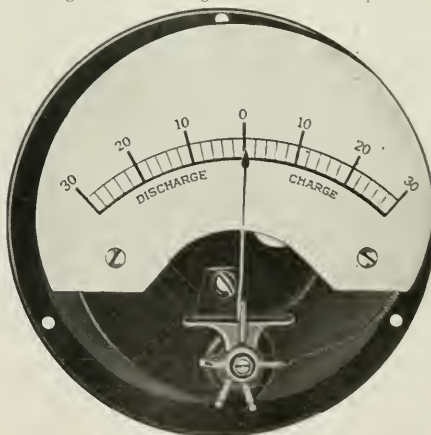


FIG. 2.—PEN ARM ON RECORDER

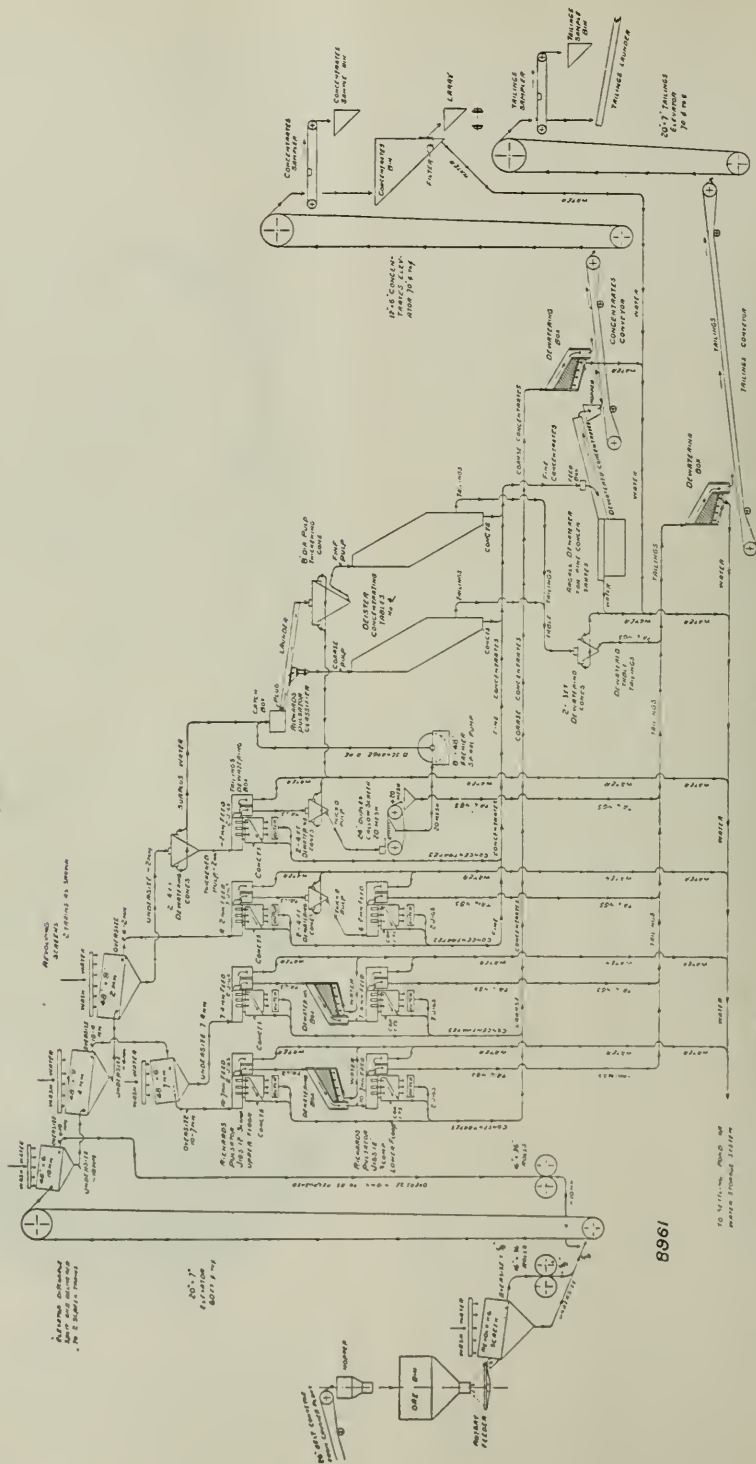
thickness, on which the fine enameled wire is wound, swings in sapphire jewels between the poles of the magnet. The pointer is of aluminium tubing twelve thousandths of an inch in outside diameter and with a wall one thousandth of an inch thick, and the whole moving element weighs only slightly over one-tenth of a gram, or the weight of one common pin. With a



AMMETER WITH COVER REMOVED

moving element so exceedingly light as this the instrument is absolutely dead beat in operation, and the pointer does not oscillate, although the instrument is subject to severe vibration.

This exceedingly light moving element also has the advantage that there is very little weight whatsoever on the pivots and bearings, assuring long life for the instrument. The scale is white enameled on brass except where the black scales with white letters are preferred.



Concentrating Low-Grade Iron Ores in Michigan

In iron mining in the Lake Superior District, as in base and precious metal mining in the West, the problem of utilizing low-grade ore is of growing importance. With the gradual exhaustion of the more desirable ore comes the demand for suitable methods of preparing low-grade ore for reduction. There are but three important iron concentrating mills in the Lake Superior region. Two of these, in Minnesota, were described in this journal in November, 1912. At both of these mills the ore is treated without crushing other than that received when it is blasted and steam-shoveled in the open pits. The system

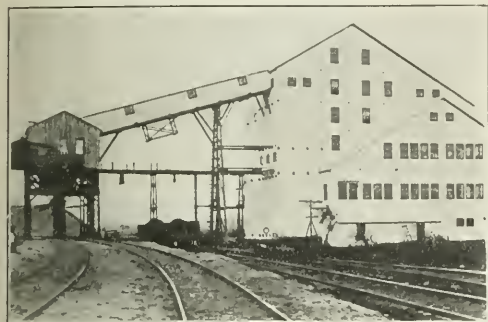


FIG. 1.—IRON-ORE CONCENTRATOR, BOSTON-AMERICAN MINING COMPANY, DIORITE, MICH.

is merely one of washing in trommels and log-washers, with concentration of the finest ore on tables.

Since the erection of these two mills in Minnesota, a third has been built by the Denver Engineering Works Company, Denver, for the American-Boston Mining Company, at Diorite, near Ishpeming, Mich. This mill is designed to treat an ore which requires finer crushing than those of Minnesota, and the feature of the process is the use of jigs, which are not used in the other two mills. Before the American-Boston mill was built, it was necessary to exercise care in the selection of ore shipped from the mine, producing only that which contained a sufficiently high percentage of iron to bear the cost of transportation and reduction. A great deal of ore was found which could not be thus mined and shipped, and it was for the enrichment of this material that the mill was built. The dual result has been a reduction in the cost of mining, and the conversion of a former waste into a valuable product. The ore now being treated is hematite in the form of alternating bands of mineral and gangue.

A general view of the mill is shown in Fig. 1, and a flow-sheet is given in Fig. 2 (p. 356). The ore is crushed in gyratories and elevated by belt conveyor to the mill bins. It then passes through revolving screens, $\frac{3}{8}$ -in. mesh, the oversize from which is crushed in rolls. The screen undersize and the roll

product are combined and elevated to two trains of revolving screens which deliver jig feeds of the following limiting sizes: 10 to 7 mm; 7 to 4 mm; 4 to 2 mm; and minus 2 mm. Each of these grades is treated in a separate group of Richards pulsator jigs, 15 of which are suitably arranged on two floors.

The jigs deliver coarse concentrates from side gates above the screens, and fine concentrates from open spigots in the hutches. The tailings from the jigs on the upper floor are dewatered and retreated in jigs on the lower floor. Simple screening has been found an efficient method of treating the tailings from the 2 mm jig, for when this product is thickened and passed over a 20-mesh Callow screen, the oversize can be discharged as tailing. The undersize is classified in a Richards pulsator classifier of the launder type, the spigot discharge concentrated on a Deister table, and the overflow similarly treated after thickening in a Callow cone.

Coarse concentrates are dewatered in a box, and fine concentrates in an Avoca spiral dewatering machine. Both products are delivered to the same belt conveyor with about 15 per cent moisture, and elevated to a filter-bottom concentrates bin. An automatic bucket sampler is placed between the elevator discharge and the bin.

The arrangement for unloading concentrates from the bins to railroad cars can be seen by reference to the photograph of the mill and the flow-sheet. Concentrates can be discharged by gravity from the bins to an electric larry which runs on a track 30 ft. above the ground, delivering the material either to the railroad loading bins or the stock pile.

A point of interest in this mill is the use of the Richards pulsator jig shown in Fig. 3. Despite the fact that the opinion formerly prevailed that these jigs were not adapted to hutch work, the richest and greatest amount of concentrates produced at the American-Boston mill is from the hutch spigots. The success of the jigs is interesting also because the ore cannot be regarded as specially adapted to jigging on account of the production of thin flat scales of mineral in crushing.

The water supply is obtained from a small lake, being pumped to a stump at the mill, whence it is raised to the mill storage tanks by centrifugal pumps. A large part of the water discharged from the mill with the tailings is recovered in a settling pond, the overflow from which gravitates to the mill sump.

The arrangement of the mill is such that most of the operations are automatic, and only four men are required per shift.

Since the mill was put in operation and adjusted, it has been running steadily. Within two weeks of the start it was producing 60 per cent iron concentrates at the rate of seven standard-gage ore cars per day. The present capacity is 500 tons in 24 hours, but space is still available for more jigs, and the management expects to increase the capacity to 900 tons per day. The local manager for the American-Boston company is Mr. J. R. Thompson, who was one of the first to appreciate the possibility of enriching the low-grade iron ore by concentration.

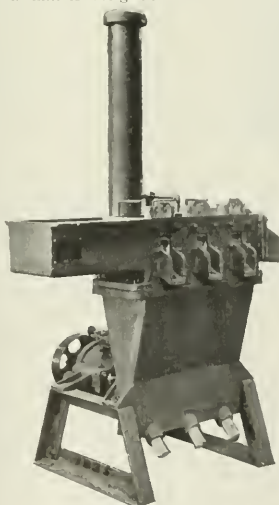


FIG. 3.—PULSATOR JIG

High Vacuum Surface Condensers is the title of Bulletin 106A, recently issued for general distribution by the Wheeler Condenser & Engineering Company, of Carteret, N. J. This bulletin takes up such subjects as the effect of condensing on steam engine and steam turbine economy; the choice of a kind of condenser; the design of high vacuum surface condensers; the influence of steam distribution; the effects of flooding on condenser surface, and explains the advantages of the Wheeler dry tube condenser. The bulletin then discusses the details of construction of Wheeler surface condensers, considering such points as the shape of shell, tube plates, tubes, packing water boxes, etc. Consideration is then given to the subject of air pumps for surface condensers. The Wheeler Condenser & Engineering Company manufactures direct-acting wet air pumps for steam engine service, also the well-known Wheeler-Edwards air pump for high-vacuum work, several types of reciprocating, rotative valve dry vacuum pumps, and also the Wheeler-Turbo air pump for the direct connection to the steam turbines of motors.

The Design, Construction and Uses of Centrifugals

By H. H. Stephens

The Weston principle of supporting centrifugal machines, providing a construction that permits a basket to rotate around its center of gravity rather than its geometrical center, is old, and is admitted by all to be the best present solution we have for handling unequally loaded charges on the centrifugal basket with the minimum amount of wear on the apparatus.

It has been, and is, the ultimate aim of many engineers and operators, to produce a centrifugal machine, for the purging of sugar and like uses, that would be continuous in its operation and thus dispense with the continual starting and stopping; the part of the operation that is most destructive to machine



FIG. 1.—DIRECT ELECTRICALLY DRIVEN CENTRIFUGAL

and which naturally cuts down output of machine and consumes much more power than if machine runs continuous.

However, the present state of the art does not reveal a practical continuously operating centrifugal, although much thought and study have been given to this. Such a machine would be ideal for some purposes but it is doubtful whether or not a continuous centrifugal could be used to universally supplant the present type of machines. For instance, a continuous machine might be devised to operate satisfactorily on first-grade fine sugars, but it is a serious question as to whether or not the same machine would successfully handle low grades and seconds on account of the entirely different characteristic necessary on the acceleration periods.

Therefore, in view of the absence of such an ideal machine, and moreover, as there is a probability of its not being universal in application it seems that the intermittent operation of the present centrifugals is a necessary evil to which the efforts of all engineers should be directed with a view to lessening the same. The mere fact that such starting and stopping is necessary, not only means a decrease in economy and efficiency, but it also means, which is extremely important to the sugar maker, that the quality and consistency of the discharged sugar absolutely depends on the acceleration periods be absolutely the same.

It thus becomes apparent that, in addition to being the most severe part of the cycle of operation, the intermittent starting imposes upon the designer the necessity of consistent, unvarying performance to maintain a standard of output both in quality and quantity; in fact the behavior of a centrifugal during this period reflects strongly on its design and construction.

Since this phase of the service is so exceedingly important, it follows that the successful centrifugal must possess the following underlying principles as upon these depend the operation of the entire machine.

1. The operation of starting and stopping must be through one lever rendering the machine as fool-proof as possible.
2. All parts of machine, especially spindle and basket, must be designed amply heavy on account of the severe strains during this operation.
3. The driving motor must be special and designed to handle the severe overloads during the starting period and must be independent of centrifugal.
4. A simple, reliable and rugged starting apparatus that will insure motor never being overloaded beyond a certain predetermined point, no matter how suddenly it is started.
5. A powerful brake, accessible for relining and designed so that the pressure applied is predetermined and not dependent upon the action of the operator.
6. A suspension with proper design of buffer to restrict oscillation and provide a cushion to absorb vibration in bearings and supporting framework.
7. A starting apparatus that must be designed so that variable acceleration can be obtained readily and quickly; and once set must be beyond the immediate control of the operator.
8. A conspicuous absence of all delicate moving parts in machine, motor, or controller.
9. The number of wearing parts and parts requiring lubrication must be reduced to a minimum, and parts such as bearings and buffers must be so placed to be taken out and replaced quickly.
10. The machine must be designed that repairs can be effected in the shortest possible time and avoid any possible tie-up during the running season.

To sum up the above, it becomes apparent that a centrifugal to give maximum output at minimum cost should possess all the essential features thus enumerated.

There are four standard methods of driving centrifugals in use at the present. Together with their characteristics, they are as follows:

1. Group belted to one driver: Low first cost, low efficiency and output, and high maintenance cost.
2. Individual water driven: High first cost, low efficiency and output.
3. Belter to individual motors: Moderate first cost, fair efficiency, expensive belt maintenance and fair output.
4. Direct electric driven: High first cost, maximum output and efficiency, minimum maintenance.

Until recently, it has been the feeling among most engineers that the belted machines as described under Class 1, represented the successful conservative combination of economy and output, but lately, upon fully studying the numerous advantages of the direct electric-driven centrifugals, this attitude has gradually but surely changed in favor of the individual motor, with the result that many modern plants are now employing or specifying exclusively direct motor-driven centrifugals.

The economic centrifugal is never the lowest in initial investment, but it is the machine giving maximum output with minimum power, having fair efficiency, giving a product of consistent quality and being free from serious breakdowns and with low maintenance costs.

The types and uses of centrifugal machines are numerous and steadily increasing. New applications of centrifugal separation

and clarification are being made daily; in fact, the field for its uses seem limitless. A few of the centrifugal processes now in common use are:

Purging.—Granular and crystallized substances, sugar, salt,



FIG. 2—OIL EXTRACTING CHIP CENTRIFUGAL

soda, sulphate of ammonia, brewers' grains, fertilizers, tankage, hides, pelts, and metal turnings.

Hydro-extracting.—Raw stock (cotton, wool, silk, jute, hair, hemp, linen, ramie); textile work (yarns, piece goods, hosiery, felt and hats, beam and spool fabrics and yarns); laundry service, bath robes and bathing suits.

Clarifying.—For filtering and separating various liquors, juices, oils, etc.

Sewage.—For the separation of sludge from fresh sewage and the reduction of moisture therein.

Nitrating.—For all separating processes in connection with nitrating work.

Dyeing.—For dyeing of raw stock, yarns, cops and various forms of textile fabrics, including sulphurous dyes.

Oil and Waste Reclaiming.—For separating and reclaiming oil from wiping waste, machinery wiping towels, rags, etc., for washing and drying waste, towels and other machinery wiping material.

Oil Extracting and Chip Centrifugals.—For recovering oil from automatic machined parts, metal chips, turnings and scrap.

The D'Olier Centrifugal Pump and Machine Company, of Philadelphia, Pa., are the pioneers on the application of the individual electric motor drive for centrifugal apparatus and have been identified with the same for a period of ten years. The cuts shown indicate a few of their standard machines. They also specialize in the design and construction of centrifugal apparatus involving the use of extreme high speeds and special construction.

Cooling Towers.—Natural-draft towers of the Wheeler-Balcke design are described in Bulletin 109, of the Wheeler Condenser & Engineering Company, of Carteret, N. J. These towers are of wood with an improved water distributing system and compact cooling stacks which cools the water on the "drop and film" principle. The walls of the tower are continued above the level of the distributors, so as to form a chimney of some height, which induces a natural draft and draws the air through the tower without any expense of driving fans. The bulletin discusses the theory of cooling, and points out the necessity of scientific design of condensers and cooling towers. It then discusses the economy of the Wheeler-Balcke construction. This bulletin is profusely illustrated with photographs of many interesting types of natural draft towers. Drawings are also shown of typical installations with different types of condensers and towers.

Absorption and Reaction Towers for Chemical Factories

Absorption Towers

Absorption towers, as generally employed in the chemical industry, may be divided into two groups:

A.—For the simple absorption of a gas by a liquid—usually forming thereby a definite chemical compound.

B.—As mixing and reaction towers for gases with *simultaneous* absorption by a liquid.

Under "A" we have towers for the absorption of sulphur dioxide, carbonic acid gas, nitric acid and nitrous fumes (residual gases from works processes and Gay Lussac towers), carbon bisulphide, acetone and the like.

In such cases the chief requirements are that the gases should pass over surfaces uniformly wet with the liquid—that such surfaces should be as large as possible—and that at the same time an eddying motion should be imparted to the gases, as they pass upwards, so that fresh gas particles constantly come in contact with the wet filling material.

Supposing that for such an absorption several towers were arranged in series, it is evident that in the first tower the absorption will proceed more readily than in subsequent towers. It is, therefore, desirable that the filling material for each tower should be suitably chosen, in order to avoid unnecessary expenditure.

The main requirements for good *absorption* are greatest possible surface per cubic foot of filling, good mixing of gas, and the correct proportioning of the ratio between the section of the tower, and the net section free for the passage of the gases—this in order that the gases may not pass too rapidly up the tower, without, however, appreciably affecting the draught.

The intending purchaser has in these days of strenuous competition a large variety of tower packings to select from. The days of coke as a filling material, except in certain very special cases, are numbered, and it has long been established that a small tower with good packing will do the same work as a large coke tower.

Plain acid-proof bricks, prism-shaped bricks, triangular, rhombohedral and trough-shaped bricks, and rings of various kinds, have all been tried, and all suffer from the same defect, namely, that they occupy a large percentage of the volume of the tower, and leave insufficient "absorption space" or "free space" for the gases. Some of these bodies seem to be designed on the principle to take an empty tower, fill it up as much as possible with packing (presumably with the intention of obtaining a maximum of wetted surface) and leave as little room for the gases as possible.

Broadly speaking, it may be said that the better and more efficient the tower filling the greater may be the velocity of the gases. The sectional area of a tower naturally determines the velocity of the gases in the tower, and hence the so-called "free space" or "absorption space" of a filling material has an important bearing on its efficiency. The "free space" is the percentage of the cubic capacity of a tower which remains free for gases.

The great success of the Guttman hollow balls as a tower packing was due to their very high "free space." Millions of these balls were sold, but they could not come into universal use because of their high cost and because they were unsuitable for large towers and dusty gases.

The latest type of filling material, the Guttman cells, are superior even to hollow balls, and not only in regard to efficiency, but also because the cells are considerably cheaper and have a very high "free space."

As will be seen from the illustrations these cells when built up in a tower form in section a regular honeycomb structure, each four cells enclosing a reaction space corresponding to an additional cell. The whole of the tower is thus symmetrically divided up into a number of equal reaction spaces, and in such a way that both the outer and inner surfaces of the cells are wetted, and must be traversed by the gases, which are ascending and diffusing through the tower filling. "Channeling" or

the formation of separate streams of gases or liquids cannot occur when cells are used, as is often experienced with other forms of filling material.

Fig. 1 shows diagrammatically four cells arranged in the same way as if they were stacked in a tower, the path of the liquor being shown by plain arrows and the probable course of the gases by means of circled arrows.

Slots are made in the top and bottom walls of the cells; the gases entering from below expand in the interior of the cell and pass out through the slots in the top. Each cell is provided at the sides with a number of projections, which rest against corresponding projections on the neighboring cell, and at the same time form the slots for the reaction space formed by four adjacent cells.

The gases are efficiently split up and distributed by the slots, and always pass through a narrow opening into a relatively large reaction space; the gases expand into and diffuse throughout the cell before again issuing through a narrow slot; the liquid flowing in the opposite direction, and forming a complete film over the cell surfaces, thus comes into most intimate contact with the gases, and very efficient absorption results. The eddies caused by the alternate expansion and contraction of the gases serve to thoroughly mix the gases and to induce further reaction or absorption.

The side walls of the cells being very steep render it practically impossible for solid matter or impurities to collect on them, while, if necessary, the cells can be readily and quickly flushed down and cleaned. For extremely *dusty gases*—such, for instance, as pass through a Glover tower—the lower third of the tower may be filled with large slabs or bricks in the usual way.

The area of the slots, and thus the area available for the passage of the gases, is so designed that the gases remain for a sufficient length of time in the tower for the reaction to take place, without, however, appreciably affecting the draught.

In order to obtain, as far as possible, a perfectly uniform distribution of liquid, distributing channels are placed on the top layer of cells in the tower, and it will be clear that, given a uniform feeding arrangement for the liquor, the inner and outer surfaces of the cells would be uniformly wetted through the agency of the distributing channels.

The cells give a very stable structure within the tower and they are built up in a regular manner, just like bricks, but without the necessity of paying special attention to their spacing, or to preserving definite intervals between them; the projections at the sides of the cells do this automatically.

The "active surface" of the large cells is 15 sq. ft. per cubic foot and of the small cells 30 sq. ft. per cubic foot, and they are carefully designed and proportioned in every way to give the very best results. Reverting to the question of "free space" we find that both sizes of cells have a free space of 71 per cent.; for all other varieties of filling material the "free space" is at most 40 to 50 per cent.

Let us now take an example:

A tower is required to deal with 1000 cu. ft. of gas per minute, and from experience and research it has been found that for efficient and economical absorption the vertical velocity of the gas up the tower should be 50 ft. per minute. It follows that, when using an inferior filling material and assuming 50 per cent. "free space" (to take the figure most unfavorable for

the cells), a tower of $\frac{1000}{50 \times 0.50}$ sq. ft. = 40 sq. ft. sectional area is required, so as not to exceed the desired vertical velocity

of the gas. Under precisely the same conditions, and em-

ploying cells only $\frac{1000}{50 \times 0.71}$ sq. ft. = 28.2 sq. ft. section are

required for the tower. It follows that the area of a tower which is filled with inferior filling material must be greater by at least 42 per cent. than that necessary when using the best filling material, the cells.

Let us now compare the total cost of two towers as above, made of acid-proof bricks and filled to a height of say 24 ft.

	Ordinary Tower.	Cell Tower.
Wetted surface, sq. ft. per cu. ft.....	10	15
Free space (maximum).....	50%	71%
(Usually between 35 and 40% only)		
Required area of tower, sq. ft.....	40	28
(To give same free volume)		
Total volume of filling material, cu. ft....	960	672
Total wetted surface in tower, sq. ft....	9600	10,080
The comparative costs under American conditions, exclusive of import duty, may be estimated as follows:		
Estimated cost of 1 tower, as above, including foundations and erection, but exclusive of all pottery parts.....	\$3,600	\$2,700
Filling material, 960 cu. ft., at \$0.70 per cu. ft.	672	
Filling material, 672 cu. ft., at \$1.40 per cu. ft.		940
	\$4,272	\$3,640
Saving		\$632

With an equally efficient tower a considerable net cash saving, therefore, results from the use of more expensive cellular filling material. The cell tower is, however, far preferable, because of the saving in ground space and buildings, smaller cost of upkeep of smaller towers and reduced quantity of liquid that has to be circulated (the quantity being proportional to the area of the tower). The last factor in particular is of great importance, as it represents an annual saving which, if capitalized, amounts to a large sum, while in addition, fewer Montejuz, etc., may be required.

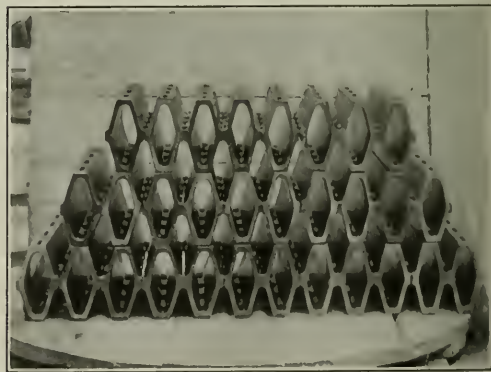


FIG. 2—A SYSTEM OF GUTTMANN CELLS
GREATEST WIDTH, 100 MM.; HEIGHT, 200 MM.; LENGTH, 500 MM.:
1000 CELLS PER CUBIC METER

The saving to be expected in the case of towers of other dimensions or with different filling material can be readily determined by a suitable modification of the figures given.

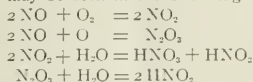
Cells in the "Nitric Acid from Air" Industry and for Artificial Silk, Celluloid and Similar Factories

One of the most important uses to which absorption towers of the above described "I" type are put to is as reaction and absorption towers for these industries.

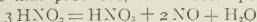
Such towers must deal with *air* containing about 2 per cent

NO, and in order to understand what takes place, it is desirable to first examine the nature of the reaction taking place when NO is absorbed by water.

What happens may be seen in the following equations:



Nitrous acid is thus produced, which decomposes as follows:



This equation demonstrates that NO is continually regenerated during the process of absorption.

The oxidation of NO to NO₂, which is shown in the equation, is a so-called "time reaction," which is therefore to a great extent independent of the nature of the filling material.

It is specially important that for such uses the tower should be equipped with filling material, which gives not only the necessary active surface and mixing action for proper absorption, but also the greatest possible "free space."

As the oxidation occupies a definite fixed time, so also the required area of the tower, as in the previous example, depends on the "free space" or "absorption space," and the same calculations hold good. Large cells are particularly suitable for securing these conditions on account of the low total installation costs of a tower or towers filled with them.

The "active surface" per cubic foot plays a smaller part in this case than in that first considered, as the chief essential for the good working of such "reaction towers" is a maximum "free space" together with sufficient "active surface."

Similar absorption plants are now being used in the artificial silk industry and in connection with celluloid and similar works. In such cases pure NO gas usually has to be dealt with, which, by oxidation with air, is diluted to a gas containing 5 to 10 per cent. NO.

The cells described in this article are being manufactured by the Deutsche Steinzeugfabrik, of Friedrichsfeld, Germany, their agent in this country being Mr. J. W. Sittig etc.), 5 Beekman Street, New York City.

Bakelite Patent Recognition

As a result of negotiations between the General Bakelite Company and the Condensite Company of America, the suits brought by the former against the latter and its customers for alleged infringement of the bakelite patents, have been withdrawn, and the Condensite company, recognizing the pioneer character of Dr. L. H. Baekeland's work, has acknowledged the validity of the bakelite patents in suit and some others, and will pay substantial royalties thereunder. The General Bakelite company will continue the manufacture of bakelite under its numerous patents, and the Condensite company will manufacture condensite under the Aylsworth patents, as well as the license just granted for such of the Baekeland patents as are broad enough to cover condensite.

This recognition of the force and validity of the principal Baekeland patents by the only other manufacturer of condensation products is a confirmation of the Baekelite company's claims for the broad scope and pioneer character of these patents. There is much that is old in the art of making phenolic condensation products, but Dr. Baekeland was the first to make practical application of what theretofore had been chiefly laboratory experiments.

Production of the Diamond in Electric Furnaces*

By Francis P. Mann

Should the results obtained by a metallurgist of Paris, E. de Boismenu, prove to be all that is claimed for them, his method is likely to work a revolution in the production of the diamond. The inventor has just published the results of his work in a brochure** which appeared at Paris, but the matter has as yet not found its way into the technical press.

Our attention having been called to this publication, it at

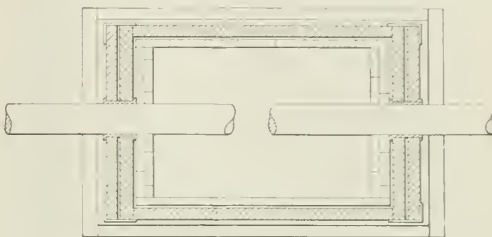


FIG. 1—ELECTRIC FURNACE FOR ARTIFICIAL DIAMONDS

once awakened our interest, especially as we formerly had occasion to observe the work of the late Professor Moissan, of the Paris University, in the formation of the diamond. We accordingly made a visit to the inventor, and he allowed us to inspect a collection of the minute diamond crystals which he obtained, the largest of these having 2.7 mm. (7/64 in.) length. However, as will be further noted, this is believed not to be by any means the possible limit of size, and the inventor is confident of obtaining large crystals of the diamond in a more extensive plant adapted for the purpose.

A comparison with Professor Moissan's process is at once suggested, and this is quite in favor of the new method. It will be remembered that Moissan based his process on the idea of using an extremely high pressure in order to crystallize carbon. Melting iron with an excess of carbon in the electric furnace, he plunges the mass into water so that the surface cools and solidifies around a still liquid inner portion. The violent contraction and resulting pressure on the inside part which is thus obtained acts to bring about a crystallization of carbon in the shape of microscopic diamonds. The largest size Moissan could obtain was a 0.7 mm. (1/32 in.), and there seemed to be no great hope of increasing this size so that the method now has purely a scientific interest.

Quite the contrary is true in the process used by M. de Boismenu. Pressure does not intervene here, and the method consists simply in electrolyzing a fused bath of calcium carbide in an electrolytic furnace by the use of direct current. The heat should last for several hours, and at the end of this time there is found decomposition of the carbide with formation of small crystals of pure carbon. As the size of the crystal depends on the length of the operation, the limit of size cannot be conjectured. It took the inventor a 12 hours' run to obtain his largest size of 2.7 mm. and his experimental plan prevented him from going further, for otherwise he had reason to believe that the size would continue increasing, to what point it is yet unknown.

The plant which he set up in order to prove the value of his theories was located at Paris and was a very primitive one. The main drawback was that the lighting company would not furnish the current except during the day on account of the disturbances to the circuit. He ran a small 50-kw motor which was coupled to a pair of electrolytic generators producing 800 to 1200 amp each at 15 to 40 volts, using these to supply his

*This article is published as an interesting news item, although various points in the process and its description are in need of further explanation.—Editor

**Fabrication Synthétique du Diamant.

electrolytic furnace. The inventor's results were obtained as far back as 1908, but for a number of reasons they have not been made public until the present time. As it was impossible to make the longer run which is needed to increase the size of the crystals, the plant was stopped during that year. The inventor is endeavoring to obtain the necessary resources for putting up a second and more elaborate plant.

The experimental furnace which he used in the present work was a simple firebrick construction, such as Fig. 1 shows, and had an inner space of 75 cm (30 in.) length by 50 cm. (20 in.) width and the same depth. The bottom was made up of layers of brick with the top part laid out so as to form a V-shaped bed, while the sides had a firebrick lining which could be replaced if need be after the operations. Upon the sole he used a packing of a mixture of powdered lime and carbon containing 80 per cent lime, and as a container for the mass of carbide in fusion he employed a semicircular-section trough made of fused carbide itself, which he found after some trial to be the best. Subsequently he reduced the inside width of the space to 12 cm. by adding firebrick.

As electrodes he used a pair of round carbons of 165 mm. ($6\frac{1}{2}$ in.) diameter and 1.50 m. (5 ft.) length. He then proposed to form a bath of melted carbide within the trough and to keep this mass in fusion by the heating effect of its resistance, as is usual in such types of furnace. The two carbons worked within the carbide trough, the positive carbon being fixed and the negative carbon adjustable in the well-known way.

With the present arrangement he was at first successful in producing small crystals of carbon on April 13, 1908, running on this occasion with a heat of six hours' duration. At the

The current was finally cut off at 4.45 p. m. At 3 o'clock he had fed in the maximum amount of carbide, and then covered the whole with somewhat large pieces of carbide under which the melted mass kept under the action of the current, and upon the whole he placed the above-mentioned lime and carbon mixture, then covered the furnace with a pair of firebrick slabs. The furnace ran in this way until the end of the test.

After allowing to cool off overnight, he opened the furnace and took out the fused mass which adhered to the two carbons. It was seen that a marked electrolysis had taken place. There was, in fact, a striking difference between the negative portion and the rest of the block.

The middle part contained calcium carbide in an apparently unchanged state, while around the positive carbon there was a mass of carbide which appeared to be finer and better crystallized than the ordinary. As concerns the part lying around the negative pole, it appeared to be made up exclusively of carbon in a light and spongy state and black color. This mass was afterwards found to contain flakes of graphite and also minute crystals of pure carbon.

Treating the black mass with water he reduced it to the state of a muddy deposit and then dried the same. He was then able to pick out small diamond crystals, which in this experiment did not reach more than 1 mm. (0.04 in.) in length. These crystals when viewed under the microscope with an enlargement of 20 diameters showed the appearance of the diamond, and beams of light directed upon them at various angles caused them to glow with the remarkably brilliant hues which characterize the diamond. The crystals which the inventor obtained in various experiments were then submitted to two eminent Paris scientists, both members of the Academy, M. Lacroix, professor of mineralogy at the Natural History Museum (Jardin des Plantes), and M. Maquenne, professor of applied chemistry at the College de France. After applying the well-known physical and chemical tests, they both pronounced the specimens to be crystals of pure carbon and consequently diamonds.

Although the crystals obtained in this way are very pure, their form is on the contrary quite irregular and variable. Some of the crystals show two and even three well-defined faces, and others show but one. Still others are rounded off entirely without showing the least face or angle, and the inventor possesses quite a number of specimens of very small size whose nature could not be perceived without the aid of the microscope, as they resemble small grains of transparent silica or melted glass. Some of them have the appearance under the microscope of rounded drops. He considers the shape of these pieces to be the best proof of the authenticity of his specimens, for it would be impossible to form such grains by an artificial process.

We remember, in fact, that Professor Moissan showed us some specimens of his products under the microscope which had the appearance of rounded drops, and he gives some considerations upon this subject in his treatise on the formation of the diamond. The specimens obtained by M. de BoisMENU, and which he kindly allowed us to observe, also gave examples of rounded and elongated masses and these appear to be of an analogous character. It would not be possible to procure such minute rounded specimens by breaking up fragments of the diamond, as all who are familiar with the subject will admit at once. We should state that the carbonaceous mass contains quite an amount of even smaller particles down to the minutest grains.

Continuing his work with the electric furnace on about the same lines as described, he made thirteen separate tests during the following two months, in all of which he was successful in producing diamond crystals. What he noticed specially during the experiments was that there was a marked proportion between the length of the operation and the size of the crystals. In fact, the crystals appear to grow in size according to the well-known phenomena of crystallization. This is one of the most important points in favor of his method for he hopes

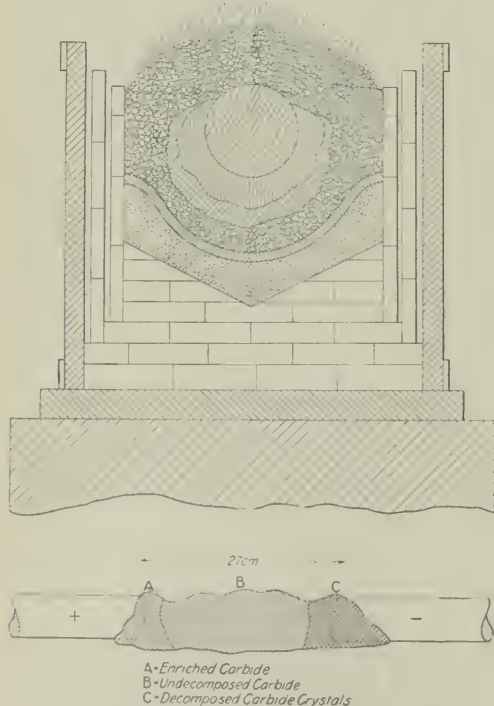


FIG. 2—RESULTS OF EXPERIMENT

start he fed in small particles of carbide around the electrodes, then gradually separating them as the mass heated up, until he finally had 3.5 kg. (8 lb.) of carbide under treatment. The maximum distance between carbons was 24 cm. ($9\frac{1}{2}$ in.).

During the last three hours of the run the electrical conditions remained rather steady at 34-35 volts and 800 amp.

in the future to secure much larger specimens by applying a longer heat.

It is noticed that by comparing the lengths of operation and size of crystals, the rate of growth is about 0.2 mm. (0.008 in.) per hour. He summarizes these results as follows, showing the lengths of his tests and maximum sizes of crystals:

Length of operation, 6 hour.	Size of crystal, 1.5 mm.
" 9 "	" 2.1 "
" 11 "	" 2.5 "
" 12 "	" 2.7 "

He was unable to run his furnace for more than twelve hours at a time, as we already mentioned, so that nothing leads up to suppose that he had arrived at a limit. Admitting the above rate, and extrapolating a 42-hours' continuous run would give diamonds of 8.7 mm., which is already a large size, and a run of 72 hours would produce the remarkable size of 15 mm. and weighing 60 carats. Naturally this is only a surmise, but there is nothing in the nature of the case to contradict the possibility of such results. What is certain is that he produced crystals of 2.7 mm. on a comparatively short run and with a primitive plant and using ordinary commercial material.

The objection, and apparently a serious one, was made that according to the phenomena of electrolysis, the carbon separated from the calcium should be found not at the negative but at the positive pole. However, the negative side of the bath was found invariably decomposed, and here were seen the crystals and never on the positive side. This anomaly is explained upon observing the working of the furnace, as during the second part of the operation, the negative region constantly gives rise to flames of a bright rose color of characteristic appearance. Calcium in the state of incandescent vapors escaping from the furnace forms these flames, and this element set at liberty at the cathode sets free the carbon with which it was united as calcium carbide. Part of the carbon remains isolated at this pole and is imprisoned and crystallizes in the mass, as the inventor supposes, while the rest of the carbon goes to the positive pole and enriches this part of the bath. The inventor's method is covered by French patent No. 4566, Aug. 24, 1907.

Paris, France.

Condensing Equipment.—The city of Cleveland, Ohio, has awarded a contract to the C. H. Wheeler Manufacturing Company, of Philadelphia, Pa., for surface condensers for the new municipal electric light plant. This plant which will be a model one in every way, will consist for the present, of three 5000-kw Allis-Chalmers turbines with C. H. Wheeler high-efficiency surface condensers having motor-driven Mullan patent vacuum pumps and motor-driven centrifugal circulating and hot-well pumps. The condensers will be installed under rigid efficiency guarantees covering performance under widely varying loads and water temperatures, the normal operation being 28½ in. vacuum, although with cold water and light loads the performance guarantee requires the production of a vacuum within ½ in. of the barometer. The contract was largely awarded on the basis of the extremely low operation cost and low maintenance guarantees made by the C. H. Wheeler Manufacturing Company.

Remedies for Condensation Under Concrete Roofs.—In a recent interview Mr. Leonard C. Wason, president of the Abertaw Construction Company, Boston, was asked what was a simple method of preventing condensation under a concrete roof of a power house in Boston. At the present time the owners are running a steam pipe under the roof to keep it warm. Mr. Wason stated that, of course, the thing to do is to insulate the roof, and he recommended taking off the gravel and putting down an insulating felt, a number of which are on the market. As an alternative Mr. Wason recommended fastening sheets of cork to the under side of the ceiling. This would cost from 6 cents to 8 cents per square foot.

Refuse Destructor Test

The results of the official tests of the Sterling refuse destructor, installed at Halifax, N. S., by the Griscorn-Russell Company, of New York City, have just been made public.

The test was made on March 8, 1913. It lasted three hours. The number of grades was three, the total grate area 75 sq. ft., the Abendroth-Root water-tube boiler heating surface 2034 sq. ft.

Weights:

Total amount refuse consumed.....	24,072	lb.
Composition House ashes	75%	18,054 "
Wet garbage	16%	3852 "
Rubbish	9%	2166 "
Lb. burned per sq. ft. grate per hr. grate No. 1		89 "
" " " " " 2		73 "
" " " " " 3		71.5 "
" " " " " Average		77.7 "

Water Evaporation:

Temperature feed water.....	40°
Boiler pressure (gauge)	145 lb. per sq. in.
Factor of evaporation.....	1.227
Total water evaporated	29,992 lb.
Evaporation from and at 212°	36,800 lb.
Equivalent boiler horsepower.....	258
Equivalent evaporation from and at 212° per lb. refuse consumed	1.52

Temperatures and Pressures:

Temperature combustion chamber Fahr. Maximum	1547°
" " " Minimum	1255°
" " " Average	1357°
Percentage C.O. 2	Average 9.44
	Maximum 12.5
	Minimum 7.25

Air pressure in ash pit.....	2.75 in. water gauge
From Regenerator	3.75 in. water gauge
Into Regenerator	4.25 in. water gauge
Temperature outside air	30°
Temperature Air from Regenerator.....	180°
Temperature Gases into Regenerator.....	475°
R. P. M. fan.....	375
Duration clinkering periods	8 min.
Interval between clinkering periods average.....	62 min.
Labor cost per ton—city wages.....	28c.

The combustion of refuse at the test was at the rate of 70 tons in 24 hours exceeding the guaranteed capacity by 40 per cent. The calorific value of the refuse was very low, the analysis of the residues on one test showing 40 per cent of fine dust, 37 per cent of ash, and 23 per cent of clinker.

Personal

Mr. Charles O. Bannister was awarded the Consolidated Gold Fields gold medal of the Institution of Mining & Metallurgy, for his paper on the theory of blast-roasting of galena.

Mr. Sidney Cornell has severed his connection with the Pittsburgh Crucible Steel Company to assume a position with the Griscorn-Russell Company, of New York City, to introduce the various products of this company into the chemical industries.

Dr. James Douglas delivered the annual commencement address before the senior class of the Colorado School of Mines, May 23.

Mr. Robert C. Gemmel, until recently assistant general manager for the Utah Copper Company, has succeeded Mr. D. C. Jackling as general manager. Mr. Jackling retains the position of first vice-president of the company, and has also been appointed managing director.

Dr. Theodor Goldschmidt, of the Tin and Chemical Works of Theo. Goldschmidt in Essen-Ruhr, Germany, is making a trip through the West to study latest metallurgical developments.

Mr. W. J. Hamilton has succeeded Mr. A. B. W. Hodges as general manager of the Cerro de Pasco copper property.

Peru. Mr. Hamilton formerly was smelter superintendent for the same company.

Mr. **Justin H. Haynes** has been retained to design a new cyanide mill for the Junta Consolidated Company, at Telluride, Colo.

Dr. and Mrs. **Paul Heroult** have returned from France on a visit to this country.

Mr. **Louis D. Huntoon** has been appointed eastern representative for the Stearns-Roger Manufacturing Company, of Denver, and will have his office at 115 Broadway, New York.

Mr. **E. W. Lawler**, well known through his long connection with the ball-mill and tube-mill business, has resumed his position with the Abbe Engineering Company, of New York City, with whom he was connected a number of years ago.

Dr. **L. D. Ricketts** is in New York for several weeks.

Mr. **J. B. Risque** has been appointed general manager for the Tennessee Copper Company, with headquarters at Copperhill, Tenn.

Mr. **Guilford D. Scholl**, formerly with the U. S. Metals Refining Company, at Chrome, N. J., has been appointed assistant superintendent of the Electrolytic Refining & Smelting Company, Port Kembla, N. S. W., Australia.

Mr. **E. G. Spilsbury**, of New York, is spending about two months on a professional trip to Colorado and Arizona.

Mr. **Bradley Stoughton**, secretary of the American Institute of Mining Engineers, has been to St. Louis in connection with the establishment of a local section of the Institute at that place.

Mr. **Maximilian Toch**, of New York City, widely known among the chemical fraternity by his extended work in the chemistry of paints, will sail for Europe on May 31 to deliver a lecture before the Paint and Varnish Society of London on Tropical Paints. On July 16, Professor Bogert and Mr. Toch, the American President and Vice-president of the Society of Chemical Industry, will attend the annual meeting of the Society in Liverpool.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Concluded).

714,599, November 25, 1902, Sidney Theodore Muffy, of Bowdre, Georgia; assignor of one-half to Runyon Pyatt, of New York, N. Y.

Relates to a process for the separation of metals from their solutions by electrodeposition and refers to the application for his apparatus patent 714,598. The gold-bearing solution, which may be a cyanide, chloride, etc., flows through the cathode of a number of precipitating boxes, in cascade; the precious metal being precipitated upon the cathode, which consists of a cellular casing containing a carbon plate, and a mass of filiform lead and zinc alloy, consisting of 20 parts of lead and 80 parts of zinc. Compressed air is blown through the cathode, called a "mattress-cathode" to assist in the circulation and depolarization. The cathode is surrounded with a fabric cover to prevent loss of any gold particles. The solution leaving the first box flows through a second, and so on; when a box is filled, it is removed and a fresh one inserted at the end of the series, each one being moved up. The filled cellular carbon mattress-cathode containing the gold and any remaining lead-zinc alloy, is incinerated, the ashes collected and carefully smelted to bullion in the usual way.

714,861, December 2, 1902, David H. Browne, of Cleveland, Ohio; assignor to the Canadian Copper Company, of Cleveland, Ohio, a corporation of Ohio.

Relates to an electrolytic process for the separation of copper and nickel from their alloys. The alloy is cast into anodes, and also converted into fragments of powder, the latter is

placed in a suitable receptacle wherein it is treated with salt water and chlorine gas, the alloy being dissolved forming cuprous chloride and nickel chloride. This solution is electrolyzed with a copper-nickel alloy anode and a suitable cathode, the liberated chlorine dissolving the anode forming a further quantity of cuprous chloride and nickel chloride; with deposition of all the copper on the cathode, the solution leaving the cell containing the nickel chloride and traces of iron which is precipitated by sodium hydroxide. The nickel chloride solution is now separately electrolyzed, the chlorine liberated at the anode being collected and used to convert the original copper-nickel alloy fragments into their chlorides, and the nickel being electrically deposited.

REFINING.

209,056, October 15, 1878, Nathaniel S. Keith, of Brooklyn, New York.

Relates to refining impure lead by making the same the anode in an electrolyte of lead acetate, lead chloride, or lead nitrate, acidulated with either acetic or hydrochloric acids, to prevent the formation of sub-salts and oxides upon the electrodes. The anodes are enclosed in bags of some textile material, such as cotton, linen, or wool, which collects the insoluble elements, such as gold, silver, antimony, copper, and other constituents of impure pig lead. The cathode consists of copper, brass, lead, carbon, or other suitable material. The lead is deposited in a pure crystalline metallic form, and may be removed periodically and melted.

214,344, April 15, 1879, Emil Andre, of Ehrenbreitstein, Germany, Assignor to Paul Holder, of New York, N. Y.

Relates to refining base metals or alloys and uses an apparatus consisting of an outer tank and two inner concentric cells each having a porous bottom. For refining an alloy of gold, silver, and copper, a saturated solution of copper sulfate, containing from three to five per cent of sulfuric acid is used. The anodes are suspended within the innermost porous cell, the cathodes in the outer tank. At the anode copper and silver go into solution, the gold falling to the bottom of the anode cell. The silver may be precipitated within the cell by protoxide of iron or any other reagent, and the copper solution removed and filtered, or the copper-silver solution may be removed and brought into contact with granulated copper, precipitating metallic silver. The purified copper solution is now returned to the cathode compartment. Within the space between the two concentric porous cells is a constantly flowing diluted acid solution to carry off any anode slime that filters through the anode cell. The exhausted catholyte is returned to the anode compartment.

264,927, September 26, 1882, Henry R. Cassel, of New York, N. Y., Assignor to the United States Bullion Refining Company, of same place.

Relates to an apparatus for the electrolytic separation of base bullion, etc., containing gold, silver, and copper. The apparatus consists of a suitable tank containing an electrolyte in which are suspended an anode of the alloy or metal to be refined, and which if granulated, is contained in a suitable bag or box, of fabric or perforated material. Between the anode and cathode is a metallic diaphragm consisting of copper wire-netting, or a bag or box filled with small pieces of copper or other metal, or simply a series of vertically suspended wires. The function of the copper diaphragm is to precipitate the silver dissolved as silver sulphate from the anode; it is collected as metallic silver from the bottom of the vat, leaving the copper sulphate substantially pure. When anodes are used in the shape of plates, etc., a box or other receptacle is hung underneath it to collect any gold slime, etc.

264,928, Sept. 26, 1882, Henry R. Cassel, of New York, N. Y., assignor to the United States Bullion Refining Co.

Relates to a process of refining alloys, base bullion, etc., and consists in enveloping the cathode in a covering of leather, parchment, canvas or other porous material that will permit the current to pass, but not permit the deposition of metal upon the cathode. The anode metals accumulate in the electrolyte.

which, from time to time, is removed and purified; the copper can then be deposited separately.

305,192, Sept. 16, 1884, James S. Howard, of Springfield, Mass., assignor to J. M. Abbott, of Waterbury, Conn.

Relates to an improved solution for refining copper, and separating gold and silver from copper, and copper and other ores. The solution consists of twenty-five parts of potassium bisulphate, two parts of hydrofluoric acid, twenty parts of sulphuric acid, three parts of any of the chemicals containing nitrogen, such as nitrate of soda, potash, etc., and fifty parts of water, and heated to about 175 deg. Fahr. The copper, or ore, is placed in the solution and a plate of carbon in contact with the copper, or the ore. A current due to local action is set up and the copper dissolved. The gold or silver present is caught by a suitable receptacle. The copper may be recovered from the solution by electrolysis, or by tin scrap, iron, etc. Several such arrangements may be connected in series, so that the current generated may be utilized to operate a copper-plating bath to recover some of the copper dissolved.

310,533, Jan. 6, 1885, Bernard Moebius, of Chihuahua, Mexico, assignor of three-eighths to William E. Pope, of Athens, Ga.

Relates to an apparatus for refining silver, and separating it from the gold, platinum, copper, lead, and other metals present. The apparatus consists of a tank or tanks containing anodes and cathodes, and an electrolyte, stirrers to keep the electrolyte in agitation, and sets of reciprocating brushes to constantly remove the anode slime from the anodes, and the deposited silver from the cathodes. A steam pipe runs through the tank whereby the temperature may be regulated. The anodes are suspended in frames which are surrounded with bags to catch the anode slimes; additional bags surround both the enclosed anodes and cathodes. Platinized carbon cathodes are suggested.

Book Reviews

Steel Rails. By Wm. H. Sellew. $7\frac{3}{4} \times 10\frac{3}{4}$ inches (20 x 28 cm.), 576 pages, 361 illustrations, 33 folding plates. Price \$12.50 net. New York City: D. Van Nostrand Company; London: Constable & Company.

The author is principal assistant engineer of the Michigan Central Railroad. The title of the book lacks definiteness, making it necessary to state that only one-fifth of the work is on the manufacture of rails, and four-fifths on properties and uses.

The first chapter is on the "development of the present section"; it is brief but comprehensive. The second chapter is on the "pressure of the wheel on the rail." This is discussed at length, and in a manner which will greatly interest track engineers. The next chapter, on "supports of the rail," is a hundred-page treatise on "ties," wherein the steel tie is exhaustively studied. Two chapters, totalling over one hundred pages, deal with "stresses in the rail" and "strength of the rail," covering the actual conditions of its use; while the last chapter covers rail specifications.

The next to the last chapter, of 135 pages, sets forth the "influence of detail of manufacture," and considering this more at length, because it is the only part of the work dealing with the chemistry or metallurgy of rails, and also because it is the part we feel most competent to pass judgment upon, we have the following to say: 22 pages on the manufacture of pig iron is elementary, at places defective, and not to be taken as other than a superficial sketch of the subject; 30 pages on conversion into steel is likewise fragmentary, but present practice is well stated and beautifully illustrated; 25 pages on "the ingot" is very well condensed, the author frequently putting his finger upon the really important but often overlooked sources of weakness; 43 pages on "influence of mechanical work" are partly very elementary and partly composed of quotations from literature. On the whole, this chapter is unequal in treatment, part being so elementary as to be rudimentary, and part being

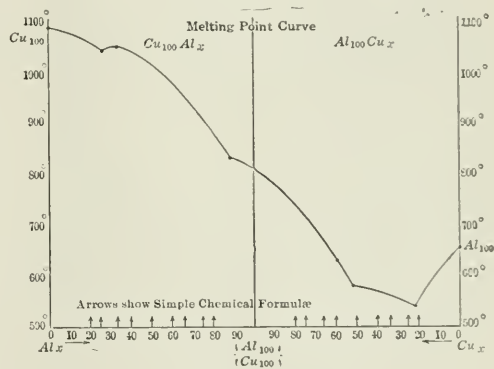
written as if by an expert for experts. We have a shrewd suspicion that the other parts of the book will meet a similar criticism from competent critics.

The work is beautifully printed and illustrated, heavy calendered paper being used throughout. It has been a work of great labor to accumulate the material presented, and no other book known to the reviewer gives as much reliable information on this subject. We congratulate the author on completing so valuable a contribution to technical literature.

* * *

Atomprozente und Gewichtsprozente: Sowie die Methoden zu Ihrer Gegenseitigen Umwandlung. By Dipl.-Eng. Fritz Hoffmann. 19 x 28 cm., 20 pages, 18 diagrams and 2 double (superimposed) plates. Price 1.80 Marks. Halle: Wilhelm Knapp.

This monograph is intended to furnish metallographists with a simple, direct method of passing from percentage composition of an alloy by weight to percentage composition in atomic numbers, and *vice versa*. Simple mathematic formulæ are given for passing from percentage on one basis to percentage on the other basis, for systems of two components and of three components. Graphical methods are then shown, whereby given the curve of properties on one basis of percentage composition it can be transformed into a corresponding curve for



the other basis. This is a comparatively simple matter in binary systems using the reference-line method; but becomes difficult for ternary systems, using the tri-axial diagram. In the latter case, a double system of reference lines is proposed, one in red ink and on oiled paper, fitting over those in black ink, so that for any spot on the diagram or curves the percentage compositions on both bases can be read off.

The work is very clever, and will help those metallographists who prefer to record or publish their results on the atom percent basis. Our conviction is that the weight-percent basis is much to be preferred for recording experimental results, and should in no case be omitted. The atom-percents are not suitable for directly recording experimental results, and are not in advantageous form for theoretical discussion. A much more logical diagram for the latter purpose is obtainable by plotting, as abscissas from left to right 1 to 100 atoms of A (to 100 of B) and from right to left 1 to 100 atoms of B (to 100 of A). Joining the two diagrams in the middle, which represents $A_{100}B_{100}$, the two halves cover all possible variations from $A_{100}B_{100}$ to $A_{100}B_{100}$ and thence to $A_{100}B_{100}$. An example is given in the adjoining diagram.

* * *

The Calorific Power of Gas. By J. H. Coste. 12mo (13 x 19.5 cm.), 310 pages, 57 illustrations; price \$2.00 net. London: Chas. Griffin & Co., Ltd.

This is a treatise on calorific standards and calorimetry, and concerns itself principally with illuminating gas. It will be useful however, to the producer and user of combustible gas for any chemical or metallurgical purpose. A large amount

of space is given to the official methods prescribed for testing gas, the higher and lower calorific powers, the calibrated and use of still-water and constant-flow calorimeters. While the information is given in a slightly old-fashioned dress, (usually in British thermal units and cubic feet), yet it is almost always reliable and satisfactory, and the work can be recommended as valuable, without any reservations.

* * *

Chemiker-Kalender, 1913. Von Dr. Rudolph Biedermann. 2 volumes, 9 x 15 cm., 661 and 402 pages. Price \$1.50 in New York. Berlin: Julius Springer.

This well-known chemical year book is better than ever. The more than a thousand pages is most carefully compiled and proof-read, so that errors seem to be practically absent. Where divergent data exists a wise choice is usually made of the most probable values. A characteristically reliable and complete German production; we cannot imagine any technologist to whom it might not be valuable.

* * *

Determinative Mineralogy. By J. Volney Lewis, Professor of Mineralogy in Rutgers College. 12mo. (13 x 20 cm.), 151 pages, 68 illustrations; price \$1.50. New York: John Wiley & Sons. London: Chapman & Hall, Ltd.

This book contains 60 pages of brief description of blowpipe tests and physical properties, including crystallography, followed by 34 double pages of carefully-prepared tables, printed on strong bond paper and neatly inserted. These latter are modelled largely on the Brush-Penfield tables, which in turn were based on the tables of Dr. Roepper, the first professor of mineralogy in Lehigh University.

We find the book very carefully compiled, showing internal evidence of the author's mastery of the smallest details of the subject, such as comes only from a long campaign of laboratory teaching experience. We thoroughly agree with his principle that "chemical composition is the most fundamental property of a mineral," and we even wish that he had used that principle still more extensively in the early stages of his classification.

A few oversights mar the high quality of the tables: e.g. the erroneous statements (p. 64) that all minerals of metallic or sub-metallic luster "black or dark-colored streaks," and all of non-metallic luster "light colored or white" streaks; on p. 78 the coat on charcoal is omitted as distinguishing molybdenite from graphite; in many cases round-about net tests are mentioned when the shorter and much more satisfactory blowpipe tests are far preferable—contrary to the author's previously expressed preference for the blowpipe tests wherever applicable; all organic minerals (coals, pitches, etc.) are omitted.

In spite of these and other imperfections, it is a creditable and useful book, which can be made still more so in a revised edition.

* * *

Oil Fuel: Its Supply, Composition and Application. By Sydney H. North, revised and enlarged by Edward Butler. 12mo (13 x 19.5 cm.), 238 pages, 107 illustrations. Price, \$2.00 net. London: Chas. Griffin & Co., Ltd.

While the use of oil as fuel is decreasing in some parts of the world, yet it is increasing in others, such as Texas and California, and this resumé contains therefore information of great value to many industries. The illustrations are almost all working drawings to scale, and show all varieties of oil burners and their attachment to various furnaces. The discussion of chemical composition of fuel oils and the conditions of their combustion is rather brief; the larger part of the book is then taken up with descriptions of burners, and their use for marine, naval, locomotive and automobile purposes, while a rather short chapter is devoted to metallurgical and chemical purposes. The book is, therefore, written more from the mechanical engineer's, or perhaps we might say from the fuel engineer's, standpoint, rather than from that of a chemist or metallurgist; but at any rate it is a useful, practical book.

Explosives: A Synoptic and Critical Treatment of the Literature of the Subject. By Dr. H. Brunswig. Translated later and annotated by Dr. Chas. E. Munroe and Dr. A. L. Kibler, of George Washington University. 12mo (13 x 21.5 cm.), 350 pages, 45 illustrations. Price \$3 net. New York: John Wiley & Sons. London: Chapman & Hall, Limited.

This is another of those works which prove the vitalizing influence of modern science on century-old industries. A great accumulation of experimental material was awaiting the informing touch of physical chemistry, and this science has annexed the whole field as a practical application of the velocities of chemical reactions and chemical equilibria at very high temperatures and pressures.

A perusal of the work is fascinating; it forms one of the finest object lessons in applied science ever written. All of the physics and chemistry of high temperatures and high pressures are put under contribution, and many principles useful in other fields are concretely illustrated. To read it understandingly requires a working knowledge of advanced physics and chemistry, especially of specific heats, latent heats of fusion and vaporization, thermo-chemistry, chemical equilibria and velocities of chemical reactions; but the reader thus equipped will be charmed with the subject.

No one practically manufacturing or testing explosives can afford to be less than master of the contents of this excellent work.

* * *

Accumuladores Eléctricos. Theory, Construction, Installation, Handling, Using and Calculation. By Gaston Ossa, Assistant in the Electrochemical Laboratory, University of Chile. 18 x 26 cm., 103 pages, 55 illustrations. Price \$1. Santiago de Chile: Sociedad Imprenta y Litografía Universo.

Asked to install a storage battery in the laboratory of the university, the author found the Spanish books on electro-technology totally lacking in practical directions for this work, and the foreign books were either too voluminous or inaccessible—at least so the author states. Perceiving the need of a short treatise for the use of the Spanish-reading public, this book was compiled. Eight short chapters cover the principles, electrodes, electrolyte, installation, testing, irregularities, manner of using, and calculation of batteries for a given service. The writing seems to be done with full command of the details of the subject, wise selection of the important points, and expressed in clear language (containing, however, interesting deviations from the pure Castilian, characteristic of our Chilean neighbors).

We assure Mr. Ossa that he has most creditably won his spurs as a scientific writer, and wish him still further occupation and success in this line.

* * *

The Principles of Electro-Deposition: A Laboratory Guide to Electroplating. By Samuel Field. 12mo (13 x 19 cm.), 383 pages, 121 illustrations. Price 6 shillings (\$1.80 net). London and New York: Longmans, Green, & Co.

An extensive but quite elementary work. The fundamental principles are well explained, but practical details are scanty. Many modern ideas or usages are not noticed or regarded. The use of the term coulometer instead of voltameter should be adopted. A kilowatt hour is designed as a B.T.U., which is still a common usage in Great Britain, but should finally be discontinued, as B.T.U. in English literature has three different meanings. It may mean either British Thermal Unit or Board of Trade Unit and as such it may mean either kilowatt hour or kilowatt. Positive and negative poles are marked +ve and -ve, respectively, instead of simply + and —.

Aside from these minor faults, which give a somewhat old-fashioned impression, the book is carefully written and will serve as a useful introduction to the subject for beginners to read.

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Electric Iron Ore Reduction in California

It is with great pleasure that we publish in this issue the first authentic report of the latest advances in the pioneer work of the Noble Electric Steel Company in Shasta County, California, in electric iron smelting. California and Scandinavia were the pioneers. Both started almost simultaneously, but independently, and developed independently, and as Mr. Crawford's most interesting article shows, the differences between Californian and Scandinavian practice and design have increased lately more and more, so as to suit local conditions. Now that electric iron smelting has become an established industry of this country the congratulations and good wishes of all interested in electro-metallurgical progress are due to the sturdy Californian pioneers and to their leader, Mr. H. H. Noble.

Testing Plants

The action of Colorado's governor in vetoing an appropriation of \$40,000 for the completion and operation of the ore-dressing and metallurgical laboratory of the Colorado School of Mines calls attention to the proposed functions of that department of the school, and to the limitations which the governor's action will place upon it. In his own defense, the governor claims a lack of funds to meet the appropriation; but the governor, be it known, is distinctly an agriculturist, and as such possibly lacks an appreciation of the needs of a mining school, especially in a state where agriculture has temporarily overshadowed mining.

At present the experimental laboratory at the school is only partly equipped. It was designed for a two-fold purpose—education and investigation, but not for ore testing in the ordinary sense of that word, meaning the treatment of an ore by known routine methods and merely reporting the results, as is done by many commercial ore-testing plants. In its educational function it is intended to afford a laboratory for the students working under the direction of their instructors. This is practically all that can be done during the next biennium, in view of the lack of funds for completing the plant and providing for metallurgical investigations. Instruction can be given in ore dressing and cyaniding, but until an adequate heating equipment is installed even this work must be curtailed during the winter. The net effect of the lack of funds will be to defer the efficient use of an admirable laboratory for two years.

Perhaps the larger function of the plant is to afford independent engineers a place to investigate methods and processes and test new inventions. Ordinary facilities for power and ore-handling machinery are at hand and there is ample space for the temporary construction and use of any device that might be needed. In this respect also the success of the plant must be deferred.

Elsewhere in this issue we note the entrance of the Canadian government into the ore-testing business. A com-

plete ore-dressing and metallurgical laboratory has been opened at Ottawa, where Canadian ores are to be tested free of charge. This laboratory also is to supply Canadian engineers with facilities for investigating problems in ore treatment, and arrangements can be made for the use of the laboratory by those who wish to undertake any research. The establishment of this laboratory is a departure in governmental affairs. Our own Bureau of Mines has taken no such step and probably will not. For a government to enter "business" is a dangerous step.

Steel Production Statistics

Official statistics of the production of steel ingots and castings in the United States have been published for 1912 by the Bureau of Statistics of the American Iron and Steel Institute. The output, in gross tons, has been as follows:

	1910.	1911.	1912.
Ingots,	25,154,087	23,029,479	30,284,682
Castings	940,832	646,627	966,621
Total	26,094,919	23,676,106	31,251,303

As 1911 was distinctly an "off year" in the iron and steel industry it may merely be remarked that there was an increase in steel output from 1911 to 1912 of 32 per cent. Of more interest is the comparison of 1912 with 1910, which was easily a record year, passing any previous year by 8 per cent. The gain from 1910 to 1912, however, was 19.8 per cent. Passing to more long-range comparisons, we observe that 1906 was a year of full output throughout, the steel mills being taxed to make deliveries, yet 1906 showed an output of only 23,398,136 tons, the gain of 1912 over that year being more than 33 per cent. Yet it is to be borne in mind that 1912 was not a year which seemed to bring out the full consumptive requirements of the country, nor did it bring out the full productive capacity of the country, though it came close to doing so. The idea that the steel industry has not grown much in the past half dozen years must be dismissed.

It is interesting to note that the production of steel ingots and casting in 1912 exceeded the production of pig iron. This was the case by a small margin in 1911, but in 1912 the margin was large, no less than 1,524,366 tons. There was, of course, a large tonnage of foundry grades of pig iron produced. The actual output of Bessemer and basic pig iron and of spiegelisen and ferromanganese in 1912 was 23,303,625 tons, and neglecting the tonnage of manganese bearing metals imported, there was an excess of 7,950,000 tons, or 34 per cent. in the steel ingots and castings produced over the blast furnace material consumed. This brings us to the point that the "production of steel ingots" as commonly reported is that, and nothing more. It is the production of ingots, not of commercial steel. It is scrap which makes up the difference, and more too, since there is an absolute loss of iron and of metalloids in the conversion into steel, but this scrap is largely self-produced, the material being produced in rolling, and eventually passing a second time over the ingot scales. There is, of course, a tonnage of old material used, but this is easily the smaller factor.

Recent gains in steel production have been confined entirely to the basic open-hearth process. Bessemer steel

has not attained its record made as far back as 1906, acid open-hearth has been practically stationary for a decade and crucible steel has recorded no clear gain, while the electric steel tonnage (18,309 tons in 1912) is too small as yet to be a commercial factor. The Bessemer output in 1912, 10,327,901 tons, fell short only of the output in 1905, 1906, and 1907, and the loss from the record year, 1906, to 1912, was only 15 per cent. Considering that meanwhile Bessemer steel has practically disappeared as material for structural shapes, has disappeared as plate material in the case of nearly all important work, and last year contributed only one-third to the total rail tonnage, it is evident that it is still much in requisition for other purposes, else it could not stand these losses without showing up quite poorly as to total tonnage.

"O Tempora O Mores"

The most modern philosopher, Henri Bergson, talks about intuitive consciousness and about class-consciousness. This philosophy so much in vogue can be turned in criticism against those who like it and the terms class and age "self-consciousness" applied. This age in its attitude of self-consciousness calls itself "the age of unrest," as if this was the only age in which the waters were always and everywhere in motion and as if society were heretofore in a static position and as if social kinetics were not actually the normal state.

But we would fain accept this description of title of "the age of unrest," for it shows so clearly the mental pose, not the mental poise of the troubled. Painting and sculpture have of late become markedly pornographic. Letters have assumed the picaresque. Social life has been colored with an unnative hue of extravagance of thought and action. Its paganistic tendencies are seen in the immense consumption of little white cylinders which emanate in a state of slow combustion, the meretricious incense of the Orient. The youthful female portion correspondingly seems to like to dress like Egyptian dancing girls. Puritanism has so waned that turkey-trotting is almost tolerated in Boston.

The condition exudes one-third sensualism, two-thirds egotism. Certainly, there are countercurrents strong and deep of balance, moderation, good taste, and intelligence. But the troubles bubble from the cauldron showing the start of some great refining action like in an open-hearth furnace when the pig and ore begin to "boil." Now the process from crude pig iron to refined steel is a turbulent one.

These troubles must be largely mental in their character, for people at no time were so well fed, so well clothed and so well protected against the elements. The inherent value of troubles to the world lies in the mental stimulus that they impart. But mental troubles are due in large part to the false and insecure base-line from which sociological conceptions are measured.

It will be found interesting to trace these phenomena on the analogy of quantitative molecular physics to their immediate causes. When a new force comes into any chemical reaction and plays no rôle except to accelerate the reaction-velocity, it is called a "catalyser." A catalyser can be sunlight, platinum black, iron oxide, an organic ferment or an alternating current of electricity. Things are moving

so fast that some catalyser must be working on society. Let us ask science what is this sociological catalyser. The answer comes from science itself, for the catalyser is science.

Science has reacted in two ways on human life. First, it has thrown up a new superstructure of facts of nature and so has profoundly affected theological thought or, better expressed, theological talk. Second, in its applied form it has created material wealth rapidly and made money-making very easy. In ordinary New Yorkese, "a big fat pie has been thrown before us and the angel-faced ones are scrambling for the best of it." To be concrete, the telephone, the telegraph, the motor, the modern high-speed tool steel, all the numerous products of the chemical and physical laboratory have made the efficient man vastly more efficient industrially speaking. He gets more, spends more, and the lesser ones envy more. Just this describes it, "getting and spending we waste our powers."

It has recurred to our mind that the advent of the flying machine is a typical case explaining well one phase of the phenomena. Nothing is so marked to-day as the disrespect in which the young hold the old. For a long time the older ones had been telling the younger ones that flying was the "dream of dreams," and even the wise elder said that it was impossible to make an aeroplane, since man weighed 150 lb. and the condor, largest of winged fowls, weighed only 70 lb. In parenthetical thought, let us acknowledge that their reasoning was perfect, but that the advent of the 1-hp-per-3-lb. gasoline motor was an unfore-seable fact.

Now, if you tell a child many times that things cannot ever happen and are wrong everytime, the respect of the child for you will surely approach zero as its limit. This has happened in the case of aeroplane development universally and has been the case in many, many other instances. It is hardly to be denied that we should expect disrespect from the young if we say that something never can be done and the children when they see that we are wrong get an enlarged, imaginatively stimulated egotism. "Most kids nowadays get the big head."

In the rich man's panic of 1903, the late Pierpont Morgan aptly described things as suffering from "undigested securities." His words can be applied in paraphrased form that there are undigested mental securities present in our life.

But, the call of the fates breaks loudly on our ears, "where are we at?" and "whither are we going?" We hear the plaintive minor key and then the strong major chord. History teaches that the world usually gets somehow what it really wants and needs. So much is clear. What the world needs now is men, not monkeys—men who can see, think, and act. History likewise teaches that the dumb, inarticulate cry of the many is heard by the few, who interpret it as a mother does the cry of her child, and by intelligence removes the cause. May we not divine that to the groans and rebellious sobs of the weltering masses will come the help of the true aristocrat, he who rules with love?

These mental securities of the present day seem paradoxical, they are so insecure. The latest science coming to a quantitative basis is psychology, which any scientist will call the greatest of sciences. When a science gets on an

exact basis, it can be quickly applied to practical uses, for "there is safety in numbers." Signs are not now wanting that there will be an application of psychology to man's needs. Of course, applied psychology will be first mixed with a large proportion of charlatanry, just as was the case with alchemy. Probably there will be an intensification of practical psychology in religion. Both these will be of value. A good many sensible people are of the opinion that the present day so-called Christianity contains too much that is sloppy and sentimental and that its strong leaven has been weakened in nineteen centuries. Would it also not be better if the preachers should "render unto Cæsar the things that are Cæsar's and to God the things that are God's," for indeed a strict law of psychology is that values of one kind can only be measured in a unit of that kind?

As Charles Proteus Steinmetz says, mathematics can be applied by analogical methods to all branches of human knowledge. In our daily mental life, the curve $y = \sin x$ or a mathematical expression of periodic phenomena applies as an analytical means to the solution of problems as surely as it does to music. The law is that things that start slowly to change, increase, reach a "temporary ultimate," slowly wane, then go on the descending curve. To apply usefully this equation of the law of compensation to the needs of man is one of the great problems of practical psychology.

The interpreters of modern thought must be master minds, for the world of men, women, and things is governed by intellectual Czars as much in these days of boisterous democracy as in olden times. In the intellectual world there is perfect freedom, tho' or since, as one views the question, its laws are inexorable. Bergson now dominates world philosophy, succeeding Spencer. So in the general catalysis produced by science, we see the equation $y = \sin x$, where the crest of the wave of materialism was reached some fifteen years ago and the imaginative and emotional spiritualism of science has started. Indeed the vogue that Christian Science has received is largely due to an instinctive popular revolt against deriving all action from materialistic causes, quite sufficient and proper in their own sphere but not of wide range. In the change among purely intellectual people, the rationalistic common-sense philosophy of William James, called "pragmatism," has been prepotent.

It seems reasonable to believe that things will change, not in one way but in another. At all events, one thing is sure, we have life and nature before us, with us, and of us. No one can take a survey of the country from a mountain top, forget himself, think of the universe and help but say, "in the twentieth century we do not exist, *we live*." If he forgets his weak, pitiful self, he will be of use to the world and become active and truly happy. As a great technical chemist says, "happiness is purely subjective."

The general catalytic effect of science on human thought and life may be likened to the incantations of the witches of Macbeth, "Toil and Trouble boil and bubble, Cauldron, boil and bubble." It is a firm deduction that science is the cause and that science in its widest sense will be the result. All the above is as plain as the nose on the face of a plain girl.

Readers' Views and Comments

"Electrochemical Spies in Niagara Falls"

To the Editor of Metallurgical and Chemical Engineering:

SIR:—On page 235 of the May issue of your journal is printed a note entitled "Electrochemical Spies in Niagara Falls." As this note is incorrect, allow the following facts to be known.

Mr. Kvernoe was not in the employ of any foreign concern at his stay here in this country. The statement that money was plentiful and freely used is also incorrect, as Mr. Kvernoe was dependent for his living on what he could earn by his work here. The statement that two accomplices of Mr. Kvernoe went about the country is a product of your informant's brisk imagination, to say no worse, and that a Swede and German were his accomplices is nothing but pure fabrication on the part of your informant.

What Mr. Kvernoe can be reproached for is that, when asking for work, he did not mention he had technical education. Mr. Kvernoe has now gone back to Norway which fact, it is to be hoped, will cause the return of your informant's equanimity.

AUG. STILLESEN,
Norwegian Vice Consul.

Niagara Falls, N. Y.

Cleaning Filter Cloths

To the Editor of Metallurgical and Chemical Engineering:

SIR: I observe in the May issue of METALLURGICAL AND CHEMICAL ENGINEERING editorial mention of methods of cleaning filter cloths, in which the use of steam under low pressure is said to have been introduced into the Black Hills by Mr. F. C. Bowman. This is in error, since I had made use of the same agent for cleaning the leaves of a clarifying filter of lime deposit during the winter early in 1912, at the same mill of which Mr. Bowman now has charge.

The idea did not originate with me, as I had seen steam used to dislodge lime accretion from the interior of solution pipes some years previous to my use of it on the clarifying leaves mentioned.

Regarding the use of steam on filters of the drum type, my experience has proved to me the futility of using it on the discharge port only; the interval of time in which the steam has to act is so short, before the leaves pass into the pulp again, that condensation takes place before the steam has time to act with much effect. A much better scheme would be to have the piping so arranged that—the vacuum being entirely off—the steam could be introduced into all the leaves simultaneously, thus obviating the counter effect of condensation. The time of treatment would also be much shortened.

There is no reason why steam could not be used on the ordinary form of leaf filter, especially where arranged in small sets or baskets. The steam need be used only a few minutes after the leaf is permeated, when water, slightly warmed, may be introduced and the softened lime is readily washed out.

The use of warm mill solutions would do much to prevent excessive deposits of lime in cold weather. W. H. WEIGAND.

Trojan, South Dakota.

New York Hydro-Electric Development and Niagara Falls

To the Editor of Metallurgical and Chemical Engineering:

SIR. Your issue of June contains two communications referring to my recently published book, "An Expensive Experiment," the points in which are of such general interest that I hope I may be permitted space for a reply.

Taking first the letter of Mr. Bennie, I would point out to him that the dilemma in which he has imaginatively placed me will be found upon examination to have no existence. The

old photograph, taken before the power development, is evidently one showing the extreme low water condition of 1895, and fully confirms the point made by the use of the official recent photograph in my book, which shows that the same results are brought about now by the diversion of water by the power companies at a time of not unusually low water flow. I regard the old picture, therefore, as confirmatory of my statement of the situation, and if I had been able to secure such a picture at the time of publication I would have used it for this purpose.

The criticism of Mr. F. A. Lidbury appears to be influenced by his appreciation of the value of the electrochemical industries which have been brought into being around Niagara Falls. I share with him, and have expressed in my book this appreciation of the utilization of hydro-electric energy in such industrial processes, but I do not believe that he or any other student of the subject will dispute my point that the partial utilization of hydro-electric energy in general electrical distribution is not economic. If the water of Niagara Falls is to be in whole or in part diverted from scenic utilization then it should be for the purpose of adding to the financial wealth of the community, and this is not to be done by dissipation in partial and intermittent use for light and general power purposes.

The reply made by Mr. Lidbury to the report of the Committee of Engineers of the United States Government, as to the indirect value of scenic Niagara, is not to the point. It is only necessary for us to imagine the result of the entire diversion of Niagara from its present course into power plants to realize the ruin that will fall upon a large portion of the surrounding communities and the losses which would spread far and wide therefrom.

I cannot, therefore, agree with your reviewer that this indirect source of business is a fallacy. It is satisfactory, however, to find that all of us are in agreement upon the main issue—the undesirability of the expenditure of State funds in hydro-electric developments, the commercial value of which is evidently a subject of doubt in a number of cases, and which, by the progressive improvements in other directions of power production, may eventually lose much of their practical value.

The State of New York is to be congratulated upon the firm attitude adopted by its Executive in disposing of the recent attempt to fasten upon it an ill-considered policy of this character.

REGINALD PELHAM BOLTON

New York City.

* * *

To the Editor of Metallurgical and Chemical Engineering:

SIR.—Referring to Mr. Bolton's communication about the photograph shown in your June issue, I do not in anywise agree with his notion that his dilemma vanishes upon examination. His difficulty is still as substantial as the rocks behind the falls themselves, notwithstanding the spray within which he seeks to conceal it.

It is not well to permit the actual point at issue to become befogged. Mr. Bolton's book contains a photographic reproduction of a portion of the falls, alleging "An Effect, in Part, of the Expensive Experiment." This I deny, and in support of my contention show another photograph "at least 20 years old," Mr. Bolton immediately jumps to the convenient and (for him) useful conclusion that my photograph is "evidently one showing the extreme low water condition of 1895" and proceeds to reach some "therefores," which unfortunately are still at variance with the facts, and "therefore" unjustified.

My photograph is not that of the extreme low water condition of 1895. It must be obvious to your readers that 1913, minus "at least 20" does not result in 1895. Of the two gentlemen concerned in the taking of the original photograph, but one sur-

vives, and he assures me that it is even older than 20 years; how much older, I need not disclose to prove my point.

Mr. Bolton's claim that he would have used such a photograph had he been able to obtain a copy seems rather disingenuous in view of the fact that it is really but a portion of one of the standard photographs of Niagara Falls, taken as typical of its beauty. Most visitors to Niagara Falls find it difficult to avoid carrying a copy home with them, due to the importunate blandishments of numerous souvenir bazaar keepers. I assure him he has my permission to obtain all the copies he wants, but he should be careful how he uses them.

With the fog thus dissipated by this discharge of fact, his dilemma clearly reappears.

Niagara Falls, N. Y.

P. MCN. BENNIE.

* * *

To the Editor of Metallurgical and Chemical Engineering—

Sir:—Mr. Bolton's reply to my criticism of his letter to the *Electrical World* requires but little comment.

It is as easy to attempt to dispose of an argument by stating that it "is not to the point" as it is to attempt to avoid a dilemma by calling it an imaginary one—and about as convincing.

Unaccountably enough, "the surrounding communities" do not seem to "realize the ruin" which has fallen upon them as a result of the present diversion of the Niagara river, nor is there any evidence that a considerable extension of the same kind of "ruin" and "losses" would be otherwise than welcome!

It would be difficult to gather from Mr. Bolton's book, and impossible to gather from Mr. Bolton's letter to the *Electrical World*, that "appreciation" of the economic value of hydro-electric power as applied to the electrochemical industries to which he now gives somewhat belated expression. It is to be hoped that, when next the question of utilization of Niagara Falls engages his pen, he will endeavor to show that he has at least a general acquaintance with the facts, and a desire, in drawing conclusions, to adopt the dispassionate and unprejudiced attitude which is expected of an engineer.

Niagara Falls, N. Y.

F. AUSTIN LIBBURY.

The Western Metallurgical Field

Effect of Spelter Prices on Production

The drop in the spelter market from approximately 7 to 5 cents per pound for metal has had a decided effect on production in the Western zinc-producing districts. Ignoring for the moment the causes contributing to the low market price, the fact remains that many of the smaller producers and some of the larger ones find themselves either unable or unwilling to mine and concentrate zinc ore under prevailing conditions. In Colorado zinc production has practically ceased at Leadville, except for some production of carbonate ore to fulfill contracts; at Rico the output has fallen off decidedly, and at Creede the production is almost without profit. The Wellington, at Breckenridge, has curtailed production to about one-third of the normal.

The condition in other states is not unlike that in Colorado. The Yellow Pine in Nevada has ceased zinc production, as has the Federal at Wallace, Idaho. Butte & Superior, at Butte, Mont. one of the largest western zinc producers, is contemplating closing its mill until market conditions improve. At Joplin, Mo., the American Zinc, Lead and Smelting Co. has closed three of its large mills; and other companies operating in sheet-ground, where the ore is low grade and difficult of extraction, have discontinued operations.

Popular opinion generally ascribes the spelter situation to prospective tariff legislation. It is doubtful, however, if this assumption is correct, for when we consider the enormous increase in production of spelter during the past two years, and the rate at which it was bought by consumers, the impression will gain ground that buyers became oversupplied and ultimately ceased buying, with the inevitable result of depressing the price for spelter. The market is now below the average, and this has had the effect of curtailing production which will continue until more nearly normal conditions obtain.

Oregon-California Mining Congress

The recent session of the Southern Oregon and Northern California Mining Congress, at Redding, Cal., was one of the most interesting and profitable meetings of that organization. Considerable attention was devoted to the subject of smelter fumes, and the effect of recent litigation on the smelting industry of California. One of the first acts taken was to appoint a committee to visit the agricultural districts surrounding the smelters, and make a personal inspection of conditions in the field and observe alleged damage from smelter fume. The report of the committee was to the effect that a very careful and exhaustive examination showed all agricultural crops in good condition; that while smelter fume had in the past been responsible for some damage, nevertheless it seemed improbable that any such damage was now being done by any fumes escaping from the Mammoth plant at Kennet. The report further deprecated the conflict waged against smelting by the farmers, and expressed the opinion that the present inimical attitude of the latter was unwarranted.

The warfare of California farmers against the smelters reached its height some time ago when the agriculturists succeeded in getting a grand jury to declare the Mammoth smelter a public nuisance. By this means they hoped to unload the burden of further expensive prosecution, and place it on the authorities of Shasta county. The latter, however, have wisely concluded not to take any steps unless the farmers supply them with adequate evidence on which to base a suit, and the conspiracy is likely to fail.

It has already been suggested in this journal that the best solution of the smelter fume question is to refer complaints to a board of arbitration or some commission which shall control the dispute between farmers and smelters, determine the conditions under which smelters may operate, and make awards for damage if any is sustained. Such an arrangement already exists between the United States government and the Anaconda Copper Mining Co., and a similar scheme is under way for the settlement of the controversy between the Selby smelter in California and the inhabitants of Solano county. There is no reasonable excuse why such an equitable arrangement should not be concluded between the farmers of northern California and the smelters in that part of the state, for the present attitude of the agriculturists is intolerable and seriously interferes with legitimate smelting operations.

Canadian Government Testing Plant

According to a bulletin recently issued by the Department of Mines, Canada, the government has established a well-equipped ore dressing and metallurgical laboratory at Ottawa, under the direction of Dr. Eugene Haanel. The laboratory is established for the purpose of experimenting on the concentration and metallurgical treatment of Canadian ores, and is equipped with apparatus in both commercial and laboratory sizes.

The standard size machinery offers facilities for crushing and screening, sampling and recording, amalgamation and concentration. The crushing machines are one each, 12-in. by 8-in. Blake crusher, 24-in. by 14-in. rolls, and 4-ft. 6-in. Hardinge conical ball mill. For screening a 6-ft. Ferraris screen is used for coarse sizing, a Keedy sizer for fine sizing and a duplex Callow screen for concentration work. Sampling is provided for by two standard Vezin machines placed in favorable position to cut out preliminary samples of coarse material. An 8-unit system of Flood automatic samplers is provided for fine material. Hand sampling can be accurately done by means of Jones riffles. All water lines are equipped with Keystone water meters to keep accurate records of water consumption.

For amalgamation and concentration the equipment includes a 5-stamp battery with 1250-lb. stamps, a 10-ft. amalgamating table and Pierce amalgamator. The mortar of the stamp mill may be arranged for inside amalgamation. Six 8-ft. Callow tanks are available for desliming and settling; two Richards pulsator classifiers of the launder type, one Richards pulsator two-compartment jig and one Overstrom sand table and Deister

slime table for concentration. Magnetic and electrostatic separation can be accomplished on several types of machines: a tandem unit Grondal magnetic separator for wet separation of strongly magnetic minerals; a Grondal magnetic cobber for dry separation of strongly magnetic minerals; an Ullrich four-pole magnetic separator for either wet or dry separation of weakly magnetic minerals; and a Huff electrostatic unit, consisting of a standard generator and two laboratory-size separators.

The laboratory units consist of Sturtevant crusher, rolls and screen; Braun pulverizer; Abbe 6-jar pebble mill; gyratory screen for grading analyses; Richards combined laboratory pulsator jig and classifier, with glass side; Grondal separator for either wet or dry work; Wilfley table; cyanide plant of 200-lb. capacity, consisting of a Parral agitator and air pump, with necessary tanks; two filter presses, and complete sets of I. M. M. and Tyler standard screens for grading purposes.

It is expected that the installation of a roasting and sintering plant will be undertaken this year. The entire plant will be operated free of charge on Canadian ores, under the following conditions: (1) Samples must be sacked and delivered to the plant free of all transportation charges. (2) Not less than 200 lb. will be accepted for small-scale tests, nor less than 5 tons for large-scale tests. (3) All testing products become the property of the Department of Mines, unless stipulation to the contrary is made before the test begins. (4) Reports of tests will be incorporated in the publications of the Mines Branch, but single copies will be given to the owner of the ore tested. Ordinarily the tests will be made by officials of the department, but arrangements can be made for competent engineers to supervise their own tests. It is expected that the plant will be ready for operation early in the month of July, 1913. All communications regarding its use should be addressed to Dr. Eugene Haanel, Director Mines Branch, Department of Mines, Ottawa, Canada.

Concentration at Cripple Creek

It is well known that the fine material resulting from crushing many of the ores of the Cripple Creek district is comparatively richer than the average ore. This is due to the fact that the planes of fracture in crushing are coincident with fine veins of high-grade mineral which become more or less pulverized in the operation. Various schemes have been adopted in the Cripple Creek district to save this portion of fine, rich mineral. Dry screening, or washing, or combination of the two, have been used. The Golden Cycle company uses dry screening, and the Elkton company has a wet washing plant. Recently the Vindicator company has decided to erect a concentrating plant, and tests are now in progress to determine the best method of procedure. It is probable that both dry and wet processes will be used in this plant, and the product will be shipped to smelters. Considerable of the fine mineral can be recovered by dry screening, and this may be followed by some form of wet dressing to recover the portion that requires attrition to be removed from the rock faces.

Company Reports

The Anaconda Copper Mining Co. presents some interesting facts regarding its operations in the annual report for the year 1912. The year was one of the best in the history of the company, and resulted in the mining and treatment of a larger tonnage of ore than for several years previous. With the marked improvement in the price for copper, the company entered into an agreement with the Butte Miners' Union to provide for an automatic variation in the wage scale to correspond with the fluctuations in the price for copper. The terms of the agreement provide: that when the monthly price for electrolytic copper is 15 cents and over, but under 17 cents, the wages of all men employed underground shall be increased 25 cents above the minimum wage of \$3.50 per day; and that when the price of copper is 17 cents and over, the wages for underground men shall be increased an additional 25 cents per day. The agreement is such that the contract is automatic in its operation, and that corresponding reductions in the in-

creased wage will take place when the price of copper falls below the figures named. Similar contracts were entered into with different classes of labor other than the above. All contracts expire on June 1, 1915.

The reduction works treated during the year 3,880,203 tons of ore and cupriferous material at Anaconda, and 1,189,039 tons at Great Falls. Of this, 4,486,873 tons came from the company's mines, 581,032 tons were purchased, and 1338 tons comprised precipitates and cleanings from the Old Reduction Works. The production of metals was as follows:

	Lb. Copper.	Oz. Gold.	Oz. Silver.
Anaconda	222,763,670	52,564	9,702,605
Great Falls	71,710,491	61,314	11,014,737

The report chronicles the attempt to develop a successful centrifugal concentrator for slime, experiments with which began in 1910. Several machines have been built in an effort to overcome mechanical difficulties, and one machine ran throughout the year 1912. While the results indicate that for certain classes of material the operation of the machine is satisfactory, there is still a doubt whether for everyday practice some one of the other machines tested might not be of equal service.

One of the sections of the Washoe concentrator was remodeled to conform to certain details of practice which had been evolved at Great Falls, and the results based on a test for three months proved it to be an absolute success.

Various experimental processes have been tested during the year, with varying success. These processes had in view improvements in concentration and the reduction of slimes and tailings.

The process for slimes promises well and will be tested on a large scale. Should it prove successful, a complete installation will be made, which will mean not only a reduction in costs, but a marked increase in output of refined copper from the same tonnage of original ore treated. The process has been perfected by employees of the company, and will be useful in treating the accumulation of many millions of tons of tailings which are available at the Washoe plant.

The Great Falls smelter will be rebuilt during 1913. The old blast furnace department, flues and stack, together with electrolytic refinery, will be retained, but the balance of the plant will be rebuilt. The greatest improvement made during the year was in the converting department, where a radical change was made in the size of converters. The "Great Falls" type of cylindrical converter has been built as large as 20 ft. in diameter, and the new smelter will be equipped with converters of this type.

The Non-Ferrous Metal Market

Prices for the non-ferrous metals during June have been quite unsettled, with a tendency to a steadily lowering market. But little business has been done, and the lack of large prospective orders probably has kept prices from going still lower on competitive bidding. Unsettled financial conditions have been reflected in metal prices, and consumers have been unwilling to buy heavily under the conditions.

Copper.—This market has been showing little life, and the liquidation of speculative holdings has tended to lower prices. The latest quotations are 14¼ to 15 cents for Lake, and 14.70 to 14.80 cents for electrolytic.

Tin.—Prices have declined steadily since our last report, and the market has been weak. Liquidation has continued, and has resulted in the quotation of spot tin below three months—something that has not occurred for some months. The last quotation is about 45¼ cents for June tin, New York.

Lead.—The market has fallen off somewhat and for the past two weeks has been unchanged at 4.17½ to 4.20 cents, St. Louis, and 4.30 to 4.35 cents, New York.

Spelter.—Despite the lower prices business in this metal has been smaller in volume than for some time. Production from some Joplin and Western mines has ceased, as the market affords no profit. The last quotations are 4.85 to 4.95 cents, St. Louis, and 5 to 5.10 cents, New York.

Other Metals.—The market for aluminum has been quiet, and little business has been transacted. Prices are 25½ to 26 cents for spot and 26 to 26½ cents per lb. for future delivery. Prices for antimony vary from 7.50 to 9 cents per lb. for various brands. Business in quicksilver is fair and prices are steady at \$40 per flask of 75 lb., New York, and \$39.50 at San Francisco.

Iron and Steel Market

June has been a somewhat quieter month than May in the finished steel trade, but production has been continued at substantially the maximum rate, the mills running chiefly upon old orders. The large amount of business which had been entered upon books has stood the test of a dull market very well, there being an almost negligible amount of cancellations and postponements.

Strictly new buying in June probably did not equal 50 per cent of the shipments, but the rate of buying was no smaller than in May. Specifications against old contracts averaged 60 or 65 per cent in June, against 70 or 75 per cent in May, showing a decrease which was to be expected as midsummer approached.

The present position is that the large mills are booked up almost fully, with perfectly trustworthy business, until nearly the close of the year, in rails, plates, structural shapes and merchant steel bars. In tin plates they have actual specifications for from 6 to 12 weeks, and in sheets and tubular goods for from 4 to 10 weeks. In wire products there is practically no business actually specified on books and specifications are coming in slowly on contracts.

The smaller mills, comprising perhaps 20 per cent of the total capacity, have relatively little business on books. Normally they do not book up extensively, endeavoring rather to obtain premiums for prompt shipment, and in this they succeeded well in the early months of the year. Now the premium business has disappeared and it is even reported that some mills in the east are shading structural shapes and plates by \$1 a ton.

With so large an amount of business already on books, it would require only a relatively small buying movement to fill the mills completely for the balance of the year. Such a buying movement is predicted and by some as early as next August. It is represented that by that time good crops will be assured and the tariff bill will probably be out of the way. So far as regards this year's business, the iron and steel industry objects to the tariff revision chiefly on account of the sentimental disturbance given to general business, though for the general future it is held that some of the readjustments required by the prospective tariff will cause trouble.

Steel prices have remained firm on the whole. Premiums for prompt delivery, of course, disappeared some time ago, but the regular mill prices have declined in only two instances from the top point reached in last year's rise, these exceptions being sheets and wire products. The latter are easily shaded \$1 or \$2 a ton, and a general revision in wire prices is likely to occur shortly, such revisions being common about July 1. Several months ago galvanized sheets declined \$2 a ton, and later black sheets declined \$1, while another decline of \$1 has occurred in the past month in black sheets. Manufactured goods, such as shafting, rivets, spikes, etc., have experienced considerable declines in the past few months.

Pig Iron

Early in June distinct hopes were expressed that the decline in pig iron was over and that more active buying would ensue, with a consequent rise in prices. These hopes have been dispelled. Pig iron has declined throughout the country an average of about 50 cents during June, against a decline of about \$2.25 during the preceding five months. There has been slightly heavier buying, but instead of this buying supporting the market it has tended if anything to weaken the situation, through buyers taking relatively small tonnages and then withdrawing, so that the market loses the prospect of their buying in a liberal

way in the near future. The decline throughout has been one actuated largely by sentiment, buyers refusing to make forward purchases as long as they observed a declining tendency, and at no time has the market had sufficient vitality to invite a general buying movement. Stocks have accumulated in a small way in some districts, but in general they are light. Production of merchant iron has been curtailed slightly in the past two or three months, but a greater curtailment is in prospect during the next two months. The market is now quotable as follows: Southern No. 2 foundry at Birmingham, \$10.75; No. 2 X foundry, Philadelphia, \$16; Buffalo, \$14; No. 2 foundry, Cleveland, \$14.75; Chicago (at furnace), \$15.50; at Valley furnaces (90 cents higher delivered Pittsburgh), Bessemer, \$16; basic, \$14.50; foundry, \$14; forge, \$13.75; malleable, \$14.25.

Steel

The scarcity of unfinished steel has continued and efforts by consumers to break the market have been largely unavailing. There is no demand to test the market for prompt sheet bars, but prompt billets bring premiums of \$1 to \$2 a ton or more, depending on size and analysis. The regular forward market is quotable at \$26.50 to \$27 for billets and \$27 to \$27.50 for sheet bars, f.o.b. maker's mill, Pittsburgh or Youngstown. Rods have declined \$1 a ton to \$29, Pittsburgh.

Finished Steel

Regular prices are as follows, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Common iron bars, 1.65 cents, Pittsburgh; 1.57½ cents, Philadelphia; 1.50 cents, Chicago.

Wire nails, \$1.75 per keg, base, nominally, \$1.80; plain wire, 1.60 cents.

Sheets, blue annealed, 10 gage, 1.75 cents; black, 28 gage, 2.25 to 2.35 cents; galvanized, 28 gage, 3.40 to 3.50 cents; painted corrugated, 28 gage, 2.45 to 2.55 cents; galvanized corrugated, 28 gage, 3.45 to 3.55 cents.

Tin plates, \$3.60 for 100-lb. cokes.

Merchant steel pipe, 79 per cent off list for 3¼ to 3 in.

Steel boiler tubes, 3½ to 4½ in., 69 per cent off list.

Standard railroad spikes, 1.80 cents, Pittsburgh or Chicago.

Button head structural rivets, 2 to 2.10 cents; cone head boiler rivets, 2.10 to 2.20 cents.

Application of Ozone to Water Purification

The New York State Department of Health has just published a "Report on the Application of Ozone to Water Purification," by Russell Spaulding, Consulting Ozone Engineer, State Department of Health.

The report comprises 45 large pages and contains profusely illustrated notes on some of the European plants using ozone for the sterilization of drinking water as follows:

The plant at Saint Maur supplying the City of Paris daily with 24,300,000 gallons of sterilized water; bacteriological tests and ozone determinations in 1907 at Saint Maur plant; results of bacteriological examinations made on filtered water sterilized by ozone at the municipal plant of the City of Paris at Saint Maur on December 10, 1909; the ozone plant at Villefranche; mineral analysis and composition of the filtered water of the River Vesubie; the ozone plant at Nice; the ozone plant at Paderborn, Germany; the ozone plant at Saint Petersburg, Russia.

The conclusions of the report are as follows:

"The records of the foregoing ozone purification plants, and their successful operation over an extended period of time, have established beyond cavil that ozone as a purifying agent is both efficient and economical, so that an extended discussion on the subject seems superfluous.

"It may be well, however, to emphasize the fact that in con-

tradistinction to all other purifying agents, ozone possesses one signal virtue in that it is absolutely non-toxic. Furthermore, it is insoluble in water to any appreciable extent, so that 'over-dosing' is impossible.

"Aside from its wonderful and unfailing bactericidal power, ozone totally removes from water all traces of odor, color and taste due to the decomposition of organic matter. Ozonized water is never flat to the taste—it is not only perfectly hygienic, but satisfies the most delicate palate.

"To summarize, ozone destroys in water all that which is harmful or distasteful and in no manner changes the mineral qualities.

"The successful application of ozone to water purification is dependent on a number of factors, none of which may be disregarded with impunity.

"The failures that have occurred have been due solely to faulty design in the electrical or mechanical devices used.

"Ozone, as such, never has and never can fail to perform its functions properly.

"The first problem that confronted the pioneers in this art was the *economical production of ozone*.

"Ozone can be produced chemically, thermally and electrically. The first two methods were soon discarded in favor of electrical methods, which, however crude at the start, offered the best promise of ultimate success.

Energy Expended in Producing Ozone

"A molecule of ozone (O_3) weighs $16 \times 3 = 48$ grams.

"Its production requires a heat expenditure of 29,600 calories corresponding to $0.4235 \times 29,600 = 12,535$ kilogrammeters.

"To obtain 1 gram of ozone there is required an expenditure of energy of

$$0.4235 \times 29,600 \div 48 = 261 \text{ kilogrammeters.}$$

"Therefore, the theoretical value of 1 horsepower hour* (270,000 kilogrammeters) expressed in grams of ozone is

$$270,000 \div 261 = 1,034 \text{ grams}$$

"And the theoretical value of 1 kilowatt hour of electrical energy expressed in grams of ozone is

$$1,034 \times 1,000 \div 746 = 1,386 \text{ grams}$$

"The *efficiency of production of ozone* is even to-day quite remote from the theoretical efficiency. The early efforts produced only 3 grams of ozone per kilowatt hour of energy. The present specific output, prevalent in the European plants described in this report, varies from 40 grams to 60 grams of ozone per kilowatt-hour of electrical energy.

"In some instances much higher efficiencies have been obtained by supplementary devices, such as refrigerators and dehydration of air to be ozonized, but these devices are not only expensive and cumbersome but they tend to complicate the apparatus as a whole and diminish its dependability.

"Efficiency in ozone production has increased with the efficiency in electrical apparatus such as dynamos and transformers, but little progress has been made in increasing the efficiency of the ozone apparatus itself.

"Aside from the generators required to supply alternating current and the transformers used to step-up the potential to its required power, the ozone apparatus consists of electrodes of metal separated by an air space, dielectrics, such as glass, being interpolated between electrodes to produce a brush-discharge. The brush-discharge acting on the air disturbs the atomic balance of the oxygen, which is normally di-atomic, in such a manner that more or less of the atoms recombine in an unstable tri-atomic form which is called ozone.

"This brush-discharge is accompanied by the phenomena of light, heat, and noise, all of which absorb energy, thereby reducing the percentage of energy available for the creation of ozone.

"The heat generated tends to cause a reversion of ozone into oxygen and furthermore lessens the dielectric resistance of the glass dielectrics, making them liable to puncture by the high-tension current—thus putting the apparatus out of commission.

"To overcome this difficulty various means have been re-

sorted to, to prevent a rise in temperature, such as refrigerating the air before it reaches the electrodes of discharge, or surrounding the electrodes with some fluid such as oil or water, for the purpose of carrying off the heat as fast as generated. These palliatives, though they make the operation of ozone generators possible, only add to the losses already entailed in the generation of the objectionable heat.

"The obvious remedy for low efficiency is, therefore, to so arrange the electrical circuit that the major portion of electrical energy supplied to an ozone generator shall be used in the conversion of oxygen into ozone, and that only a small portion shall be lost in a non-productive manner.

"The non-production of heat should be sought rather than its elimination by cooling means.

"To summarize then, the main desiderata in generating ozone are: 1, a supply of alternating electric current at low cost; 2, an efficient transformer to obtain high tension; 3, ozone electrodes that do not generate enough heat to (a) disrupt the dielectrics, (b) cause reversion of ozone to oxygen, (c) require external means for cooling; 4, ozone electrodes that will approach the theoretical efficiency much more closely than the various systems now in use.

"Such an ozone generator does exist and when offered to the public will revolutionize the art. Pending such time it would be improper to give a full description of it.

"The *efficiency in the application of ozone* to the purification of water is dependent entirely upon mechanical factors.

"Bacteria are immediately destroyed and converted into CO_2 when brought into contact with ozone. It becomes, therefore, necessary to insure this contact beyond possible failure. To do this the water must be broken up into very fine particles and this must occur in the presence of and in contact with ozone or ozonized air at a proper concentration.

"The only failures in ozone water-purification plants that have so far occurred have been due to a failure in breaking up the water in sufficiently fine particles and in properly contacting these small particles with ozone. I speak of failures in the technical sense, that is, failure to produce odorless, tasteless, colorless water, entirely free from pathogenic germs, by means of ozone.

"Failures might occur from a commercial point of view, because of excessive overhead charges or too high a cost of maintenance due to low efficiency or unreliability of apparatus.

"Two much-advertised plants on the American continent—one at Lindsay, Ontario, and the other at Ann Arbor, Mich.—have been failures. These failures are due only to faulty engineering in installation; there is an ample supply of ozone, but the water is not properly broken up and contacted with the ozone."

The author finally feels "justified in urging upon the authorities of the state of New York the advisability of giving to ozone, as a means for purifying potable waters, their most earnest consideration.

"There is certainly no means known to science that is more reliable or absolute than ozone with which to overcome pollution and protect the public health.

"The latest developments in the art have brought ozone water-purification well within the boundaries of economical municipal administration."

Water Power Decision of the U. S. Supreme Court

A decision of immense and serious importance to water-power interests has just been handed down by the U. S. Supreme Court in the case of the United States, plaintiff in error, versus the Chandler-Dunbar Water-Power Company, et al. In view of the effect which this decision may have in discouraging in future the investment of large sums of money in the development of the water-powers of streams which now are non-navigable, but the water of which may be declared by Congress to be necessary for navigation through a system of canals some day in the future, we give a long abstract of the decision. The decision is printed as Senate Document No. 51.

*The report reads here horsepower instead of horsepower hour. As this is an evident mistake, it has been corrected here and in the following lines.—Editor.

By Act of Congress of March 3, 1909, the War Department was directed to take over all lands and property north of the present St. Mary's falls ship canal throughout its entire length and lying between the said ship canal and the international boundary line at Sault Ste. Marie, the act declaring that the ownership of such lands and property by the United States "is necessary for the purposes of navigation of said waters and the waters connected therewith." Provision was also made in the act for condemnation proceedings to determine compensation for the property taken, and for revocation of all permits to water-power companies on the lands specified.

Condemnation proceedings were accordingly instituted and the compensation finally awarded was as follows:

(a) To the Chandler-Dunbar Company, \$652,332. Of this \$550,000 was the estimated value of the water-power.

(b) To the St. Mary's Falls Power Company, \$21,000.

(c) To the Edison-Sault Electric Company, \$300,000, which has, however, been settled by stipulation.

(d) To the Michigan Lake Superior Power Company, nothing.

From these awards the Government, the Chandler-Dunbar Company, the St. Mary's Falls Power Company, and the Michigan Lake Superior Power Company had sued out writs of error.

The errors assigned by the United States challenged the allowance of any compensation whatever on account of any water-power right claimed by any of the owners of the condemned upland, and also the principles adopted by the District Court for the valuation of the upland taken. On the other hand, the several corporations who sued out writs of error, complained of the inadequacy of the award on account of water power claimed to have been taken, and also of the valuation placed upon the several parcels of upland condemned.

The opinion of the Supreme Court, written by Justice Lurton, then reads as follows:

"It will be seen that the controlling questions are, first, whether the Chandler-Dunbar Company has any private property in the water-power capacity of the rapids and falls of the St. Mary's River which has been 'taken' and for which compensation must be made under the fifth amendment to the Constitution; and, second, if so, what is the extent of its water-power right and how shall the compensation be measured?

"That compensation must be made for the upland taken is not disputable. The measure of compensation may in a degree turn upon the relation of that species of property to the alleged water-power rights claimed by the Chandler-Dunbar Company. We, therefore, pass for the present the errors assigned which concern the awards made for such upland.

"The technical title of the Chandler-Dunbar Company includes the bed of the river opposite its upland on the bank to the middle thread of the stream, being the boundary line at that point between the United States and the Dominion of Canada. Over this bed flows about two-thirds of the volume of water constituting the falls and rapids of the St. Mary's River. By reason of that fact and the ownership of the shore the company's claim is that it is the owner of the river and of the inherent power in the falls and rapids, subject only to the public right of navigation.

"While not denying that this right of navigation is the dominating right, yet the claim is that the United States in the exercise of the power to regulate commerce may not exclude the rights of riparian owners to construct in the river and upon their own submerged lands such appliances as are necessary to control and use the current for commercial purposes, provided only that such structures do not impede or hinder navigation and that the flow of the stream is not so diminished as to leave less than every possible requirement of navigation, present and future.

"This claim of a proprietary right in the bed of the river and in the flow of the stream over that bed to the extent that such flow is in excess of the wants of navigation constitutes the ground upon which the company asserts that a necessary effect

of the act of March 3, 1909, and of the judgment of condemnation in the court below, is a taking from it of a property right or interest of great value, for which, under the fifth amendment, compensation must be made.

"This title of the owner of fast land upon the shore of a navigable river to the bed of the river is at best a qualified one. It is a title which inheres in the ownership of the shore, and, unless reserved or excluded by implication, passed with it as a shadow follows a substance, although capable of distinct ownership. It is subordinate to the public right of navigation, and however helpful in protecting the owner against the acts of third parties, is of no avail against the exercise of the great and absolute power of Congress over the improvements of navigable waters. That power of use and control comes from the power to regulate commerce between the States and with foreign nations. It includes navigation, and subjects every navigable river to the control of Congress.

"All means having some positive relation to the end in view which are not forbidden by some other provision of the Constitution are admissible. If, on the judgment of Congress, the use of the bottom of the river is proper for the purpose of placing therein structures in aid of navigation, it is not thereby taking private property for a public use, for the owner's title was in its very nature subject to that use in the interest of public navigation. If its judgment be that structures placed in the river and upon such submerged land are an obstruction or hindrance to the proper use of the river for purposes of navigation, it may require their removal and forbid the use of the bed of the river by the owner in any way which in its judgment is injurious to the dominant right of navigation. So, also, it may permit the construction and maintenance of tunnels under or bridges over the river, and may require the removal of every such structure placed there with or without its license, the element of contact out of the way, which it shall require to be removed or altered as an obstruction to navigation."

Various former opinions are quoted and the Supreme Court's decision continues as follows:

"The conclusion to be drawn is, that the question of whether the proper regulation of navigation of this river at the place in question required that no construction of any kind should be placed or continued in the river by riparian owners, and whether the whole flow of the stream should be conserved for the use and safety of navigation are questions legislative in character; and when Congress determined, as it did by the act of March 3, 1909, that the whole river between the American bank and the International line, as well as all of the upland north of the present ship canal, throughout its entire length, was 'necessary for the purposes of navigation of said waters and the waters connected therewith,' that determination was conclusive.

"So much of the zone covered by this declaration as consisted of fast land upon the banks of the river, or in islands which were private property, is, of course, to be paid for. But the flow of the stream was in no sense private property, and there is no room for a judicial review of the judgment of Congress that the flow of the river is not in excess of any possible need of navigation, or for a determination that if in excess the riparian owners had any private property right in such which must be paid for if they have been excluded from the use of the same.

"That Congress did not act arbitrarily in determining that 'for the purposes of navigation of said waters and the waters connected therewith,' the whole flow of the stream should be devoted exclusively to that end, is most evident when we consider the character of this stream and its relation to the whole problem of lake navigation."

The decision thus gives a review of the situation of lake commerce and of the growth of canal traffic at the Soo. "Millions of public money have already been expended, in the construction of canals and locks, by this Government upon the American side, and by the Canadian Government upon its own side of the rapids, as a means by which water craft may pass

around the falls and rapids in the river. The commerce using these facilities has increased by leaps and bounds."

"The upland belonging to the Chandler-Dunbar Company consists of a strip of land some 2,500 feet long and from 50 to 150 feet wide. It borders upon the river on one side, and on the Government canal strip on the other. Under permits from the Secretary of War, revocable at will, it placed in the rapids, in connection with its upland facilities, the necessary dams, dikes, and forebays for the purpose of controlling the current and using its power for commercial purposes, and has been for some years engaged in using and selling water power. What it did was by the revocable permission of the Secretary of War, and every such permit or license was revoked by the act of 1909."

"That it did not thereby require any right to maintain these constructions in the river longer than the Government should continue the license needs no argument. They were placed in the river under a permit which the company knew was likely to be revoked at any time. There is nothing in the fact which favors the estopped in law or equity. The suggestion by counsel that the act of 1909 contemplates that the owner should be compensated not only for its tangible property, movable or real, but for its loss and damage by the discontinuance of the company's license and its exclusion from the right to use the water power inherent in the falls and rapids, for commercial purposes, is without merit.

"The provisions of the act in respect of compensation apply only to compensation for such 'property described' as shall be held private property taken for public uses. Unless, therefore, the water-power rights asserted by the Chandler-Dunbar Company are determined to be private property the court below was not authorized to award compensation for such rights.

"It is a little difficult to understand the basis for the claim that in appropriating the upland bordering upon this stretch of water, the Government not only takes the land but also the great water power which potentially exists in the river. The broad claim that the water power of the stream is appurtenant to the bank owned by it, and not dependent upon ownership of the soil over which the river flows has been advanced. But whether this private right to the use of the flow of the water and flow of the stream be based upon the qualified title which the company had to the bed of the river over which it flows or the ownership of land bordering upon the river, is of no prime importance. In neither event can there be said to arise any ownership of the river. Ownership of a private stream wholly upon the lands of an individual is conceivable; but that the running water in a great navigable stream is capable of private ownership is inconceivable.

"Whatever substantial private property rights exist in the flow of the stream must come from some right which that company has to construct and maintain such works in the river, such as dams, walls, dikes, etc., essential to the utilization of the power of the stream for commercial purposes."

"That riparian owners upon whom public navigable rivers have in addition to the rights common to the public certain rights to the use and enjoyment of the stream, which are incident to such ownership of the bank, must be conceded. These additional rights are not dependent upon title to the soil over which the river flows, but are incident to ownership upon the bank. Among these rights of use and enjoyment is the right, as against other riparian owners, to have the stream come to them substantially in its natural state, both in quantity and quality. They have also the right of access to deep water, and when not forbidden by public law may construct for this purpose wharves, docks, and piers in the shallow water of the shore. But every such structure in the water of a navigable river is subordinate to the navigation, and subject to the obligation to suffer the consequences of the improvement of navigation, and must be removed if Congress in the assertion of its power over navigation shall determine that their continuance is detrimental to the public interest in the navigation of the river."

"It is said that the 12th section of the act of 1909 authorizes the Secretary of War to lease, upon terms agreed upon, any

excess of water power which results from the conservation of the flow of the river and the works which the Government may construct. This, it is said, is a taking of private property for commercial uses and not for the improvement of navigation. But, aside from the exclusive public purpose declared by the eleventh section of the act, the twelfth section declares that the conservation of the flow of the river is 'primarily for the benefit of navigation and incidentally for the purpose of having the water-power developed, either for the direct use of the United States or by lease . . . through the Secretary of War.'

"If the primary purpose is legitimate, we can see no sound objection to leasing any excess of power over the needs of the Government."

"The conclusion, therefore, is that the court below erred in awarding \$550,000, or any other sum, for the value of what is called 'raw water,' that is, the present money value of the rapids and falls to the Chandler-Dunbar Company as riparian owners of the shore and appurtenant submerged land."

The decision then takes up the award for the upland taken.

The awards to the Chandler-Dunbar Company includes certain sums for special values: The value of the upland strip fixed at \$65,000 was arrived at in this manner:

(a) For its value, including railroad side tracks, buildings and cable terminals, including also its use, "wholly disconnected with power development or public improvement, that is to say, for all general purposes, like residences, or hotels, factory sites, disconnected with water power, etc., \$20,000."

(b) For use as factory site in connection with the development of 6500 hp, either as a single site or for several factories to use the surplus of 6500 hp not now used in the city, an additional value of \$20,000.

(c) For use for canal purposes an additional value of \$25,000.

The United States excepted to the additional value allowed in consequence of the availability of land in connection with the water power supposed to be the property of the Chandler-Dunbar Company, and supposed to have been taken by the Government in this case. It also excepted to so much of the awards as constituted an additional value by reason of availability for locks and canals.

"These exceptions so far as they complain of the additional value to be attached to these parcels for use as factory sites in connection with the development of horsepower by the Chandler-Dunbar Company, must be sustained. These 'additional' values are based upon the erroneous hypothesis that that company had a private property interest in the water power of the river, not possibly needed now or in the future for purposes of navigation, and that that excess or surplus water was capable, by some extension of their works already in the river, of producing 6500 hp.

Having decided that the Chandler-Dunbar Company as riparian owners had no such vested property right in the water power inherent in the falls and rapids of the river, and no right to place in the river the works essential to any practical use of the flow of the river, the Government cannot be justly required to pay for an element of value which did not inhere in these parcels as upland."

"The requirement of the fifth amendment is satisfied when the owner is paid for what is taken from him. The question is what has the owner lost, and not what has the taker gained."

"The exception taken to the inclusion as an element of value of the availability of these parcels of land for lock and canal purposes must be overruled. That this land had a prospective value for the purpose of constructing a canal and lock parallel with those in use had passed beyond the region of the purely conjectural or speculative. This land was the only land available for the purpose. It included all the land between the canals in use and the bank of the river. Although it is not proper to estimate land condemned for public purposes by the public necessities or its worth to the public for such purpose, it is proper to consider the fact that the property is so situated that it will probably be desired and available for such a purpose."

The Relative Importance of Principles and Practice in Education.*

By Dr. James Douglas

My first visit to Colorado was made in 1872. My mission was to report on the California mine in Gilpin county, which had been sold to a Scotch company, and was being restored to the original owners by other methods than those of bargain and sale. As a mine expert I got a lesson in how not to conduct legitimate mining. On the other hand I saw the first successful Western attempt at smelting argentiferous and auriferous copper ores at the old Black Hawk works of Senator Hill; and I made the acquaintance of a brilliant young man, Richard Pearce, in charge of the Swansea works, at the bend of Clear Creek, below Georgetown. This was only forty years ago, but the changes which have been made in metallurgy since then suggest many reflections on methods of scientific education, and on the equally important habits in self-training we should cultivate and practice on ourselves.

Early Metallurgical Practice

Let us glance at these changes. The Boston and Colorado smelter, under the management of Professor Hill, who had been a professor at Brown University, commenced operations in 1867 with two hearth roasting furnaces and two reverberatories, 15 ft. by 9 ft., heated by wood as fuel at \$5 per cord. The lump sulphides were calcined in heaps. Each reverberatory received its charge of two tons of roasted ore and roasted concentrates, which constituted a self-smelting mixture and yielded a matte of about 40 per cent copper. This was shipped to England for refining and separation.

But it was Mr. Pearce to whom the credit is due for taking the lead in enlarging the reverberatories, increasing their capacity and reducing the fuel consumption. In his address as President of the American Mining Institute of Mining Engineers in 1889,¹ he tells us that "the furnaces at Argo, to which place the works were moved from Black Hawk, had been constructed with a capacity of twenty-five to twenty-seven tons per day, and with a fuel consumption of one to three." The following table, supplied me by him for my Cantor Lectures,² illustrates the steady growth of the reverberatory:

Year	Size of furnace ft. in. ft. in.	Area of hearth, sq. ft.	Area of fire box, sq. ft.	Ratio of hearth to fire box, sq. ft.	Tons of ore smelted in 24 hrs.	Tons of ore smelted in 24 hrs.	Tons of coal con- sumed in 24 hrs.
1878	15 0 x 9 8	108	22.55	4,800:1	12	..	5
1882	17 10 x 10 4	143	22.50	6,351:1	17	..	7
1887	21 2 x 12 8	202	24.75	8,161:1	24	..	9
1891	24 4 x 14 2	262	28.50	9,192:1	28	..	10
1893	30 0 x 16 0	390	32.50	12,000:1	35	43	13
1894	35 0 x 16 0	456	32.50	14,030:1	..	50	13.5

But startling as was that progress for that day and generation, it was slow in comparison with that of late years, not so much by increasing the size of the reverberator as by improving its operation.

Development of the Reverberatory

Mr. E. P. Mathewson, in his paper before the recent eighth International Congress of Applied Chemistry, on the "Development of the Reverberatory," says:

"The next step in development was made in Butte, Montana, by the Colorado Smelting Company, this plant being at that time affiliated with the Argo works, so that Mr. Pearce's influence was apparent. The step referred to was the lengthening of the hearth to 50 feet, with consequent increase in capacity to 105 tons in 24 hours.

"The first furnace of this size built from the Colorado Smelting Company's plans, was constructed at the Butte and Boston plant in Butte, in the year 1900."

At Cananea, under Dr. Ricketts, the use of oil as fuel and the recovery of waste heat were studied with care. But the highest results seem to have been attained at the plant of the Steptoe Company at McGill, Nevada, with California oil as fuel.

Mathewson says: "A record performance at McGill, communicated by Superintendent Sorenson on December 17th, 1911, No. 1 furnace smelting 660 tons of total charge on an oil consumption of five-eighths of a barrel of oil per ton of charge.

Steptoe Plant at McGill, Nevada.

Record Tonnage on December 17th, 1911.

Record of Running of No. 1 Reverberatory Furnace, and Analysis of Charge.

Total charge per furnace day, tons.....	666.
Oil fired per furnace day, bbl.....	421.
Coal equivalent of oil fired, tons.....	124.0
Total charge per bbl. of oil tons.....	1.58
Oil, bbl. per ton of total charge.....	.63
Equivalent gross coal as % of total charge.....	18.60*

The substitution of oil for coal has, from the point of view of cheaper operation and the control of the heat, taken place wherever the difference in the cost of the unit of heat value does not forbid it. The low cost of smelting now attained in the reverberatory is attributable also to the conversion of the waste heat, as it escapes from the throat of the furnace, into steam. This loss has always been recognized, though with the old type of boilers it was never found practicable to remedy it; but in this, as in most other metallurgical processes, the mechanic has co-operated with the metallurgist to consummate what each alone would have failed to accomplish. From forty-five to fifty-five per cent of the heat generated by the fuel is recovered as steam. The combination of the reverberatory with the blast furnace, instead of the often unreasonable exclusive preference for the one over the other, is now finding many advocates. At the Cooper Queen works at Douglas we use both, and there we have imitated Cananea in fettling our reverberatory furnaces by pouring in ore of suitable fusibility along the wall of the furnaces through openings in the roof.

Influence of Bessemer Converting and Electrolytic Refining

But the development of the reverberatory would not have revolutionized the smelting and refining of auriferous and argentiferous copper ores had not two other great inventions intervened during the short period of time under our review. I refer to the introduction of the pneumatic method, through the Bessemer converter, and the refining of copper by electrolysis.

It was about 1882 that the news of M. Manhes' successful application of the converter to the concentration of copper induced Mr. Franklin Farrel to introduce the pneumatic method into his Parrot works in Butte. Difficulties, of course, beset him, but they were so slight that by 1885 he had six stands in successful operation. Since then, the principle and main features of practice being accepted, the progress, both in concentrating to bullion and incidentally in learning how to smelt crude ore in the converter, has been such that the converter has become more essential to the copper metallurgist than to the steel worker. It has grown in size from 4½ feet in diameter to 20 feet, and from a capacity of one and a half tons of matte per charge to thirty-three tons.

Instead of an acid lining requiring frequent replacement, the shell is now lined with basic bricks so refractory that, under proper precautions, 15,000 tons of copper are poured from a single lining, this first lining being still in use; and owing to the acid ingredients of the charge a much wider latitude of ores can be allowed for the acid ingredients of the charge, necessary to eliminate the iron from the matte, than when special clays and quartz had to be selected on account of their plastic or other qualities.

The converter has thus become a smelting furnace as well as an appliance to eliminate the older methods of concentration

*Commencement Address, Colorado School of Mines, May 23, 1913.

¹Transactions, A.I.M.E., Vol. xviii, 1889.

²Cantor Lectures before the Society of Arts, London.

by repeated roastings and smeltings, which involved so much fuel and time and labor.

And almost simultaneously with the practical introduction of the converter, improvements in the generator to produce electric current in large quantities and cheaply, made possible the electrolytic refining of copper and the separation of all impurities, including gold and silver.

First Electrolytic Refining of Copper

When I was superintendent of the Chemical Copper Company's works at Phoenixville, in 1878, Mr. Franklin Farrel, to whom the copper industry owes more, perhaps, than to any one man of the vanishing generation, induced me to make an experiment in the electrolysis of copper, at first on a small scale, with battery current, and then on a working scale, with generators, and in vats arranged in series and in multiple arc.

Our ignorance was replaced by the knowledge and gracious assistance of Mr. Edward Weston of Newark, who loaned me three of his nickel-plating dynamos, and, better still, came to Phoenixville more than once to advise and help me out of difficulties. I suppose I may claim the merit of making, in this country, the first electrolytic copper by the ton, but the merit is really due him, who in this and innumerable other instances has concealed his disinterested work for his favorite science and pursuits under a thick veil of modesty and generosity.

My idea at first was that the cathode might be rolled into coherent sheets. I soon discovered my mistake, but the copper was all bought at high figures by the Shaker community, who found that the cathodes yielded an extraordinarily good quality of metal for anodes in their depositing vats. The Chemical Copper Company was always in difficulty. It started with a shortage of capital, and never covered the shortage. The works came to an ignominious close. The company was not prosperous. The price of copper in 1878 and 1879 was at a low ebb, but took a sudden jump to twenty-five cents before the close of 1879. I had a nice stock of copper on hand and in the vats and was doing well; but the temptation to realize on such a soaring market was irresistible, and I was ordered to refine everything I could find, at the sacrifice of even the precious metal contents.

Excuse this personal digression. It was one of the blind steps in the progress of events which had a momentous influence on your immediate neighborhood; for the rapidity of the pneumatic process and the perfection of the electrolytic process inevitably supplanted the more intricate, though more interesting, process from a metallurgical standpoint, Pearce's modification of the Ziervogel method, as practised first at Black Hawk and then at Argo.

Prices Paid for Ore by the Boston & Colorado Smelter

Up to the date of Mr. Pearce's alliance with Senator Hill the mattes, as I have said, were shipped to England. And the prices paid for ore and concentrates were low. Fortunately, as a compensation, the value of silver was high. Mr. Pearce, in his presidential address to the American Institute of Mining Engineers in 1889, gives the proportion of values paid the miners in those early days. He says:

"The following table, which I have prepared from data collected by a friend, will show the commercial advantages which the miner has experienced by the progressive development of the smelting industry. For the sake of comparison, I have selected ores which have no special value as fluxes, or as aids to smelting, and will consider them merely in relation to their intrinsic value, and endeavor to show the returns which the miner gets now, as compared with the figures of eighteen years ago. In other words, the net percentage value of an ore to the miner today is compared with the value for the successive periods from 1871.

"The value of the silver has been figured from \$1.29 per ounce in 1871, down to 93 cents in 1899; and for the years prior to the resumption of specie payment the premium on gold is taken into consideration. The slight falling off in

1874 and 1875 was due to the depression following the financial panic in the fall of 1873. For a time silver ores were rather a drag on the market, and prices consequently fell off.

Table Showing Percentage of the Total Value of Ores Paid to Miners Each Year During a Period of Eighteen Years.

Place	Year	Contents in silver per ton	Percentage of total value paid to the miner
Black Hawk.....	1871	100 oz.	65
	1872	100 oz.	65
	1873	100 oz.	65.5
	1874	100 oz.	53.6
	1875	100 oz.	60
	1876	100 oz.	67.2
	1877	100 oz.	64.3
	1878	100 oz.	65
Argo.....	1879	100 oz.	70
	1880	100 oz.	74
	1881	100 oz.	74
	1882	100 oz.	76
	1883	100 oz.	76.5
	1884	100 oz.	81
	1885	100 oz.	77
	1886	100 oz.	80
	1887	100 oz.	80
	1888	100 oz.	82
	1889	100 oz.	84

"The difference between the maximum and minimum is 30.4 per cent.

"I have avoided making any figures to show the changes in value of gold-ores from year to year, but, going into details, I may state that a Gilpin county gold-ore which would net the miner 53 per cent of its value in 1870 would now yield him 80 per cent, a difference of 27 per cent; and on a somewhat lower grade of ore the difference is 33 per cent."

Since then the gold would yield the miner 90 per cent. Comparing the prices paid for concentrates in 1874 and 1889 he quotes 24 per cent of the value as that of 1874 and 78 per cent as that in 1889.

Professor Hill in 1872 defended himself in the *Central City Register* against the accusation of extortionate charges based upon unjust comparison of his furnace treatment with California milling practice. The professor says:

"The Boston and Colorado Smelting Company are treating ores, of which the gross value of the gold and silver, estimated in currency, is \$50, \$100, and \$150, at a cost to the miner of \$35, \$40, and \$45 respectively; that is, for ores which contain \$50 per ton, currency value, all over \$35 is paid to the seller, and for ores containing \$100 per ton, also all over \$40 is paid to the seller, and so on. For intermediate grades a *pro rata* charge is made.

"This company also pays for the copper \$1.50 for each per cent on the dry Cornish assay, which is the assay on which all copper ores are sold.

"No one who is acquainted with the facts will deny that the ores of Colorado are the most complex which are worked on this continent, containing, as they do, mixed with the sulphurets of antimony, arsenic, zinc, and lead, and a refractory gangue. Neither can any one deny that the actual costs of all the principal elements employed in smelting, viz., fuel, labor, firebricks, and iron, are more than double here what they are east of the Mississippi River, and much higher than they are in California."

Dr. Raymond, in his report as Commissioner, for 1870,* gives the following as the scale of prices paid at the Hill works:

"For ore containing per ton—	Is paid of the value
2 ounces gold.....	20 per cent
3 " "	30 " "
4 " "	40 " "
5 " "	45 " "
6 " "	50 " "

"For silver, seventy-five cents per ounce is paid, after deducting as many ounces of silver as there are per cent of copper in the ore. For copper \$2 for each per cent, deducting one-half per cent from the amount indicated by wet assay

*Mineral Resources West of the Rocky Mountains, 1872. R. W. Raymond.

*Ibid., 1870.

No account is taken of quantities less than one ounce of silver, one per cent of copper, or one-quarter ounce of gold. The above rates are in coin."

Early History of Smelting in Colorado

In 1875 Professor Eggleston published a paper on the Boston and Colorado Smelting Works,⁸ after Mr. Pearce had joined Professor Hill and was separating the precious metal. His description is significant of the small scale on which work was still conducted, and the high terms on which ore was bought. He says:

"The works are thus located in the very center of the gold and silver producing regions of Colorado, and are also most favorably situated with regard to transportation. They treated in 1874 30 tons of ore and tailings in 24 hours, and produced 700,000 ounces of silver, 12,000 ounces to 15,000 ounces of gold, and 225 tons of copper. With matte from Alma their production in 1875 will be 110,000 ounces of silver, 25,000 ounces of gold, and 250 tons of copper.

"The gold ores are divided into three classes. The first class consists of auriferous copper pyrites containing from 2 to 10 per cent of copper, 2 ounces to 10 ounces of gold, and 2 ounces to 10 ounces of silver. These ores average 4 per cent of copper, $3\frac{1}{2}$ ounces of gold, and 6 ounces of silver. The second class are tailings from the gold mills, consisting of pyrites with about $1\frac{1}{2}$ per cent of copper, $1\frac{1}{4}$ ounces of gold, and four ounces of silver. The third class consists of tellurium ores: which have a very silicious gangue, and contain 100 ounces to 200 ounces of gold, and 6 ounces to 10 ounces of silver. These ores come mostly from Boulder county, and are often worth \$10,000 to \$15,000 to the ton.

"The silver ores of the first class consist of surface ores, mostly free from sulphur, containing 70 per cent of silica. They contain 100 ounces of silver and 5 to 6 per cent of lead, and no gold. Those of the second class are sulphurets, rich in blende and poor in galena and pyrites; they contain 150 ounces of silver, 15 per cent of zinc and lead, and no gold."

The next contribution to the literature of our subject was the presidential address from which I have quoted.

By that time the smelting industry of Colorado had expanded from about 20,000 tons per year in 1877 to 596,594 tons in 1888, and Argo was treating some 200 tons of copper-bearing material from all sections of the Rocky Mountains, including the matte from the company's famous branch works in Butte, operated under the superintendence of Mr. Williams.

When Mr. Pearce had introduced the Ziervogel method of separating copper and silver, substantially as practised in Germany and England, and his own modification of parting the gold, the heavier expense of shipping the matte to England or Germany was avoided, and, as the above table showed, the miners shared in the saving.

But the method was delicate, and costly, involving not only skill but many operations. And consequently it has not been able to compete with matte concentration in the converter, and the perfect separation of the precious from the baser metal in the electrolytic vat. The requiem of this famous enterprise was thus pronounced⁹ in "Mineral Resources for 1910," Part I. Page 196:

"The year witnessed the dismantling of the plant of the Boston and Colorado Smelting Co. and the passing of this pioneer in the copper smelting industry of the West. Although this company has been successful in its long period of operations, the management did not consider it wise to rebuild the plant, which was becoming out of date. This pioneer plant has had a long and successful career, but copper smelting plants constructed in the State in recent years have been far less successful and have been able to operate but intermittently."

The moral in its effect on us as students is to regard methods, even the most ingenious and scientifically perfect, as merely stepping stones across the river of industrial life, the further side of which is still far away.

Pre-eminent Value of Principles in Education

Looking back, therefore, on the changes in these two branches of mining and metallurgy, namely, in the improvements and changes of methods and the higher price which the miner receives today for the products of his industry, we are forced to recognize how, in our system of education, the study of principles should occupy a more prominent place than the study of practice. Even our acceptance of principles, outside of mathematics, should be held as open to modification, but in the main the fundamental laws of physics and chemistry may be accepted, as we interpret them, to be correct and our only safe guides.

Every improvement made in your own region has been through a better understanding of these laws, and carrying them into operation.

For instance, the effect of the injection of air into molten metal was perfectly understood before Bessemer brought mechanical ingenuity as well as chemical science to bear upon the solution of the pneumatic method. The practicability of extracting the carbon to the exact point at which pig iron is converted into steel proved to be so difficult that the Bessemer process would probably have had a limited range of usefulness had not the suggestion of another chemist been adopted, to oxidize all the carbon, then add to the charge a specific amount of carbide of manganese with a known quantity of carbon to re-carbonize, while the manganese absorbed any dissolved oxide of iron. The whole success of this momentous improvement, which was not discovery at all, depended upon the application of known facts to meet certain practical conditions.

Following along the line of steel manufacture, the adoption of the basic lining in the converter, following the Thomas-Gilchrist proposal to line the open-hearth furnace with a basic material to eliminate the phosphorus was a simple application of known chemical facts.

Development of the Copper Converter

Turning from the pneumatic method as applied to iron to similar methods as applied to copper, we see Holloway carrying his smelting of ore and concentration of matte up to the point where metallic copper began to form; then the occlusion of the tuyeres in the bottom of the steel converter by the chilled copper; and the loss of years in the adaptation of this simple method to the concentration of copper till M. Manhes, adopting a type of converter with elevated tuyeres, which Mr. Bessemer had himself patented, brought the tuyeres within reach of a punching bar, and thus substantially revolutionized the metallurgy of sulphides of copper.

The wonderful progress in the study of the generation of electricity, its transmission, its conversion into power, and its electrical action has been made within the short period we have been reviewing. It is a special branch of study and research, the intricacies of which the average student cannot thoroughly master, but with the general laws of which he should be familiar. The carrying out of any mining scheme which utilizes electricity involves the employment of an expert in the person of an electrical engineer, but if you are to be an efficient manager of the mine, mill or furnace plant you must personally know what can be done with this mighty but mysterious force, though you may have to leave the handling of it to your electrician.

Another element which of late we have called to our use for service at a distance from the point of its generation is compressed air. To determine whether in each particular case transmission by electricity or by compressed air or by a combination of both should be adopted involves on the part of the manager a fair acquaintance with the fundamental laws of electricity and pneumatics, if he is to be the controlling factor in the administration, instead of a puppet in the hands of his subordinate officers. He may not be familiar with all the practical appliances or the latest improvements for applying these principles, but he must be sufficiently master of the subject to be the master of his staff.

⁸Transactions, A.I.M.E., Vol. iv, 1875.

⁹Mineral Resources of the United States, 1910, Part I, page 196.

Value of Industrial Chemistry in Development of the West

There is another field which the mining and metallurgical engineer of the future will have to cultivate, and that is industrial chemistry. It is, of course, a branch of industry in which co-operation between the chemical manufacturer and the miner and metallurgist must be closer and keener than it is at present. In fact, our metallurgical activity has been greater than the chemical, partly due to the fact that the market for chemical products has been widely distant from the possible sources of some of the more necessary chemical ingredients. But this distance is being gradually eliminated.

One of the most important iron and steel works of the country is already established at Pueblo, and there is no reason why within a limited period important chemical industries should not travel westward with the westward wave of population. Then the demand for sulphuric acid and that immensely valuable and yet noxious gas, sulphurous acid, which we are pouring into the atmosphere all over the West, will come into urgent demand.

But I think that the miner and metallurgist should not wait. He should himself take the lead in developing a Western chemical industry. We are wasting the basic ingredient—sulphuric acid. You have within easy reach probably the largest deposits of phosphate of lime rock in the world. The farmer will need superphosphates and it rests with you to supply him, and in doing so develop new mining industries and recover a valuable by-product.

Problems in Metallurgy of Rare Elements

In addition, your State possesses probably more rare minerals than you have any idea of, the recovery of which and the extraction of their precious contents will require of you a closer study of chemical operations than has been usually demanded of the students of metallurgy. Take, for instance, your carnotite ores. The separation of the three valuable elements—radium, uranium, and vanadium—should become immediately a branch of metallurgy.

In addition to the sulphurous acid which now goes to waste from your furnace stacks, the waste gases from your coke ovens should pave your roads with asphalt and fertilize your fields with sulphate of ammonia. It is evident, therefore, that the alliance of mining and metallurgy and the chemical industry must become in the future much closer than it has been in the past.

And to get that total out of the material we treat, or if not the total more than has heretofore been recovered, will be one of the tasks of the future. The decline in the value of the ores of the Gilpin, Cripple Creek and other districts is reflected in the falling off of the mineral production of the State.

This decline must be compensated for, not only by reducing the losses in treatment of the standard elements recovered, but by saving others now wasted. Looking over the table of your mineral products, I see that this process is already in operation, for zinc appears for the first time in the list in 1902, and the value of this once neglected element saved in 1912, \$8,591,623, was greater than that of your silver, \$5,023,960; and it alone enables you to maintain your position in the ranks of producing States.

But there are other valuable ingredients—let us call them by-products—which might be utilized, and to do so the assistance of the industrial chemist must be secured. You, as supplying the material on which he works, must know more of technical chemistry than has heretofore been thought necessary.

Fundamental Investigations Required for Future Prosperity

We are living at the close of a period of almost unnormal activity and prosperity. We have had half the new world to exploit; and, with steam and electricity as our aids, in little more than a generation we have depleted the fertility of the soil and emptied our mines of their richest ore, taking from each the stores nature had accumulated nearest the surface. The harvest has been abundant and cheaply garnered; but a future generation cannot reap crops without planting,

and in our case, if mining prosperity is to be maintained, it must be by saving every product and by-product, and widening, therefore, the field of our operations and consequently of our studies.

The experience here of the past generation demonstrates how rapidly practice has changed under the application of scientific principles. The conclusion, therefore, is that, as a fundamental maxim of education, our studies should aim at a better understanding of the deep and broad laws which underlie all practice rather than the ephemeral methods of the day, and thus train our minds in habits of original research rather than obediently and servilely following the practice of the past or even of the present. The study of methods, however, cannot be neglected; but, if familiar with the principles, you are better able to use your principles in assisting you to devise methods for carrying them into practice.

The danger of too much knowledge of principles is that the scholar becomes so conscious of his shortcomings in practice and sees so keenly the possible perfection of practice ahead that he does not devote himself with sufficient energy to working out and practising the imperfect, practical methods and using tools he has at his hand to work with. Some of the cleverest men I have had to do with have developed this fault. It should be your endeavor to familiarize yourselves with the scientific basis of the subjects you have to deal with without destroying your practical ingenuity and being depressed by the evident shortcoming of your best efforts to carry principles into action.

Electro-Analysis of the Copper Alloys

By J. G. Fairchild

In the analysis of brass and bronze, a common method of procedure consists in the oxidation and removal of tin and antimony by means of nitric acid, followed by their solution in ammonium hydrosulphide. This solution, after separation of the small amounts of the sulphides of copper, lead and iron, is then electrolyzed with the addition of potassium cyanide as reducing agent. The strong tendency of this alkaline solution to form polysulphides prevents the complete deposition of the metals. Although this treatment is apt to fail, yet the necessary operations are short. This is an important factor in a commercial laboratory. The old method of purifying the oxides of tin and antimony by fusion with sulphur and soda, with the subsequent weighing of the stannic oxide after antimony has been removed, appears to be too long to admit of practical use. Furthermore, to gravimetrically determine tin is liable to result too high and is recommended only when dealing with traces. With the solution of this first problem, the remaining successive operations offer little difficulty.

The alloy is dissolved in 15 cc. nitric acid, 1:1, and evaporated nearly to dryness, or until the liquid becomes a thick syrup. It is convenient to use a beaker of 250 cc. capacity. Addition is made of 5 cc. more acid with 50 cc. of water and the solution is boiled for a few minutes. After filtering, the stannic oxide is washed well with a hot acid solution of ammonium nitrate. A convenient strength is 2 per cent of the salt with 2 per cent of nitric acid. This wash aids in freeing the tin from copper, besides preventing the tendency of the precipitate to be carried through the paper.

The precipitate is washed back into the beaker and dissolved in five grains each of ammonium oxalate and oxalic acid*. The solution is diluted to 100 cc. and boiled till clear.

The filter paper is next placed in the solution and disintegrated by boiling, the solution is refiltered and the paper pulp is washed with hot water. When tin is under 10 per cent, as in most brasses, it is needless to purify the solution.

Several color determinations of copper occluded by this amount of tin have shown usually less than 1/10 of 1 per cent. Since copper is the greatest impurity lead and iron count for still less.

*One grain of NH_4Cl must be added here to aid in the deposition of tin.

If it is desired to make a separation of this copper, or if antimony is present, heat the solution to about 95 deg. C. and treat with H_2S for forty-five minutes according to G. W. Thompson's method.

The tin is filtered from the antimony and copper sulphides which are washed well with hot water. The precipitate is washed back into the beaker with a little water and treated with 2 grams of KOH.

The liquid is warmed till the sulphides blacken, when all the antimony will be in solution. Add 50 cc. concentrated HCl to the antimony in a 200 cc. Erlenmeyer flask, together with one gram of $KClO_3$, and boil till the sulphur is oxidized and the ClO_2 is all expelled.

While still warm add one gram of KI to the solution, cool quickly in ice water with further addition of cold water and titrate the antimony with thiosulphate.

The small amounts of copper and lead are dissolved in a little strong HNO_3 , evaporated to near dryness and added to the main copper solution.

Tin.—Boil the filtrate from the copper and antimony to expel H_2S and transfer hot to a dish of 100 cc. capacity. The surface of this dish should be 1 square decimeter. A nickel dish can conveniently replace the usual one of platinum. Use a current density of 0.8 amp per square decimeter and pass a slow stream of air through the solution.

In order to prevent loss by bursting of the air bubbles, it is convenient to pass the stem of the spiral anode up through a small inverted glass funnel. The glass tip through which passes the compressed air is inserted beneath the funnel. It is also safer to further cover the dish with a pair of split watch glasses. The air circulation need be only two bubbles per second.

Deposition is complete in about 2½ hours, as indicated by a cessation of the vigorous gasing of the electrolyte. At this point alkalinity begins, as shown by a litmus paper test. The electrolyte is siphoned off as usual and the dish, after rinsing with alcohol, is ignited and dried over a Bunsen flame. The deposit of tin made in this manner is dense and brilliant.

Phosphorus.—The electrolyte from the tin determination contains all the phosphorus, which is oxidized by evaporation with nitric acid and precipitated by ammonium molybdate as usual.

Copper.—This metal may be deposited in three hours by using the same current density as above. Lead is obtained simultaneously on the spiral. The weight of peroxide for most brasses is less than 20 milligrams—an equivalent of 1.73 per cent metal on the one gram sample. If the lead is so large in amount that it falls off the spiral and adheres to the copper, dissolve both metals and deposit the lead first with reversed poles.

Zinc.—The electrolyte from the copper and lead determinations is evaporated rapidly over a Bunsen burner. Use a large porcelain casserole and bake the residue on a hot plate till ammonium nitrate is completely volatilized. This salt must be expelled before the deposition of zinc. When the residue is thoroughly dry, cool and introduce 1 cc. concentrated H_2SO_4 with a little water. Heat till the zinc and iron are completely dissolved. Dilute to 50 cc. with the addition of 5 grams NaOH and a few grains of Na_2O_2 . The zinc readily dissolves as sodium zincate.

After heating the solution for a few minutes, filter through a plug of asbestos and glass wool inserted in a glass funnel for the removal of iron, etc. If the zinc amounts to more than 20 per cent take half of the solution for electrolysis. More NaOH is added to make the strength to per cent, and the electrolyte is heated to about 60 deg. C. By using the copper-coated platinum dish for zinc an extra weighing is dispensed with.

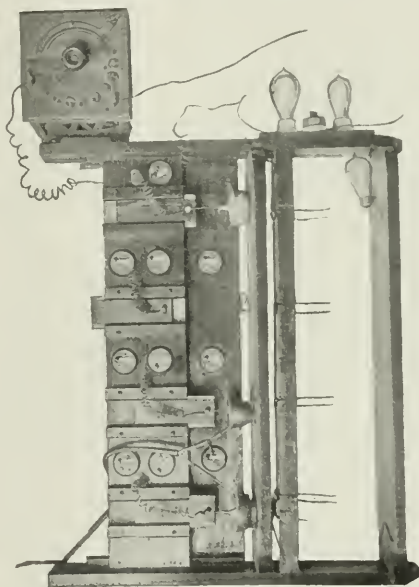
Zinc is completely deposited in from two to two and one-half hours. This point is determined by withdrawing 2 cc. or 3 cc. in a test tube, boiling out the dissolved oxygen and adding an excess of NH_4HS . After a thorough warming and shaking the solution should remain clear throughout its depth.

Aluminium gives the same reaction here as zinc, so re-examine the electrolyte if a cloud should appear.

Iron.—Dissolve the residue on the asbestos with dilute HCl. This small amount of iron is conveniently and accurately estimated by reduction with KI and titration with thiosulphate. Copper, if not already completely removed, will interfere with this method for iron.

Manganese.—This will be found with the iron. The two oxides are dissolved in hot nitric acid, 1:1, by the aid of bisulphite. The solution is divided into two equal parts. One is evaporated to SO_2 fumes with addition of some H_2SO_4 , after which HCl is added and titration made as before. The other is oxidized with sodium bismuthate and the manganese titrated as in iron and steel.

Nickel.—This metal, if present, can be determined after the titration of iron. The solution is reduced with SO_2 , made alkaline with NH_4OH , then acid with acetic acid and treated



ELECTROLYTIC BOARD AND WIRING

with H_2S while warm. Small amounts of NiS are best weighed as NiO .

Aluminium.—This metal may be looked for in the electrolyte from the zinc and determined as usual.

Suggested Procedure When Dealing with White Metals

No better scheme can be recommended than that of G. W. Thompson, with one or two exceptions. According to his procedure, the alloy is dissolved in aqua regia, lead chloride crystallized out with alcohol and the filtrate evaporated to dryness. It is then made alkaline with KOH and oxidized with H_2O_2 . A large excess of ammonium oxalate and oxalic acid is next added, and the antimony, copper and traces of lead separated by means of H_2S . The tin is electro-deposited.

The antimony is separated from the copper and lead by solution in KOH, then oxidized with HCl and $KClO_3$ and titrated. Copper and lead are dissolved in nitric acid and electrolyzed. Any iron will be weighed with the tin.

Zinc, if present, appears as insoluble zinc oxalate which is filtered from the soluble tin and converted to sodium zincate for electrolysis.

An exception is made to his procedure in that lead chloride is converted to nitrate and electrolyzed. Use 15 cc. HNO_3 in the volume of 100 and 0.25 amp per square decimeter

The initial temperature of the electrolyte should be about 60 deg. C. in order to prevent the formation of metallic lead on the cathode. Two hours is sufficient for 150 milligrams of peroxide. During the last half hour the current density is increased to 0.5 amp.

Notes and Precautions.

For the determination of copper, an oval-shaped crucible of the same metal and of 100 cc. capacity will be found very convenient and inexpensive. In order to prevent any attack of the crucible by the acid electrolyte, connections are made before the solution is introduced.

Chromic acid is an excellent addition agent for copper, in that it prevents the formation of nitrous acid.

So long as the last deposit remains bright and adherent the successive layers need not be removed.

The outside of this crucible should be gold plated in order to prevent oxidation of the copper. Plating of the inside is optional.

Zinc is also weighed in the same crucible.

A containing vessel of this shape affords better circulation of the electrolyte.

When starting the action of the compressed air, dip the glass tip into a beaker of water; otherwise some of the electrolyte might be blown out by too sudden a rush of the air.

In determining tin in white metals, it is better to so portion the solution as to provide for not more than 0.2 gram for electrolysis. This amount can be deposited in three hours, but addition of more oxalic acid to the electrolyte may be necessary.

The largest amount deposited in this laboratory has been 0.3 gram.

If a check determination is wanted on a high tin electrolytically deposited, a good method consists in the solution of the drillings in a flask provided with a Bunsen valve. A few pieces of marble are then added and the flask cooled in ice water. The tip of a burette is furnished with a delivery tube which is passed into the flask. By this means the stannous chloride can be sucked up into the burette in order to avoid oxidation as a result of pouring it through the air. Delivery is then made into a weighed amount of boiling ferric chloride solution and the excess of tin is titrated back with iodine.

When dealing with high antimony, such as found in type metal, Mohr's method is recommended. This involves solution with HCl and HClO₄, reduction with KI and titration of the liberated iodine with thiosulphate.

A good air circulation is essential in all the electrolytic determinations, especially so in those of lead and zinc. The safe limit of deposit for 1 square diameter of surface has been found to be 150 milligrams of lead peroxide.

In converting zinc to zincate, be sure that solution is complete, both in H₂SO₄ and subsequently in NaOH. Wash the asbestos plug with 10 per cent NaOH. Not more than 0.2 gram has been deposited thus far, but this limit may possibly be increased by an addition agent, such as glue. This colloid has the property of making the zinc crystals smaller and hence more adherent. It also offers an obstruction to the vigorous evolution of oxygen gas from the anode, and thereby lessens the danger of mechanical loss of metal from the electrolyte.

By determining the copper, lead and tin in two dishes at the same time, a bronze analysis may be completed in about one day.

Caustic potash is much less efficient than the soda for solution for zinc, as the latter forms a more soluble compound with zinc than the former.

With a view of eliminating the evaporation to dryness of the copper electrolyte containing the zinc, the suggestion is made to apply the lactic acid electrolyte as given in Smith's Electrochemical Analysis.

I wish to credit Mr. F. W. Dugan with the design and construction of the electrolysis board and wiring.

Laboratories of Dr. Chas. F. McKenna,
New York City.

The Modern By-Product Coke Oven

One of the most interesting papers read at the recent meeting of the American Iron and Steel Institute was presented by C. A. Meissner, Chairman of the Coke Committee of the United States Steel Corporation, on the modern by-product coke oven. The complete paper comprises 64 pages and is very full and complete. On account of its length only a summary of its contents may be given here. One special section of the paper which should be of particular interest to our readers will be printed in full in our next issue.

The author first gives coke and coal statistics of the world and of Germany and the United States. The United States Steel Corporation in 1912 used a total of 40,877,862 tons of coal. For coking purposes, the Steel Corporation used 24,401,577 tons of coal. They produced a total of 16,719,387 tons of coke, of which 5,164,547 tons were made in by-product ovens and 11,554,840 tons were made in bee-hive ovens.

After giving the principal historical figures of the development of the coke industry, the author takes up the advantages of by-product coke ovens over bee-hive coke ovens and makes, concerning the cost, the following interesting remarks. They have studied the comparative cost of by-product coke ovens versus bee-hive coke ovens very carefully and find that this depends entirely on the location of the bee-hive oven plant and the attendant conditions of such a location. Comparing, for instance, a by-product plant at Gary, Ind., with a bee-hive plant in the Pocahontas, West Virginia, region, both on a large scale, and taking into consideration the mine investment, houses required for workmen and everything connected with the construction ready for operation of either type, they find that the bee-hive oven installation complete costs more than the by-product oven installation complete for the same number of tons coke produced per year. Comparing the above by-product plant with a similar bee-hive oven installation in the Connellsville region, where all conditions are more favorable and where the difference in the yield of coke from coal coked in the bee-hive oven and in the by-product oven is very materially less than it is in the Pocahontas region, they find that the cost of a bee-hive oven installation complete is considerably less than the cost of a by-product oven installation complete, per ton coke produced per year.

A feature of supreme importance in the use of the by-product oven is the greater coke yield obtained in this type than in the bee-hive oven from the same coals. This amounts to about 23 per cent. to 25 per cent. in the low volatile coals, such as Pocahontas, and from 5 per cent. to 12 per cent. in the high volatile coals, dependent on their original content of volatile matter. This subject is not fully understood and generally appreciated and yet is of vast importance in the conservation of our coal fields. A concrete example will be of interest:

	POCAHONTAS Bee-Hive Coke Oven Plant.	COAL COKED IN By-Product Coke Oven Plant.
Number of ovens.....	6,154	560
Coking time	72 hours	17½ hours
Yield of coal to coke.....	60%	82%
Net tons of coal required to produce 2,880,000 tons coke per year.....	4,800,000	3,512,000
Net tons of coal saved per year by use of by-product coke ovens for above coke production		1,288,000

This amount of coal saved, if it were coked in by-product ovens, would produce about 1,000,000 tons of coke per year.

Then follow coal data, a discussion of the quality of by-product coke for blast furnace use, construction features of by-product coke ovens, notes on coke quenching and screening, illustrated descriptions of various types of coke ovens (Semet-Solvay, Koppers, Otto, Collin, Coppee, Still), and a discussion of by-product recovery operation and apparatus.

Progress of Electric Smelting at Heroult, California

An Authoritative Report of the Progress Made at the First American Electric Iron Ore Reduction Plant. By John Crawford, Plant Manager

The plant of the Noble Electric Steel Company is situated on the banks of the Pitt River at Heroult, Shasta County, California, a station on the Sacramento Valley & Eastern Railroad, which branches off the Southern Pacific Railroad at a station called Pitt, lying about six miles to the west of Heroult. The plant consists of one 2000-kw and one 3000-kw iron furnace of the long and narrow type, one 2000-kw iron furnace of the shaft type (not now operated), one 2000-kw steel furnace of the tilting type (not now in operation), twenty charcoal retorts of the horizontal type of 2¼-cord capacity, a refinery to handle the by-products of the wood distillation, and four 5-ton unit lime kilns

There is on the coast an abundance of scrap cast iron, and foundries making what are spoken of in the East as heavy castings are few in number; hence the popular demand is for a soft high-silicon iron which is a good scrap carrier and can be easily machined when in light castings.

Thus it is apparent our problem became one not merely of making pig iron successfully but of making iron with a silicon content of from 2 to 3 per cent economically. When a blast furnace works "off" it ordinarily means only a slight concession in price to get rid of the low-grade iron, while with us here if iron runs under 1 per cent silicon it means a large concession in price to dispose of it. As electric furnaces like blast furnaces have the faculty of misbehaving at times conditions imposed on us the necessity of having a furnace which would respond readily to alterations in the furnace burden and still be of large enough capacity to be efficient.

History

It may be of interest to run briefly over the experiments conducted at Heroult which have led up to our present furnaces. For some twenty-five years one of the stockholders of this company has controlled a deposit of magnetite of considerable extent which is exceptionally high-grade and runs very low in sulphur and phosphorus.

The following analysis represents the average of several scarfs taken across a quarry face 70 ft. wide and 45 ft. high. The quarry face has now been extended to a width of 115 ft. and is still in ore.

Analysis of Magnetite

SiO ₂	3.43	per cent
Al ₂ O ₃	0.81	" "
CaO	0.70	" "
MgO	0.32	" "
MnO	0.28	" "
CuO	Trace	" "
Fe ₂ O ₄	70.63	" "
Fe ₂ O ₃	14.56	" "

100.73 per cent

Fe	97.86	" "
P	0.0116	" "
S	0.021	" "

Geologically the deposit has been formed by deposition from ascending waters and lies between the quartz-augite-diorite of this region and a metamorphosed (McCloud) limestone. The analysis of this limestone which is also very pure is as follows:

SiO ₂	1.02	per cent
Al ₂ O ₃	0.61	" "
MgO	1.12	" "
CaO	53.80	" "
FeO	0.20	" "
CO ₂ (by diff.)	43.25	" "

100.00 per cent

In 1906 these properties were brought to attention of H. H. Noble, president of the Northern California Power Company who, after some correspondence with Dr. Paul L. T. Heroult, built in 1907, a 1500-kw furnace of the three-phase resistance type. Pig iron was produced in this furnace, but mechanically it was weak.



FIG. 1—VIEW OF HEROULT LIME KILNS TO RIGHT, FURNACE BUILDINGS TO LEFT

of the continuous type. Beside these there is a power substation and the necessary machine shops, office, residence, and furnace buildings, etc.

Introduction

To build an electric furnace for making pig iron which can be operated as a technical success is quite an achievement, but this having been accomplished, there still remains the very considerable problem of making it an economic success.

With approximately the same costs for power, charcoal, and stock, local conditions have caused us to abandon the type of furnace which we are told in Sweden they are operating commercially.

If a certain type of furnace is found to be best suited to make a grade of iron for which the demand is limited, either a market must be found for this product, and that market educated to accept it, or else the design of the furnace, or method of operation must be altered so as to make a grade of iron for which there is a market already established. In Sweden, conditions permitted them to adopt the former and simpler course, while we at Heroult have had to resort to the latter.

The principal users of pig iron on the Pacific Slope and Far Western states are custom foundries. Specialty foundries, such as those making stoves, bath tubs, pipe, etc., are still relatively few, as are also open-hearth steel furnaces, so that to operate electric furnaces successfully on the Pacific Coast, grades of iron must be produced which meet the demands of general foundry purposes.

In May, 1908, there was designed by E. E. Cox and others under the management of D. A. Lyon, a furnace of the shaft type embodying the principle of a shaft terminating at its base in a bosh which was super-imposed on an arched crucible through which the electrodes penetrated at an angle. This furnace which was put in operation in the summer of 1909, by a singular coincidence is almost identical with the furnace built at Trollhättan, Sweden, about the same time. As descriptions



FIG. 2—FURNACE BUILDING

of the Swedish furnace have appeared so frequently in technical journals, I believe the design of this type of furnace is generally familiar, so I will omit a detailed account.

This furnace was operated intermittently until the spring of 1911. As chemist at the plant during part of this time, I was enabled to observe the mechanical and metallurgical workings of the furnace closely and was imbued with the idea that it could be made economically efficient.

But from the style of its construction, it is apparent that it could not be made to respond readily to changes in the burden, and in order to make consistently high-grade foundry iron this is an essential. In a blast furnace, too great an excess of coke in the burden can be taken care of by increasing the quantity of air blown in, but in an electric furnace this is, of course, not feasible, as the oxygen would attack the electrodes. The excess of carbon must be taken care of by an excess of oxygen put into the furnace through increasing the ore in the burden.

The necessary excess one might think could be calculated to a nicety, but because of unknown factors practically the excess must be added gradually until the controls on the slag from the iron and the general working of the furnace show the desired result to have been accomplished. This sometimes takes several days.

If the excess of carbon has been allowed to proceed too long, the furnace will, of course, "freeze up." The slag will give up part of its lime content to form calcium carbide, part of its alumina to form aluminium carbide, part of its silica to form silicon carbides, and part to form ferrosilicon, and part of the carbon remaining turns to beautiful sparkling flakes of graphite. Remarkable examples of molecular replacement of the carbon in the charcoal by silicon carbide has also been noted, making beautiful specimens of petrified charcoal. I once saw a furnace present these phenomena, though there is no excuse for it, as the decrease in the amount of stock going into the furnace and the daily controls on the slag and metal should give ample warning. The matter of too little carbon gives less trouble, and if the furnace is producing low silicon and carbon iron should give none at all.

The question may be fairly put: Why cannot the necessary carbon in the burden be calculated within sufficiently close limits to prohibit any possibility of trouble? Theoretically, of

course, it can be. By daily analyses of the furnace gases taken at regular distances as they ascend from the crucible up the stack the ratio in which the carbon is actually being oxidized to CO and CO₂ can be approximately determined, and this together with the controls on the slag and metal will keep the carbon within safe limits if the furnace is running with a low carbon burden; that is, making low-silicon iron.

When, however, the furnace is running on a high carbon burden, calculated to make a 3 per cent silicon iron, it is apparent that the carbon must be carefully regulated. Insufficient carbon not only lowers the grade of the iron, but introduces difficulty by throwing an excess of SiO₂ into the slag. Too much carbon lowers the efficiency of the furnace besides leading to the difficulties mentioned above.

Practically, it is always necessary to carry an excess of carbon over theoretical calculations to take care of the atmospheric oxygen and moisture which is occluded in the charcoal and the atmospheric air which is inevitably drawn into the furnace after it has been in use in spite of every practical precaution to prevent it. It is evident, of course, that these are to a considerable extent unknown factors; hence the charcoal must be continually varied, as the control analysis indicates that it is too high or too low.

So, while the shaft type of furnace gave promise of economic success, it operated on the low-silicon, low-carbon, white iron (which our Swedish friends have dignified by the name of pig steel), it was shut down for making foundry iron.

Following this, an attempt was made under the direction of S. T. Wellman to work out the theory of reducing the ore with the proper charcoal and flux to sponge iron in a series of small, portable electric furnaces which were then picked up one at a time by a mechanical charging machine like those used in charging open-hearth steel furnaces and their contents emptied into an electric smelting furnace similar in design to the arc-type of tilting steel furnaces. Mechanical difficulties and inability to control the metallurgy of the process caused it to be abandoned after a very brief period of operation.

General Description of the Present Furnaces

Subsequent to this, R. E. Frickey and others after considering the causes of failure of the previous furnaces designed a



FIG. 3—ELECTRIC PIG IRON

2000-kw three-phase furnace of the long and narrow type having four electrodes delta-connected and suspended between five charging stacks. This is the type of furnace which we are now operating and which seems best adapted to our needs.

The first furnace was put in operation in November, 1911, and has been working since, except for periods when operations were suspended to effect advantageous mechanical and electrical changes. All was not smooth sailing, of course.

Pioneer work never is, but we became satisfied that this type of furnace promised economic success and a companion furnace of 3000 kw capacity was built.

This type of furnace consists of a rectangular steel shell, 28 ft. long and 10 ft. wide, lined with standard furnace brick. The lower part of the furnace is battered four ways and corresponds to the blast furnace crucible, while the upper part of the furnace corresponds to the bosh or smelting zone of the blast furnace. The furnace is surmounted by five charging stacks, 18 ft. high, which are battered so as to prevent the descending burden from hanging up. Between the stacks the top of the furnace is arched over and through the center of these arches the electrodes penetrate vertically into the charge.

The electrodes used are of Acheson graphite, 12 in. in diameter, and are connected with a tapered male and female joint.

Up until now, no arrangements have been made on these furnaces to utilize the furnace gases as the way in which to utilize them best had to be studied rather as an economic than a metallurgical problem; that is, whether the saving in charcoal effected by circulating the gases back through the furnace and taking advantage of the reducing action of the CO is greater than the saving in fuel by burning them under lime kilns, char-

trodes on the low-tension side consist of eight pieces of $\frac{3}{4}$ -in. by 8-in. copper bus-bar connected in delta from the three transformers to the four electrodes.

The electrode holders are water-cooled cylindrical stuffing boxes made of 98 per cent copper. The electrodes are sus-

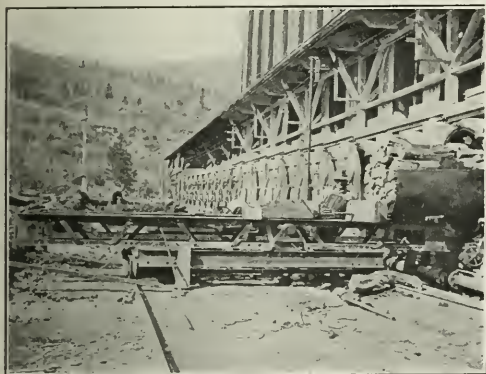


FIG. 4—CHARGING RETORT—SHOWING TRANSFER TROLLEY CAR

coal retorts, or elsewhere. Our observations have led us to adopt the latter course.

As to the claims made for the cooling effect of the circulated gases on the furnace roof, I have never considered this worthy of practical consideration on a resistance-type furnace. If the charge is descending regularly it will protect the roof and if it is not, the slight lowering of temperature caused by the cool gases will not prevent the heat radiated from the electrode from melting out the roof.

Electrical Description

Power is supplied by the Northern California Power Company three-phase at 60,000 volts into our own substation. Here are six Westinghouse 1500-kw oil-immersed and water-cooled lowering transformers wound for a voltage ratio of 38,100 to 2200 volts, Y-connected on the high-tension side. The high-tension leads pass through a General Electric oil switch by which the furnace load is cut in or out.

The three 750-kilovolt-ampere furnace service transformers which set close up to the rear side of the furnace furnish three-phase currents at from 40 to 80 volts. They are specially designed General Electric transformers, oil-immersed and water-cooled and capable of standing an overload of 25 per cent. On the 2200-volt primary side are brought out eight current taps for voltage regulation. These are taken to an oil-immersed individual solenoid-operated contactor panel which is connected with the switchboard to give fifteen steps for voltage regulation.

The connections from each of the transformers to the elec-



FIG. 5—FOOT OF GRAVITY TRAM

truded in these from above and the annular space between the electrode and the holder is packed with a specially prepared graphite capable of being compressed to a density equal to that of the electrode itself. This packing material offers no more resistance to the passage of the current than the electrode and because of its unctuous nature permits the electrode to be raised and lowered without breaking electrical contact.



FIG. 6—FURNACE No. 6

Electric control is through a switchboard on which are mounted, in rows of three, a separate set of meters for each phase. A set consists of a Thompson ammeter, a Thompson voltmeter, a Thompson wattmeter, and a Weston power-factor meter. Beneath the last named are three hand wheels to con-

trol the voltage variation and below these are three knife switches to control the load on the board. A separate recording wattmeter for each phase is also set on this board.

For operating the voltage control and the circuit-breaker, there is a $7\frac{1}{2}$ -kw motor-generator set comprising a 125-volt direct-current generator, direct connected to a 10-hp induction motor. Above these on a panel are mounted a circuit-breaker, an ammeter, voltmeter, and the necessary switches.

From an electrical standpoint, this type of furnace has worked far more smoothly than any with which I am familiar. When everything is normal inside of the furnace the instruments will show little variation throughout the twenty-four hours, except that the power-factors which average respectively 90, 85 and 70 per cent, improve a few points after the furnace is tapped and gradually fall off again as the molten iron accumulates in the bottom of the crucible.

Metallurgy

The stock is charged into the five charging stacks previously mentioned on the basis of 500-lb. units of iron ore. This small charging unit while it entails extra labor on the feed floor has the advantage of mixing the ore, charcoal, and flux as intimately as if the charge were bedded and homogeneity of charge is very essential.

With this type of electric furnace at least the ore is reduced to a very much greater extent by actual contact with the carbon than by the action of the CO in the stack gases. Some examples of gas analyses will bear this out.

Gas Analyses

CO ₂	O	CO	CH ₄	H
8.6	0.2	57.2	16.0	0.8
9.2	1.4	56.2	12.2	1.2
7.8	0.15	57.1		...
6.0	0.20	67.9		...
8.1	0.10	64.8		...
4.2	0.25	60.7		...
5.0	0.40	66.6		...

From these analyses it will be noted that the ratio of CO to CO₂ is very high, but by utilizing the calorific value of the gas,

scends; hence, less surfaces of carbon are exposed to be oxidized by the ore and there is a less intimate mixture of the two. Reduction of the ore takes place more slowly, the silicon in the iron is lowered, the power consumption per ton increases, and thus the efficiency of the furnace is reduced. However, by adopting certain precautions in crushing the stock and feeding same into the furnace I have operated on a mixture of 60 per cent coke and 40 per cent charcoal with a very fair degree of furnace efficiency and the grade of the iron was kept up to No. 2 foundry.

The possibilities of operating electric iron furnaces with coke

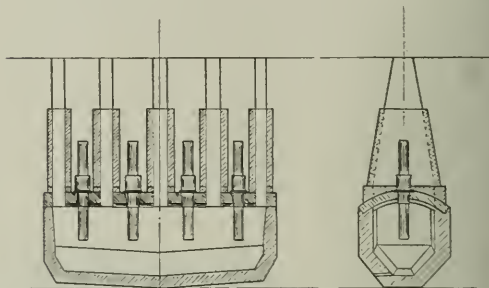


FIG. 8—SECTIONS OF NO. 7 FURNACE

instead of charcoal seem to me to offer a very interesting and necessary field for investigation. At present successful operation of electric iron furnaces depends among other things on an abundant and fairly cheap supply of charcoal. This generally limits their field of activity to well-timbered regions which are usually isolated and where freight rates are high. I believe that many of our coals which make a very poor metallurgical coke for blast furnace use on account of their low crushing strain might be found to make a satisfactory fuel for electric furnace use.

The fact that our ore is of so high a grade renders our metallurgical problem somewhat different from that usually encountered. The ore as it comes from the quarry will often run for weeks at a time as low as $2\frac{1}{2}$ per cent SiO₂, and from 67 to 68 per cent Fe; so that to make pig iron which will run from 2 to 3 per cent Si, it is necessary to augment the silica in the ore by the addition of barren quartz. The requisite amount of lime or limestone is added so as to give theoretically a slag running about 47 per cent SiO₂. The following slag analysis will indicate the extent of the silica variation:

SiO ₂	Al ₂ O ₃	FeO	CaO	MgO
54.00	20.70	1.30	12.08	1.17
50.20	28.60	2.40	16.03	2.36
46.13	27.20	0.65	23.10	3.31

Ordinarily slags containing more than 50 per cent SiO₂ are too viscous to run well, but our slags are fluid up to 54 per cent SiO₂. This may be due to the high alumina ratio. As we only make from 125 to 140 lb. of slag per ton of pig, it is found more economical to permit a small percentage of the iron to escape in the slag than to attempt to reduce it. The depth of the green color in the slag also serves as a rough indication for the furnace-men as to whether the carbon ratio of the burden is becoming too high or too low.

In calculating the charcoal for the burden we assume all the carbon burns to CO, as we find it necessary any way (for reasons previously given) to carry an excess of charcoal in order to make high silicon iron. Thus to make 2.75 silicon iron the theoretical quantity of charcoal (containing 85 per cent fixed carbon) necessary is 35 per cent of the pig, whereas there is actually used about 40 per cent. Inasmuch as any necessary change in the burden is distributed over five stacks the furnace responds very rapidly and practically permits of the silicon in the iron to be controlled within a limit of 0.5 per cent.

The iron is tapped three times a day into sand pig-beds. A sample is taken from each bed from a pig to inches long cast for



FIG. 7—NOS. 6 AND 7 FURNACES; PIG BED BEFORE NO. 7 FACE

the high charcoal consumption will be balanced to a very appreciable extent.

I have experimented to some extent with the possibility of utilizing coke or gas charcoal instead of charcoal.

But the 72-hour metallurgical coke such as I have tried offers two objections: First, its electrical conductivity is so good that much of the current passes between the electrodes in the upper part of the furnace. The smelting zone is thereby raised and the furnace runs hot on top with attendant melting of the arches and cold at the bottom.

Second This coke because, of its density and high crushing strain does not break down like charcoal as the burden de-

the purpose. The drillings from the several beds are ground together for analysis. The sow and pigs are lifted from the beds by a traveling crane with grab-hooks and broken by a drop used in connection with an electric magnet.

Our system of grading is somewhat like that used in grading the Southern iron, except that we grade entirely by silicon content as the sulphur and phosphorus run uniformly under 0.04 per cent when we operate on charcoal alone.

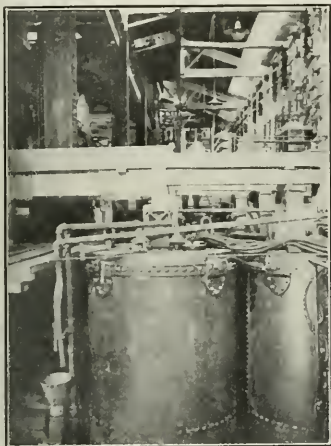


FIG. 9—SHOWING TRANSFORMERS AND LOW-TENSION TERMINALS (BUS BARS)

The system adopted by the American Foundrymen's Association for sampling carload lots is followed and we sell if necessary on a guaranteed silicon content within 0.25 per cent limits. Our grade card is as follows:

No. 1	"	Low	2.50	2.75
No. 2	"	High	2.25	2.50
No. 2	"	Low	2.00	2.25
No. 3	"	High	1.75	2.00
No. 3	"	Low	1.50	1.75
No. 4	"	High	1.25	1.50
No. 4	"	Low	1.00	1.25

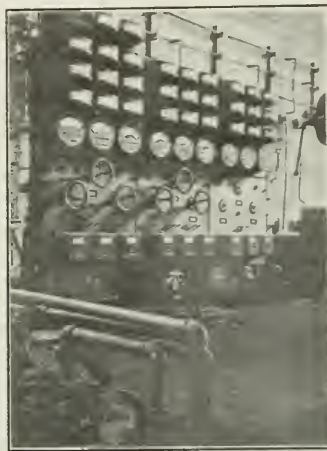


FIG. 10—ELECTRICAL INSTRUMENT PANEL

Though we generally limit our production to the four foundry grades.

I made some experiments during the early operations of this furnace to determine the relative variation of the silicon and carbon in the iron according as the carbon content of the bur-

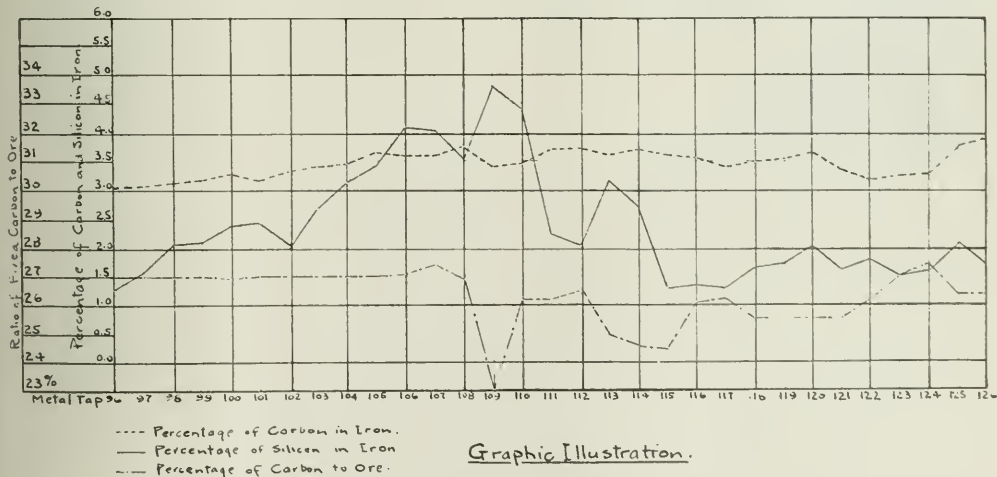


FIG. 11—VARIATION OF SILICON AND CARBON

	Silicon	Per Cent
No. 1 Silvery	4.5	5.0
No. 2	4.0	4.5
No. 1 Soft	3.5	4.00
No. 2	3.0	3.5
No. 1 Foundry High	2.75	3.00

den was increased or decreased. A portion of these results has been plotted graphically in Fig. 11. Each number represents a metal tap made at regular eight-hour intervals. It will be noted that while the silicon responds very readily to changes in the carbon ratio of the burden the carbon in the metal as would be expected, shows little variation. While the

temperature of the metal was not determined the same load was carried throughout so that approximately the same temperature existed after the first thirty-six hours. The silica in the slag is not shown on this plot, but it naturally rose and fell inversely with the silicon in the iron.

Since it is made with charcoal one would not expect the fracture of electric furnace iron to be as coarse as that of coke iron of equal silicon content, but the fracture of our iron is much finer and more uniform than that of charcoal blast furnace iron of the same grade. A very noticeable characteristic is the homogeneity of the fracture and the almost entire absence of segregations and "hard spots." It is also distinguished by its toughness, so that we have to cast it with very deep notches.

This iron has been used for making almost every article which comes within the province of the foundryman, and we have yet to hear a complaint where it has been used intelligently. The softer irons have already won a reputation for themselves as softeners and scrap carriers. Some analyses taken at random of carload lots of high, medium, and low grade iron will illustrate how constant all the other metalloids are except the silicon.

Analyses on a 200-ton lot shipped to a foundry for making steel castings and sold on a guarantee of from 2.75 to 3 per cent silicon and a maximum of 0.04 per cent of sulphur and 0.04 per cent of phosphorus:

Silicon	2.88 per cent
Combined carbon	0.09 " "
Graphite carbon	3.38 " "
Sulphur	0.028 " "
Phosphorus	0.031 " "

Analysis on a 100-ton lot shipped for general foundry purposes—silicon guaranteed from 2.25 to 2.50 per cent:

Silicon	2.42 per cent
Combined carbon	0.27 " "
Graphite carbon	2.94 " "
Sulphur	0.036 " "
Phosphorus	0.023 " "

Analysis on a 100-ton lot shipped to stove works to be mixed with high-phosphorus iron for stove castings. Sold on guarantee of from 1.75 to 2.00 per cent silicon and 0.04 per cent maximum of phosphorus:

Silicon	1.78 per cent
Combined carbon	0.60 " "
Graphite carbon	2.86 " "
Sulphur	0.028 " "
Phosphorus	0.030 " "

Furnace Efficiency and Conclusion

The efficiency of this type of furnace increases in slightly more than direct ratio to the increase in load, and it is of interest to note that as in the case of the electric steel furnaces the faster the furnace is operated the cooler the walls and roof are, and the smoother it works generally. I offer in explanation the theory that the faster smelting takes place the faster the cool charge can descend to protect the arches and walls.

As for technical efficiency, the long and narrow type of furnace is not the equal of the shaft type carrying the same load, though we have kept our power consumption as low as 2200 kw-hours per ton of pig when carrying a load of 3000 kw, but the long and narrow type offers a possibility of building several furnace units onto each other like copper blast furnaces. By this means the radiation losses and electrical losses will be cut down and the efficiency of what are in single-unit furnaces the end-electrodes will be greatly increased.

A bank of three or four units arranged in this way will admit of as simple metallurgical control as a single-unit and will have an efficiency, I believe, equal to a shaft type of furnace carrying the same load.

Further, it can be arranged so that part of the furnace can be frozen up and repaired while the remainder is operating, and for this reason its yearly output should exceed the shaft type.

From the nature of its construction it is capable of being made more nearly fool-proof metallurgically, mechanically, and electrically than the shaft type, and further the electrode consumption is lower, due to the fact that the electrodes penetrate vertically into the charge.

While I hardly agree with the prophecies made by some that electric furnaces for producing pig iron will eventually be competitors of blast furnaces even in the regions where economic conditions make the latter possible, I do feel that where electric power can be obtained cheaply and where coke and freight rates are high and for making superior grades of iron, electric reduction furnaces will enable many large bodies of iron ore to be worked which would otherwise remain idle and that the electric iron furnace both of the shaft type and of the long and narrow type, each in the field best adapted for it, will make steady progress.

Heroult, Shasta County, California.

British Iron and Steel Institute

The autumn meeting of the (British) Iron and Steel Institute will be held in Brussels, on Monday, Tuesday, Wednesday, and Thursday, September 1st to 4th, 1913.

The provisional program of the meeting is as follows:

Sunday, August 31. Secretaries' office open at the Palais des Academies, Brussels, for the registration of members' names and the issue of badges and programs.

Monday, September 1. Opening meeting in the Hall of the Palais des Academies. A selection of papers will be read and discussed. In the afternoon visits will be made to places of interest in Brussels. In the evening a reception will be held by the Burgomaster at the Hotel de Ville.

Tuesday, September 2. Meeting in the morning for the reading and discussion of papers, at the Palais des Academies. Afternoon visits to Colonial Museum and the Parc de Tervueren. In the evening his majesty King Albert will receive the members at the Royal Palace, Brussels.

Wednesday, September 3. A special train will leave in the morning for Ghent where a visit will be paid to the International Exhibition now being held in that city.

Thursday, September 4. Alternative excursions will be made to Liege and Charleroi. The Liege excursion will include a visit to the works of Messrs. John Cockerill & Co., Seraing, to the works of the Ougree-Marihay Company, and to the Copee Coke Oven Gas Plant at Athus-Grivegnée. The excursion to Charleroi will embrace visits to various metallurgical, glass and other works in the vicinity of that town.

The British members will travel on a special train, leaving Charing Cross on the afternoon of Saturday, August 30, at about 2.30 p. m.

Dr. Arthur Cooper is the president, and Mr. G. C. Lloyd, 28 Victoria St., London, S. W., is the secretary of the Institute.

New Platinum-Osmium Alloy.—The limited supply and high price of iridium, which is usually used to produce "hard platinum," has led Dr. Fritz Zimmermann, of Newark, N. J., to experiment with other metals for the purpose of producing a hard alloy of platinum. He has found that osmium makes such an alloy, and that much less osmium than iridium is required to produce alloys of similar properties. From 1/2 per cent to 10 per cent osmium will yield an alloy with platinum which has the hardness, tensile strength and electrical properties obtained when a much larger percentage of iridium is used. The osmium-platinum alloy must be prepared from refined metals. The formula has recently been patented.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in March, 1913, was 60. The total quantity of ore milled during the period was 2,373,273 tons. There were 9,087 stamps in operation with an average duty of 8.8 tons per 24 hours. Tube mills in commission numbered 285. The yield for the month was 799,552 fine ounces of gold. The yield per ton milled was 6.5 dwt. gold.

The Manufacture of Petroleum Products*

By Dr. F. C. Robinson,

Chief Chemist of the Atlantic Refining Co.

It is a real privilege to address you, a body of engineers, on the present subject—the manufacture of petroleum products—because all of these products are or might be used in some engineering operation. I have frequently had occasion to inform my friends that the oil refinery does not manufacture dyes, acetanilid and allied products, nor perfumes. We do not make perfumes.

The first group of products from petroleum is made up of gasolines and naphthas. There is some confusion among the various names, benzine, gasoline, naphtha, etc., but the best practice is to use the word gasoline for any mixture of light hydrocarbons intended for use in any kind of vaporizer, i. e., to be gasified in a gas machine, gasoline torch, gasoline stove or automobile carburetor. Also to confine the word naphtha to mixtures of hydrocarbons intended for some purpose that requires a very good odor, such as the naphtha used by cleaners, varnish makers, soap makers, etc. In this scheme, the word benzine finds no place. Gasolines and naphthas vary in average boiling point according to the use for which they are intended, but all lay between 125 deg. F. and 280 deg. F. In all cases it is essential that they be free from all heavy hydrocarbons that do not evaporate from the hand.

The next group consists of several grades of lamp oil. Lamp oil is a mixture of hydrocarbons whose average boiling point is about 450 deg. F., entirely freed on the one hand from gasoline or naphtha and on the other hand from the heavy hydrocarbons that belong to gas oil and lubricating oil and that would make the oil act badly in the lamp.

The next class is gas oil. While oils of all degrees of volatility have been used, the most economical for the gas maker consists of a mixture of heavy hydrocarbons with an average boiling point of 600 deg. to 650 deg. F. It must be practically free from gasoline and lamp oil on the one hand and from the heavy lubricating oils and asphalt on the other.

The next group is fuel oil. This oil occupies a peculiar position. It must be oil and must not contain gasoline and must be of such a consistency that it can be pumped through pipes and burners, but except for these restrictions one oil is practically as good as another for fuel. The light oils have a slightly higher heat of combustion per pound, but the heavier oils have a slightly higher heat of combustion per gallon. For some purposes, such as oil engines, special oils are required, but, in general, fuel oil is made up of oils that cannot be used for any better purpose.

The next group is that of spindle oils—neutral and paraffin oils. This important group includes hundreds of light lubricating oils designed for use on thousands of different light machines, including gas engines. They must be free from gasoline and lamp oil in order that their flash test shall be high enough to prevent loss by evaporation. The important point to that is that they shall have the proper viscosity for the use intended.

The next group is that of steam cylinder oils, which consist of the heaviest hydrocarbons contained in certain crude oils. In this case also the flash must be such that the oils will not evaporate in a steam cylinder and must have the proper viscosity for the use intended at the temperature of the cylinder.

The next group—paraffin wax—consists of a mixture of hydrocarbons of the paraffin series about $C_{25}H_{52}$ to $C_{35}H_{70}$. The commercial article is rated according to the melting point, which varies from 100 deg. to 135 deg. F.

The next group—vaseline or petrolatum—consists of the higher members of the paraffin series which settle from crude oil mixed and inseparable from some of the oily constituents of the crude. It is marketed as the light-colored material

used in medicine and for toilet purposes or as the dark-colored sticky material used in large quantities by the makers of oiled paper.

The next group—the dust laying oils—consists of petroleum asphalt in solution in oils similar to gas oil. The basic idea in their manufacture is that the solvent will slowly evaporate, leaving the dust particles covered with a sticky adherent film. These oils have proven successful as a cheap means of laying dust.

The next group—the road binders—consists of petroleum asphalt properly fluxed with heavy petroleum oils that will not evaporate and of such qualities that they will bind the road materials together both in summer and winter.

The next group—coke—contains but one member. This material being almost entirely free from ash, is used very extensively by makers of electric carbons.

1. These are the desired products. Now let us look at the crude oils from which they are obtained.

2. There are about as many varieties of crude oil as there are oil fields, but the refiner recognizes three distinct types, because each type must be handled by different methods. Viz., (1) The paraffin base crude similar to that found in Pennsyl-

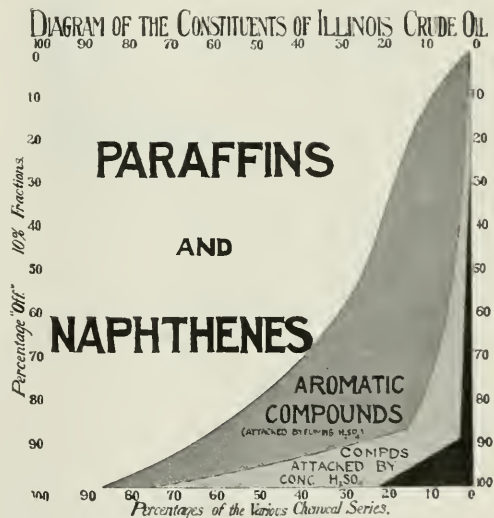


FIG. 1.—DIAGRAM OF THE CONSTITUENTS OF ILLINOIS CRUDE OIL

vania and West Virginia and being essentially light colored crudes containing paraffin. (2) Asphalt base crudes similar to those found in Texas and California and being essentially black and containing no paraffin. (3) Mixed base crudes similar to those found from Ohio to Oklahoma and being essentially mixtures of paraffin and asphalt base crudes.

3. In order to obtain some idea of the chemical and physical nature of the crude oil let us imagine a sample of mixed base crude brought into the laboratory for a thorough examination. The chemist would probably distill the sample in a vacuum or in some similar manner in order to avoid destructive distillation and would save the various fractions separate. He would not distill off more than 90 per cent because the heaviest 10 per cent cannot be distilled without breaking it down into simpler molecules. He will then start to examine the various fractions and will keep a record similar to Fig. 1. The horizontal lines indicate the ten fractions.

The first fraction will be a light mobile mixture of hydrocarbons whose average boiling point is about 227 deg. F. The second is a slightly darker and slightly less mobile mixture of hydrocarbons whose average boiling point is about 295 deg. F. The third cut again darker, heavier and less mobile, boiling

*A paper read before the Engineers' Club of Philadelphia on December 21, 1912. From the April, 1913, issue of the *Proceedings of the Engineers' Club of Philadelphia*.

point 369 deg. F. The fourth cut still heavier and 460 deg. F. boiling point. The fifth cut is about 530 deg. F. boiling point. The remaining cuts are increasingly heavier, more viscous and darker in color and the residue in the still is a soft pitch.

The chemist now asks what these ten fractions are chemically and he recognizes four groups of compounds in each fraction that the refiner may have to isolate or remove. First the coloring matter indicated in the diagram by the black area.

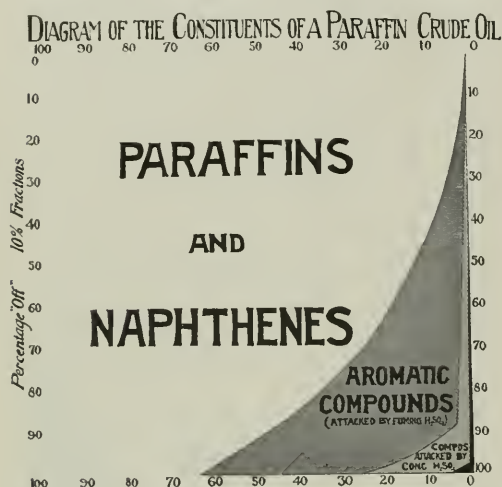


FIG. 2—DIAGRAM OF THE CONSTITUENTS OF A PARAFFIN CRUDE OIL

If he can isolate this group by bone black or fuller's earth. When isolated in a pure state it is a jet black brittle compound which is very similar to the purest asphalt.

A second important class of compounds is the material soluble in strong sulphuric acid. The low boiling members of this group represent the odor-bearing compounds of the crude, while the higher members are rich in sulphur, and are easily oxidized. The refiner frequently has to remove a portion of this class.

A third group is that of aromatic hydrocarbons, benzol, naphthalene, anthracene, etc., which may be removed by agitating the oil with fuming sulphuric acid.

The remainder of the crude oil unattacked by fuming sulphuric acid is made up of the naphthenes and paraffin series. You will notice that the amount of coloring is very small in the low boiling fractions, but very high in the high boiling fractions, and conversely that the amount of paraffins and naphthenes in the low boiling fractions is very large while in the residue it is very small.

Looking now at similar diagrams for a paraffin base crude (Fig. 2) and an asphalt base crude (Fig. 3); you will notice first that the paraffin base crude contains very little, the asphalt base crude very much color or asphalt. Also that the paraffin base crude contains very much and the asphalt base crude very little paraffin and naphthenes. Also that the paraffin base crude contains very little and the asphalt base crude very much of the aromatic group. Such information as these charts set forth enable the refiner to decide what products he can make and indicate in a general way the methods he must follow. For example the charts would show: (1) The percentage of gasoline that the crude contains; (2) the percentage of lamp oil; (3) the percentage of lubricating oil, etc. However, you will please understand that this is not a practical assay of crude oil. It is merely an attempt to give you a chemical picture, so to speak, showing the distribution of the four important groups that the refiner has to deal with; also that each group consists of thousands of individual hydrocarbons that are never isolated even in the laboratory.

4. Now, starting with crude oil, it is the task of the refiner to isolate the commercial products that I have shown you. You will see the processes outlined on these two charts, and I wish to make especially clear that they represent two distinctly different methods of refining, so much so that one refiner may decide to use one of them to the exclusion of the other, or he may decide to use both of them.

He will be guided in this decision by his local conditions. If it be his desire to produce the maximum amount of gasoline and lamp oil he will use the method marked "Cracking Distillation." (Fig. 4.) If, on the other hand, he wishes to produce the maximum yield of the heavy lubricating oils and petroleum asphalts, he will use the method marked "Fractional Distillation," (Fig. 5), which means that he will simply separate from the crude oil the various fractions which compose it, while the refiner who uses the cracking process actually breaks down these heavy fractions by destructive distillation in a manner similar to the production of benzol and gas by the destructive distillation of coal.

The first step in the cracking process is the cracking distillation. The crude oil is pumped into stills containing 500 to 1000 barrels which consist simply of horizontal steel cylinders made of sheets of half-inch boiler steel riveted together and provided with manholes on top and ends; with pipe for pumping oil into stills; with vapor line for conducting the distilled vapors to a condenser; with safety valve; with combustion chamber underneath; with fractional air condensers and with water condenser, and with pipe for conducting away the gases involved during the distillation.

Such a still is nearly filled with, *e. g.*, mid continent crude oil and fires are lighted. When the temperature of the oil in the still has reached 175 deg. to 200 deg. F. some gases, consisting largely of butane and pentane, are given off and presently the highest naphtha starts to distill over. The firing is continued; the temperature in the still becomes gradually

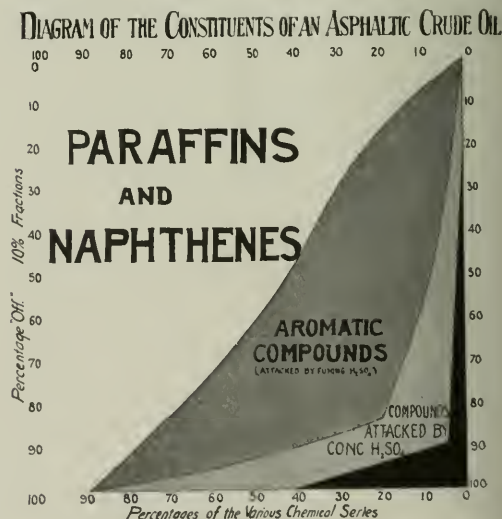


FIG. 3—DIAGRAM OF THE CONSTITUENTS OF AN ASPHALTIC CRUDE OIL

higher; the distillate becomes gradually heavier until the temperature in the still reaches about 325 deg. F., at which point about 6 per cent or 8 per cent of crude naphtha (200 deg. F. boiling point) has distilled over. This is set aside as crude naphtha.

The distillation is continued until the temperature in the still has reached about 475 deg. F. for crude heavy naphtha and represents 13 per cent to 15 per cent of the crude and has

an average boiling point of about 300 deg. F. The distillation is then continued until the temperature in the still has reached about 625 deg. F. for natural lamp distillate, which represents about 16 per cent to 18 per cent and has an average boiling point of about 450 deg. F.

paraffin wax and the line of lubricating oils called paraffin oils. It is no longer desirable to carry on a cracking distillation because this would result in the destruction of the valuable products desired. The distillation is continued in such a way as to avoid cracking as much as possible (is distilled fast) either in

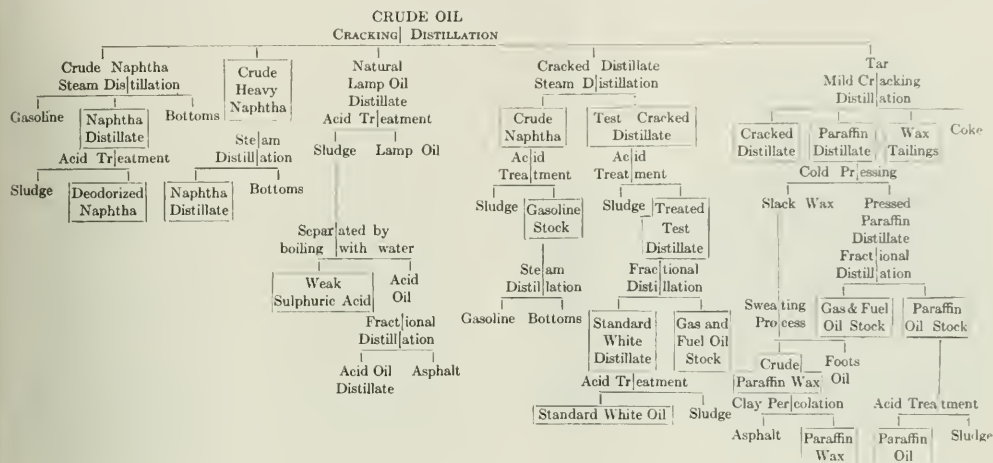


FIG. 4—DIAGRAM OF CRACKING DISTILLATION OF CRUDE OIL

When the still has reached this temperature, cracking or destructive distillation sets in. The fires are slackened in order to distill very slowly and this slow distillation is continued until the temperature in the still reaches 675 deg. to 700 deg. F., producing a distillate with an average boiling point of about 550 deg. F., but containing some gasoline, some lamp oil and much heavier oil which I have designated as gas and fuel oil stock. The yield of this oil is about 20 per cent.

You will notice that this cracking distillation is very different

the same still or more commonly in separate smaller stills called tar stills.

This tar still distillation is carried on very rapidly in order to produce the maximum yield of paraffin distillate (about 22 per cent). In addition to the paraffin distillate, there is also produced by destructive distillation about 15 per cent of cracked distillate. At the end of the distillation the stream becomes so heavy that it will sink in water and is then known as wax tailings, which amounts to about 1 per cent of the

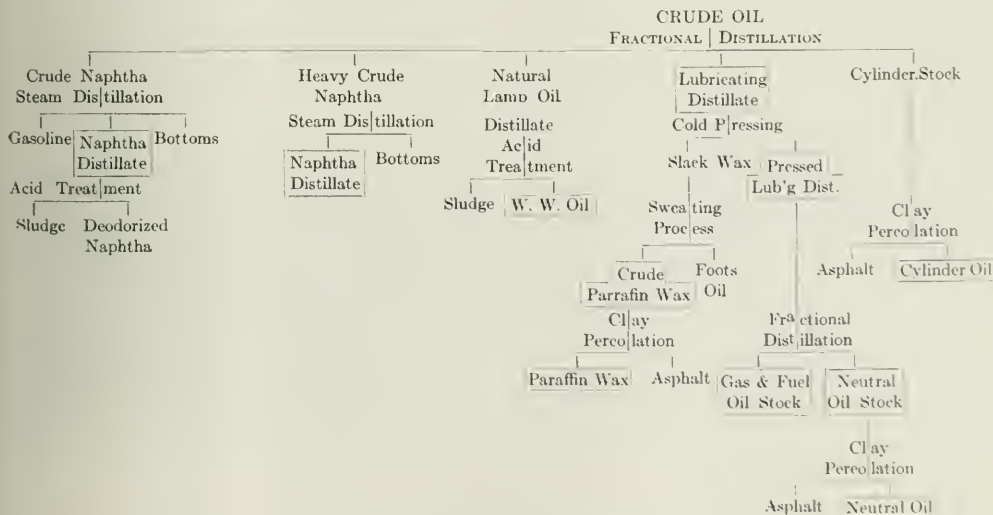


FIG. 5—DIAGRAM OF FRACTIONAL DISTILLATION

from an ordinary fractional distillation; heavy molecules have been broken down into lighter ones by submitting them to temperatures at which they are unstable.

There yet remains in the still a heavy black tar representing about 42 per cent of the crude oil. This is the source of

crude oil. When the distillation stops there remains in the still nothing but coke, amounting to about 4 per cent of the crude oil.

Now taking up the various fractions; first the crude naphtha. This is again distilled; first, in order to separate it into

the various gasolines and naphthas that compose it and secondly to separate it from the small amount of bottoms or light lamp oil that it contains. This is done in a still which is heated by steam usually by injecting live steam directly into the gasoline.

When the distillation starts, some gas is given off, then the lightest distillate appears at the trap usually about 90 deg.

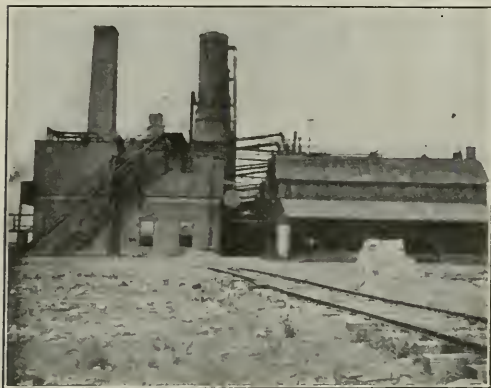


FIG. 6—CRUDE STILL

Beaume gravity. The distillate gradually gets heavier until all the gasoline has distilled off. The receiver is then changed and the naphtha distillate is separated. At this point about 90 per cent has distilled off leaving a bottom about 10 per cent. This bottom is essentially lamp oil and is used as such. The heavy crude naphtha is handled in the same manner except that it contains little or no gasoline and contains about 50 per cent of bottom or lamp oil. The cracked distillate is also distilled with steam to remove about 4 per cent of crude naphtha.

Up to this point all the operations have been different types of distillation. The next step in handling the naphtha distillate from both sources, the lamp oil distillate and the crude naphtha from cracked distillate and the test cracked distillate, is the acid treatment. You have seen in the crude diagrams that all the fractions contain a certain percentage of material attackable by sulphuric acid so that this reagent affords a convenient means for removing color and odor from the remainder of the hydrocarbon distillate.

In practice the naphtha distillates are agitated with about 5 per cent by volume, the lamp oil distillates with about 1.5 per cent of sulphuric acid, oil of vitriol, for about a half hour. The color and odor bearing compounds combine with the acid producing a heavy black viscous mass called acid sludge, which settles to the bottom of the vessel. The sludge is drawn off and the oil washed with water and alkali to remove all traces of acid and is then ready for the market.

The sludge from all acid treatments is separated into unstable products. This is accomplished by boiling it with water, which results in the dilution of the acid and renders it incapable of holding in solution the impurities.

The weak acid (30 deg. to 50 deg. Beaume) settles to the bottom, is drawn off and reconcentrated. The upper layer consisting of the impurities is known as acid oil. This acid oil is separated by fractional distillation into a light distillate which consists of all the evil odors that the original distillate contained and a residue consisting of the asphaltic compounds that were removed by the acid.

You now have before you all the processes used in the manufacture of naphtha and lamp oil.

The next general subject is the handling of the paraffin distillate, which is the direct source of the paraffin wax and all of the paraffin oils. The first step is the process of cold press-

ing. The distillate is first cooled to 20 deg. to 30 deg. F. by pumping through pipes surrounded by cold brine, thereby causing the paraffin wax (amounting to about 10 per cent of the distillate) to solidify. This solid 10 per cent mixed with the 90 per cent liquid oil, forms a soft mush which is pumped through a filter press. That which stays in the press is called the slack wax and amounts to about 20 per cent of the paraffin distillate. The 80 per cent that goes through is called pressed distillate.

The slack wax, consisting of about equal parts of oil and wax, is then put through a process peculiar to the oil business, known as the sweating process. It consists of cooling the mixture until it has become a solid cake and then very gradually warming it. The crystals of the paraffin form a network through which the oil is distributed and when the mass is warmed the oil sweats out and drips away. It always carries with it some wax in solution, but the final result is that the oil all sweats out leaving the paraffin wax in a fairly pure state.

This sweating process separates the slack wax into crude paraffin wax and what is known as Foots oil. The latter still contains much paraffin, which is removed by putting it again through either the cold pressing or sweating process.

The crude paraffin wax is then put through another process that is peculiar to the oil business, that of clay or bone black percolation for the purpose of removing asphaltic coloring matter and thereby changing the crude paraffin to refined colorless paraffin. The clay used for this purpose has properties similar to those of bone black, *i. e.*, it absorbs and retains tarry and asphaltic compounds. It is found in Florida and Georgia where it is mined, roasted, broken up and sifted to separate grains such as you see here. It is very porous and light, weighing only about 2.3 as much as water. This clay, or fuller's earth, as it is commonly called, is put into large upright cylinders holding 10 or 20 tons and provided with a finely perforated bottom. The crude wax is melted and poured on top of the clay. It trickles down through the clay bed and passes through the perforated bottom.

The first drippings from such a filter bed are absolutely colorless, but as the filtration progresses, the color becomes more and more like crude wax. A ton of clay yields 5 or 6 tons of first-quality paraffin wax. The amount of asphalt or coloring



FIG. 7—TAR STILL

matter retained by the clay is exceedingly small and is removed by burning the clay in a cement kiln of the usual type, thus regenerating the clay for subsequent use.

I have explained that the paraffin distillate is the source of paraffin wax and light lubricating oils and have explained the various steps in the preparation of paraffin. The light lubricating oils are made from the filtrate from the cold presses—the pressed paraffin distillate—by putting it through

the process of fractional distillation, thereby separating it into a distillate of light oils that go to make up the gas and fuel oil stock and a residue in the still called paraffin oil stock, representing from 15 per cent to 50 per cent of the charge of the still depending on the quality desired.

This type of distillation is also peculiar to the oil business. The desired product is a heavy oil so that all cracking must be avoided in order to produce the maximum yield of this oil.



FIG. 8—AGITATORS

The result is accomplished by using the very important process of distillation with bottom steam or fractional distillation. A still is charged with the pressed paraffin distillate and fires are lighted. The temperature in the still rises and when it has reached about 400 deg. F., the distillation begins.

Shortly after this the live steam is injected into the oil in distillation through perforated pipes placed near the bottom of the still. No water accumulates in the still because the temperature is so high. The steam passes upward through the oil as an inert gas and passes through the condenser with the oil vapors and is condensed there with the oil. The effect of this current of steam through the oil is exactly that of a vacuum distillation, i. e., it lowers the boiling point of the oil in distillation and allows a heavy oil to be distilled at temperatures below the temperature of destructive distillation. You will notice that cracking sets in at about 630 deg. F. Without the use of steam, the distillation in question would require that the still be heated to about 750 deg. F. This would result in the destruction of the desired oil. With steam it can be carried on with a maximum temperature of 600 deg. F., thus entirely avoiding destructive distillation.

The paraffin oil stock is a dark colored unattractive looking material which is transformed into the valuable paraffin oils of commerce by treating with sulphuric acid in the manner already described. The treating loss in this case is from 10 per cent to 30 per cent. The whole cracking process described thus far is designed to produce the maximum yield of gasoline and lamp oil from crudes containing asphalt.

We will now turn to the method of refining the light colored non-asphaltic crude oils from which the valuable cylinder oils may be made. The object in this case is to avoid all destructive distillation in order to produce the maximum yield of the very heavy lubricating oils. The still is charged with the crude oil, fires are lighted, the crude naphtha is distilled off as in the other distillations, but when the temperature is well above the boiling point of water, steam is injected into the oil as before described.

Under these conditions the crude naphtha has distilled off when the temperature in the still has reached about 280 deg. F., while without steam the still temperature was about 375 deg. F. The yield from this kind of crude is about 13 per cent.

The heating is continued, more and more steam being injected, the distillate becoming heavier and heavier until the heavy crude naphtha has distilled off. At this point the temperature in the still has reached about 330 deg. F., while without steam at this point the temperature was 475 deg. F. The yield of this oil is about 13 per cent.

The distillation is still continued until the natural lamp oil distillate has distilled off. At this point the distillate in the still is only 500 deg. F., while without steam it was 630 deg. F.

Now instead of trying to increase cracking by slowing the rate of distillation, the amount of steam is increased and the distillation is carried on as fast as possible to avoid cracking. This is continued until the lubricating distillate, amounting to about 28 per cent of the crude has distilled over. At this point the temperature in the still has reached about 620 deg. F., and the distillation is stopped, leaving the cylinder oil stock in the still.

You will recall that in the cracking process the temperature in the still reached 850 deg. F., and yielded a residue in the still of solid coke. This whole distillation has been conducted in such a manner as to produce the best quality and largest yield of this residual oil, the source of all steam cylinder oils.

The various fractions from the stills are put through exactly the same processes as the corresponding fractions from the cracking distillation, except the lubricating distillate. The crude naphtha and the crude heavy naphtha are fractioned and some of the distillates treated with acid.

The lamp oil distillate is treated with acid as before described. The lubricating distillate, amounting to about 28 per cent of the crude, is put through the same process for the removal of wax and gas oil that the paraffin oil was put through, the only difference being that the resulting lubricating stocks are not usually treated with sulphuric acid, but are brought to the desired color by percolating through fuller's earth to produce the neutral spindle oils of commerce.

The cylinder stock remaining in the still, and amounting to 8 per cent to 13 per cent of the crude oil, is ready for shipment as unfiltered cylinder oil. Or it may be percolated through clay to produce the filtered oils of commerce. In order to show the wonderful effects of fuller's earth on the color of oils, I have selected a series of samples showing (1) cylinder stock that was pumped into the filter, (2) the first oil that

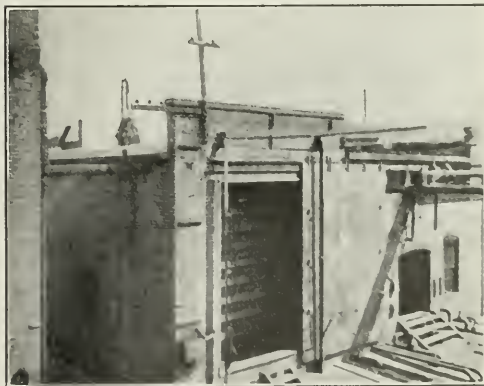


FIG. 9—SWEATER

percolated through, (3) a sample when the percolation was half completed, and (4) a sample at the end of the percolation. At this point the clay must be restored.

You will notice that I have described the cracking distillation of a mixed base crude and the fractional distillation of a paraffin base crude. Now, what would result if the mixed base crude had been put through the process of fractional distillation? The crude naphtha would, of course, distill off, then the

crude heavy naphtha, then the natural lamp oil, then a distillate containing wax and lubricating oil corresponding to the lubricating distillate, leaving in the still a residue consisting of a soft pitch and amounting to about 15 per cent of the crude oil. The maximum temperature in the still would be about 630 deg. F. This residue has the property of being vulcanized by the action of air at about 500 deg. F., or by sulphur at about 400 deg. to 450 deg. F. This process is carried on in a still and consists in heating the residue to the desired temperature and pumping air through the mass for one to two days, depending on the hardness desired. The resulting hard pitch is the petroleum road binder of commerce.

* * *

In the discussion which followed Mr. Price asked as to the probable future supply of fuel oil or gasoline. Mr. Irish said that "the supply of fuel and gas oils must necessarily be a function of the crude oil production of the world. While the amount of crude oil produced has within the past few years been greater than at any other time since the discovery of petroleum, and consequently indicates an ever increasing supply, we are at present and have been for the last two or three years, suffering from a falling off in our production. We are informed that the crude oil stocks in the United States, east of the Rocky Mountains, are decreasing at the rate of about 40,000 barrels a day.

"That means to us that the supply has fallen off and the demand has increased, until the demand has overtaken the production, and we are drawing on accumulated stocks, and so long as that continues, exhaustion of the supply of gas and fuel oil will, of course, come nearer.

"But the supply of those oils has been supplemented by the Mexican crude. . . .

"The Mexican crude is new to the refiner, and it would be unsafe and unwise to predict what can be made from it. We know it has an asphalt base, and is limited in possibilities to the production of those things Mr. Robinson has described as being peculiar to the asphalt base crude oil. It means that a large proportion of such crude oil will find its way into the fuel oil market as rapidly as it can be brought into this country."

In reply to a question by Mr. J. C. Parker as to the length of pipe lines Mr. Irish replied that the pipe line system is complete from Oklahoma to the Atlantic coast, so that Oklahoma crudes are distributed in New York and Philadelphia.

Temperature Conversion Table

In our March issue, 1910 (Vol. VIII, p. 123) we published a temperature conversion table by Dr. Leonard Waldo for converting Centigrade into Fahrenheit degrees.

In the Bulletin of the American Institute of Mining Engineers, of April, 1913, Dr. Waldo publishes a second table for conversion from Fahrenheit to Centigrade degrees. This is herewith reproduced. Like his former table it is based on the arrangement well known from logarithmic tables. Besides convenience it has the advantage of exactness.

As the table will be found almost self-explanatory, the following examples will be sufficient to explain its use: -246.0° F. = -151.11° C. - 3.33° C. = -154.44° C. 3762° F. = 2071.11° C. + 1.11° C. = 2072.22° C. 2423.5° F. = 1326.666° C. + 1.666° C. + 0.277° C. = 1328.609° C.

In his paper in the Bulletin of the American Institute of Mining Engineers Dr. Waldo remarks that for mental calculation the easiest rule to convert Centigrade into Fahrenheit is to add 32° to twice the Centigrade reading less one-tenth of the product:

Thus 1000° C. = 2000° - 200° + 32° = 1832° F., and to convert Fahrenheit to Centigrade, subtract 32° from the Fahrenheit reading, multiply by 10 and divide by 2 and 9 successively:

Thus 1000° F. = (1000 - 32) × 10 ÷ 2 ÷ 9 = 537.7° C.

Another and often very convenient method is by use of a graphic construction. A diagram illustrating this graphical method is given in the paper

Data on figure indicate recurring decimals.

F.°	0	10	20	30	40	50	60	70	80	90	F.°	0	10	20	30	40	50	60	70	80	90
C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.
-400	-240	-245.0	-251.1	-258.0	-265.6	-273.9	-282.8	-292.2	-302.1	-312.6	-323.7	-335.4	-347.7	-360.6	-374.1	-388.2	-402.9	-418.2	-434.1	-450.6	-467.7
-300	-184	-190.0	-196.1	-202.0	-208.6	-215.9	-223.7	-232.0	-240.8	-250.1	-259.9	-270.2	-281.0	-292.3	-304.1	-316.4	-329.2	-342.5	-356.3	-370.6	-385.4
-200	-128	-134.4	-141.0	-148.0	-155.6	-163.9	-172.8	-182.2	-192.1	-202.6	-213.7	-225.4	-237.7	-250.6	-264.1	-278.2	-292.9	-308.2	-324.1	-340.6	-357.7
-100	-73.3	-78.8	-84.4	-90.0	-95.6	-101.1	-106.6	-112.2	-117.7	-123.3	-128.9	-134.5	-140.1	-145.7	-151.3	-156.9	-162.5	-168.1	-173.7	-179.3	-185.0
0	-17.7	-23.3	-28.9	-34.4	-40.0	-45.6	-51.1	-56.7	-62.2	-67.8	-73.3	-78.9	-84.4	-90.0	-95.6	-101.1	-106.6	-112.2	-117.7	-123.3	-128.9
100	17.7	23.3	28.9	34.4	40.0	45.6	51.1	56.7	62.2	67.8	73.3	78.9	84.4	90.0	95.6	101.1	106.6	112.2	117.7	123.3	128.9
200	84.4	90.0	95.6	101.1	106.6	112.2	117.7	123.3	128.9	134.5	140.1	145.7	151.3	156.9	162.5	168.1	173.7	179.3	185.0	190.6	196.1
300	140.1	145.7	151.3	156.9	162.5	168.1	173.7	179.3	185.0	190.6	196.1	201.7	207.3	212.9	218.5	224.1	229.7	235.3	240.9	246.5	252.1
400	204.4	210.0	215.6	221.2	226.8	232.4	238.0	243.6	249.2	254.8	260.4	266.0	271.6	277.2	282.8	288.4	294.0	299.6	305.2	310.8	316.4
500	268.0	273.6	279.2	284.8	290.4	296.0	301.6	307.2	312.8	318.4	324.0	329.6	335.2	340.8	346.4	352.0	357.6	363.2	368.8	374.4	380.0
600	332.0	337.6	343.2	348.8	354.4	360.0	365.6	371.2	376.8	382.4	388.0	393.6	399.2	404.8	410.4	416.0	421.6	427.2	432.8	438.4	444.0
700	396.0	401.6	407.2	412.8	418.4	424.0	429.6	435.2	440.8	446.4	452.0	457.6	463.2	468.8	474.4	480.0	485.6	491.2	496.8	502.4	508.0
800	460.0	465.6	471.2	476.8	482.4	488.0	493.6	499.2	504.8	510.4	516.0	521.6	527.2	532.8	538.4	544.0	549.6	555.2	560.8	566.4	572.0
900	524.0	529.6	535.2	540.8	546.4	552.0	557.6	563.2	568.8	574.4	580.0	585.6	591.2	596.8	602.4	608.0	613.6	619.2	624.8	630.4	636.0
1000	588.0	593.6	599.2	604.8	610.4	616.0	621.6	627.2	632.8	638.4	644.0	649.6	655.2	660.8	666.4	672.0	677.6	683.2	688.8	694.4	700.0
1100	652.0	657.6	663.2	668.8	674.4	680.0	685.6	691.2	696.8	702.4	708.0	713.6	719.2	724.8	730.4	736.0	741.6	747.2	752.8	758.4	764.0
1200	716.0	721.6	727.2	732.8	738.4	744.0	749.6	755.2	760.8	766.4	772.0	777.6	783.2	788.8	794.4	800.0	805.6	811.2	816.8	822.4	828.0
1300	780.0	785.6	791.2	796.8	802.4	808.0	813.6	819.2	824.8	830.4	836.0	841.6	847.2	852.8	858.4	864.0	869.6	875.2	880.8	886.4	892.0
1400	844.0	849.6	855.2	860.8	866.4	872.0	877.6	883.2	888.8	894.4	900.0	905.6	911.2	916.8	922.4	928.0	933.6	939.2	944.8	950.4	956.0
1500	908.0	913.6	919.2	924.8	930.4	936.0	941.6	947.2	952.8	958.4	964.0	969.6	975.2	980.8	986.4	992.0	997.6	1003.2	1008.8	1014.4	1020.0
1600	972.0	977.6	983.2	988.8	994.4	1000.0	1005.6	1011.2	1016.8	1022.4	1028.0	1033.6	1039.2	1044.8	1050.4	1056.0	1061.6	1067.2	1072.8	1078.4	1084.0
1700	1036.0	1041.6	1047.2	1052.8	1058.4	1064.0	1069.6	1075.2	1080.8	1086.4	1092.0	1097.6	1103.2	1108.8	1114.4	1120.0	1125.6	1131.2	1136.8	1142.4	1148.0
1800	1100.0	1105.6	1111.2	1116.8	1122.4	1128.0	1133.6	1139.2	1144.8	1150.4	1156.0	1161.6	1167.2	1172.8	1178.4	1184.0	1189.6	1195.2	1200.8	1206.4	1212.0
1900	1164.0	1169.6	1175.2	1180.8	1186.4	1192.0	1197.6	1203.2	1208.8	1214.4	1220.0	1225.6	1231.2	1236.8	1242.4	1248.0	1253.6	1259.2	1264.8	1270.4	1276.0
2000	1228.0	1233.6	1239.2	1244.8	1250.4	1256.0	1261.6	1267.2	1272.8	1278.4	1284.0	1289.6	1295.2	1300.8	1306.4	1312.0	1317.6	1323.2	1328.8	1334.4	1340.0
2100	1292.0	1297.6	1303.2	1308.8	1314.4	1320.0	1325.6	1331.2	1336.8	1342.4	1348.0	1353.6	1359.2	1364.8	1370.4	1376.0	1381.6	1387.2	1392.8	1398.4	1404.0
2200	1356.0	1361.6	1367.2	1372.8	1378.4	1384.0	1389.6	1395.2	1400.8	1406.4	1412.0	1417.6	1423.2	1428.8	1434.4	1440.0	1445.6	1451.2	1456.8	1462.4	1468.0
2300	1420.0	1425.6	1431.2	1436.8	1442.4	1448.0	1453.6	1459.2	1464.8	1470.4	1476.0	1481.6	1487.2	1492.8	1498.4	1504.0	1509.6	1515.2	1520.8	1526.4	1532.0
2400	1484.0	1489.6	1495.2	1500.8	1506.4	1512.0	1517.6	1523.2	1528.8	1534.4	1540.0	1545.6	1551.2	1556.8	1562.4	1568.0	1573.6	1579.2	1584.8	1590.4	1596.0
2500	1548.0	1553.6	1559.2	1564.8	1570.4	1576.0	1581.6	1587.2	1592.8	1598.4	1604.0	1609.6	1615.2	1620.8	1626.4	1632.0	1637.6	1643.2	1648.8	1654.4	1660.0
2600	1612.0	1617.6	1623.2	1628.8	1634.4	1640.0	1645.6	1651.2	1656.8	1662.4	1668.0	1673.6	1679.2	1684.8	1690.4	1696.0	1701.6	1707.2	1712.8	1718.4	1724.0
2700	1676.0	1681.6	1687.2	1692.8	1698.4	1704.0	1709.6	1715.2	1720.8	1726.4	1732.0	1737.6	1743.2	1748.8	1754.4	1760.0	1765.6	1771.2	1776.8	1782.4	1788.0
2800	1740.0	1745.6	1751.2	1756.8	1762.4	1768.0	1773.6	1779.2	1784.8	1790.4	1796.0	1801.6	1807.2	1812.8	1818.4	1824.0	1829.6	1835.2	1840.8	1846.4	1852.0
2900	1804.0	1809.6	1815.2	1820.8	1826.4	1832.0	1837.6	1843.2	1848.8	1854.4	1860.0	1865.6	1871.2	1876.8	1882.4	1888.0	1893.6	1899.2	1904.8	1910.4	1916.0
3000	1868.0	1873.6	1879.2	1884.8	1890.4	1896.0	1901.6	1907.2	1912.8	1918.4	1924.0	1929.6	1935.2	1940.8	1946.4	1952.0	1957.6	1963.2	1968.8	1974.4	1980.0
3100	1932.0	1937.6	1943.2	1948.8	1954.4	1960.0	1965.6	1971.2	1976.8	1982.4	1988.0	1993.6	1999.2	2004.8	2010.4	2016.0	2021.6	2027.2	2032.8	2038.4	2044.0
3200	1996.0	2001.6	2007.2	2012.8	2018.4	2024.0	2029.6	2035.2	2040.8	2046.4	2052.0	2057.6	2063.2	2068.8	2074.4	2080.0	2085.6	2091.2	2096.8	2102.4	2108.0
3300	2060.0	2065.6	2071.2	2076.8	2082.4	2088.0	2093.6	2099.2	2104.8	2110.4	2116.0	2121.6	2127.2	2132.8	2138.4	2144.0	2149.6	2155.2	2160.8	2166.4	2172.0
3400	2124.0	2129.6	2135.2	2140.8	2146.4	2152.0	2157.6	2163.2	2168.8	2174.4	2180.0	2185.6	2191.2	2196.8	2202.4	2208.0	2213.6	2219.2	2224.8	2230.4	2236.0
3500	2188.0	2193.6	2199.2	2204.8	2210.4	2216.0	2221.6	2227.2	2232.8	2238.4	2244.0	2249.6	2255.2	2260.8	2266.4	2272.0	2277.6	2283.2	2288.8	2294.4	2300.0
3600	2252.0	2257.6	2263.2	2268.8	2274.4	2280.0	2285.6	2291.2	2296.8	2302.4	2308.0	2313.6	2319.2	2324.8	2330.4	2336.0	2341.6	2347.2	2352.8	2358.4	2364.0
3700	2316.0	2321.6	2327.2	2332.8	2338.4	2344.0	2349.6	2355.2	2360.8	2366.4	2372.0	2377.6	2383.2	2388.8	2394.4	2400.0	2405.6	2411.2	2416.8	2422.4	2428.0
3800	2380.0	2385.6	2391.2	2396.8	2402.4	2408.0	2413.6	2419.2	2424.8	2430.4	2436.0	2441.6	2447.2	2452.8	2458.4	2464.0	2469.6	2475.2	2480.8	2486.4	2492.0
3900	2444.0	2449.6	2455.2	2460.8	2466.4	2472.0	2477.6	2483.2	2488.8	2494.4	2500.0	2505.6	2511.2	2516.8	2522.4	2528.0	2533.6	2539.2	2544.8	2550.4	2556.0
4000	2508.0	2513.6	2519.2	2524.8	2530.4	2536.0	2541.6	2547.2	2552.8	2558.4	2564.0	2569.6	2575.2	2580.8	2586.4	2592.0	2597.6	2603.2	2608.8	2614.4	2620.0
4100	2572.0	2577.6	2583.2	2588.8	2594.4	2600.0	2605.6	2611.2	2616.8	2622.4	2628.0	2633.6	2639.2	2644.8	2650.4	2656.0	2661.6	2667.2	2672.8	2678.4	2684.0
4200	2636.0	2641.6	2647.2	2652.8	2658.4	2664.0	2669.6	2675.2	2680.8	2686.4	2692.0	2697.6	2703.2	2708.8	2714.4	2720.0	2725.6	2731.2	2736.8	2742.4	2748.0
4300	2700.0	2705.6	2711.2	2716.8	2722.4	2728.0	2733.6	2739.2	2744.8	2750.4	2756.0	2761.6	2767.2	2772.8	2778.4	2784.0	2789.6	2795.2	2800.8	2806.4	2812.0
4400	2764.0	2769.6	2775.2	2780.8	2786.4	2792.0	2797.6	2803.2	2808.8	2814.4	2820.0	2825.6	2831.2	2836.8	2842.4	2848.0	2853.6	2859.2	2864.8	2870.4	2876.0
4500	2828.0	2833.6	2839.2	2844.8	2850.4	2856.0	2861.6	2867.2	2872.8	2878.4	2884.0	2889.6	2895.2	2900.8	2906.4	2912.0	2917.6	2923.2	2928.8	2934.4	2940.0
4600	2892.0	2897.6	2903.2	2908.8	2914.4	2920.0	2925.6	2931.2	2936.8	2942.4	2948.0	2953.6	2959.2	2964.8	2970.4	2976.0	2981.6	2987.2	2992.8	2998.4	3004.0
4700	2956.0	2961.6	2967.2	2972.8	2978.4	2984.0	2989.6	2995.2	3000.8	3006.4	3012.0	3017.6	3023								

Cyanide Practice in the Black Hills, South Dakota—I

By H. C. Parmelee.

The small area covered by the Black Hills in southwestern South Dakota holds an important place in the annals of metallurgy. Its development has followed the lines so common to many of the older mining districts, beginning with the simple processes of placing for free gold in the stream beds, continuing through the list of amalgamation, smelting and chlorination, and finally gaining prominence by the successful and extensive use of cyanidation which is today the prevalent method of gold recovery. As in some other western mining districts, also, the high-grade ores which originally attracted the miner have been exhausted, and the stability of the district now rests on the possibility of treating at a profit the large tonnages of more refractory, low-grade ores which remain. Metallurgical success in the Black Hills today depends on good business management combined with technical skill. The best known example is the Homestake Mining Company, the success of which has been a stimulus for many smaller but successful ventures, as well as the basis for some flagrant, illegitimate promotions that have done immeasurable harm. In these notes the practice at the Homestake mills will not be touched upon, as the subject has recently been thoroughly covered by Messrs. Clark and Sharwood,¹ metallurgist and chemist, respectively, for the company. The gist of their article has already appeared in this journal.²

General Conditions.

The majority of the ores now treated in the Black Hills range in value from \$2 to \$10 per ton. Some higher grade ores are still mined, but only in small quantities and incidentally with those of lower grade. In most cases constant assaying of daily mine samples must be resorted to in order to determine whether the ore may be treated at a profit, for it is not possible to be guided wholly by the appearance of the ore, nor to assume that all parts of the same deposit are uniformly valuable. Silver occurs in practically all the ores; sometimes in greater quantity, but usually in less value, than gold.

For a detailed study of the different kinds of ore, the reader is referred to Irving.³ The ores now being treated are all highly silicious, containing from 70 per cent to 85 per cent silica. They vary in hardness from the "hard blue" ore, which is dense, unoxidized and fine-grained, down through the coarse and porous quartzites, to the soft, oxidized shales. The unoxidized ores contain from 2 per cent to 10 per cent sulphur in the form of pyrite, but the oxidized ores are free from this element and have the characteristic red color of iron oxide. Tungsten in the form of wolframite has been found in commercial quantities, some high-grade gold-tungsten ore having been shipped from the Wasp No. 2. Traces of tungsten throughout some of the quartzites are not uncommon. Stibnite also has been identified, while traces of copper are general. Tellurium occurs, presumably as sylvanite, but the telluride ores are not common, nor do they offer much prospect of becoming an important source of gold.

Power and Labor.

The use of electric power is general throughout the district, which is traversed by the lines of the Consolidated Light & Power Company. The tariff is based on a flat rate of \$1 per month per installed horsepower, plus two cents per kw-hour for current consumed. Monthly bills up to \$300 are net; 10 per cent discount is allowed on the second \$300; 20 per cent on the third \$300; and 30 per cent on all over \$900. The Homestake company has its own hydro-electric power plant.

Mill labor is well paid, the daily wage ranging from \$3 to \$3.50 according to the class of labor performed. In some cases the standard wage is increased for specially reliable and effi-

cient service, or for the performance of additional service. Foremen receive from \$4 to \$4.50.

Grinding in Solution at Golden Reward Mill.

Current practice at the Golden Reward mill in Deadwood may not be taken as fairly representative of the company's ideas and intentions, although the mill equipment includes many of the modern appliances used in cyanidation. An entirely new mill at another location has been contemplated for some time, and many experiments have been and are still being made with a view to adopting the best machinery and methods in the new plant.

The problem at the Golden Reward is similar to that of a custom mill, for the company treats a variety of ores from several of its mines which are under lease. The result is an ever-varying grade, character and quantity of mill feed, in which each class of ore might yield better results if given the special treatment to which it is best suited. The present sand and slime departments cannot, therefore, give the same results which would obtain on a uniform feed. Some of the ore is soft and oxidized, forming a large proportion of slime; another class is hard, dense and refractory in extraction; and still another is clean quartzite, forming little slime, and easily leached in the sand tanks. The grade varies from \$4 to \$15 per ton, and probably averages \$7 to \$8.

Ore Weighed and Sampled.

As delivered to the mill, the ore is weighed in carload lots, crushed dry by gyratory and rolls, and automatically sampled by buckets attached to a chain traveling through the falling stream of ore. This system gives an accurate sample of the mill feed, provided the stream of ore is falling uniformly and continuously.

Lime is added dry at the crusher, where the ore is reduced to about 1 in. size and elevated to storage bins above the Chilean mills. The latter receive a continuous feed by means of a modified Challenge feeder. Mill solution is added and the ore is ground to pass ton-cap screen having slots 0.021 in. wide. Under these conditions the Chilean mills show a capacity of 4.5 to 5 tons per hour. The discharged pulp is then classified into sand and slime in Akins and drag classifiers. The latter were used altogether prior to the introduction of the Akins, but gave unsatisfactory results under the heavy load imposed on them. At present only one remains and is doing fair work under a light load.

Screen Analyses on Chilean and Classifier Products.

The conditions for the following tests were: Chilean feed, 1 in. and finer; mill speed, 30 r.p.m.; screen, ton-cap 0.025-in. slot; capacity, 5.2 tons ore per hour; about 6 tons solution per ton of ore; one-third of Chilean discharge diverted to drag classifier and two-thirds to Akins, this being the condition under which the drag operated at highest efficiency. Even under these conditions it will be apparent from the figures below that the drag classifier is not as efficient as the more modern machine.

Owing to the variable physical character of the ore it is difficult to give consistent figures on screen life and metal consumption in the Chilean mills. Ton-cap screen has been found best suited to local conditions. Three different makes of steel have been used for rolls and dies, viz., Midvale, Standard and Chrome. The latter has shown varying life in different sets, but is used as being less expensive. A set of three rolls and one die of the different brands have crushed the following tonnages of ore before removal due to wear: Midvale, 9040 tons; Standard, 9347 tons; Chrome, (three sets) low, 6435; high, 9521; average, 8083 tons.

Sand Treatment.

The classified sand is distributed in the ordinary way to leaching tanks. The cycle of treatment varies according to the proportions of sand and slime being made, and it is customary to treat the sand as long as the tanks are not required for new charges. The first solution applied is that decanted

¹Bulletin No. 98, Inst. Min. and Met.

²January, 1913, pp. 50-51; February, 1913, pp. 105-106.

³U. S. G. S. Bulletin No. 225, pp. 123.

CHILEAN MILL DISCHARGE

Mesh	Per Cent
+ 20	0.155
+ 40	6.78
+ 60	9.195
+ 80	3.28
+100	6.89
+200	20.08
-200	53.62
	100.00

CLASSIFIER SAND PRODUCTS

Mesh	Drag %	Akins %
+ 20	1.30	1.33
+ 40	23.60	25.16
+ 60	17.94	25.98
+ 80	3.81	3.01
+100	13.81	10.43
+200	28.63	23.78
-200	10.91	10.31
	100.00	100.00

CLASSIFIER SLIME PRODUCTS

Mesh	Drag %	Akins %
+ 20	0.00	0.00
+ 40	1.58	0.23
+ 60	1.16	0.44
+ 80	0.49	0.27
+100	0.78	0.65
+200	3.90	7.60
-200	92.09	90.81
	100.00	100.00

from the thickening slime. This is an unusual procedure, for one would expect to find this solution flowing directly to the precipitation department rather than used as a leaching solution. The sand tanks, however, act as clarifiers of the solution, and the value of the latter is materially enhanced in its passage through the sand. This first application continues for three days, after which successive washes are applied, bringing the total time of sand treatment up to 14 or possibly 15 days.

Vacuum Filtration for Slime.

The slime from the classifiers is thickened in Dorr continuous thickeners, but no attempt is made at counter-current washing. Air lifts are used for the transfer of pulp from one tank to the next, and gentle air agitation is used in the storage tanks to keep the slime pulp from settling. As stated above, the solution decanted from the thickeners is used as the first leaching solution on the sand tanks. The total time of slime treatment is about 36 hours, at the end of which time the pulp has a consistency of one part solids to one and a half parts solution, and is ready for treatment in the Moore vacuum filter.

At times difficulty is experienced in washing soluble gold from the slime cake, and this has led to further treatment of the slime prior to its final discharge. It has been observed that, although the gold was not readily washed from the cake, nevertheless it seemed to diffuse rapidly through the water in the unloading tank after the cake was blown off the leaves. The liquor in the unloading tank thus became too valuable to be wasted. Accordingly the usual practice has been modified, and before the discharged cake is finally run to waste it is thoroughly mixed with the solution in the unloading tank and pumped to circular, conical-bottom tanks, in which are suspended filter leaves made of wooden frames covered with poultry netting and canvas. No vacuum is applied, but a clear solution worth from thirty to fifty cents per ton is decanted, precipitated separately from the regular gold solution, and allowed to run to waste. The slime settles to the bottom of the tanks and is discharged periodically as long as it runs thick.

Precipitation is accomplished in the usual manner on zinc shavings. The gold solution has a value of \$2 per ton, and the zinc-box tailing about two to four cents per ton. Sodium cyanide

is used in the mill solution, which has a strength equivalent to 1.5 lb. KCN per ton. Consumption of chemicals per ton of ore treated during the last fiscal year was as follows: NaCN, 0.572 lb.; CaO, 8.956 lb.; Zn, 0.681 lb; and acid 0.248 lb.

Settlement with the different leasers is made on the basis of extraction results. A sample of one kilo is taken from each carload shipped during the month, and this sample is subjected to tub tests to determine percentage extraction

Oil-Fired Roaster at Astoria Mine.

Ore from the Astoria mine, owned by the Golden Reward company, is wholly unlike that of other mines in the district. It contains an average of 10 per cent sulphur, and in the raw state yields only 18 per cent or 20 per cent extraction by cyanidation. Preliminary roasting, however, greatly improves both the chemical and physical condition of the ore, and makes it amenable to cyanide treatment. With a view to treating this ore on a large scale, a roasting plant has been erected at the mine, consisting of a Wedge 20-ft., 7-hearth, mechanical roasting furnace, with the necessary complement of crushing, drying and cooling apparatus. The furnace is adapted to burn oil, having six burners disposed at equal intervals on the hearth next to the bottom.

Only one test run of about 75 tons has been made, using some dump ore which was not as favorable to good results either in roasting or cyaniding as freshly mined ore; nevertheless an extraction of 75 per cent was made when the ore was crushed to ¼-in. size, roasted and treated unclassified in leaching tanks. The furnace showed a capacity of 100 tons per 24 hours, reducing the sulphur content of the ore from 8 per cent to 0.2 per cent. Oil consumption was 9 gallons per ton of ore, at a cost of 39 cents per gallon. The temperature on the fuel hearth was 1300 deg. F., and about 300 deg. less on each succeeding upper hearth. The plant has demonstrated satisfactorily the feasibility of producing a roasted ore that will be readily amenable to cyanidation; and the continued success of this experiment on the part of the Golden Reward company is expected to be of material benefit to the industry in the Black Hills.

Further Roasting Experiments.

Since the completion of the large-scale test, further experiments have been continued on a laboratory scale to determine the effect of different temperatures and of a water wash prior to cyaniding. The tabulations below contain details of a typical roasting-cyaniding test, from which some general deductions can be drawn. It appears that a low-temperature roast, followed by a water wash, gives higher extraction of gold and lower consumption of lime than a high-temperature roast without preliminary washing. The consumption of cyanide does not vary greatly in either case. The extraction is good on both coarse and fine portions, though slightly better on the latter, and in less time. The ultimate selection of a method will depend on the cost of the various operations. The low temperature roast will require less fuel, and the time will be reduced, but the expense of washing will have to be taken into consideration.

Experiments also have been made by grinding the entire ore, coarse and fine, in a tube mill in cyanide solution. Portions were removed successively for treatment by air agitation and settling. The results of such a test are shown in the table below. A striking feature to be noted is that the percentage extraction is high on the 2-hour treatment, falling off sharply at 4 hours, and then gradually rising with increased time until at 51 hours it is the same as at 2, reaching its maximum at 74 hours. This feature was noted in several tests, and shows the ready solubility of the gold. The increased extraction at 74 hours would in no way compensate for the time and expense involved, nor for the consumption of KCN and CaO, which also increased steadily with the time.

The consumption of KCN ranged from 0.3 lb. to 0.7 lb., and of CaO from 14 to 17 lb., increasing steadily with the time.

Still other experiments are now in progress on the hard blue ore, to determine the effect of roasting prior to cyaniding. This

PERCOLATION OF COARSE PORTION—SIZE, 1-INCH

Temp. of Roast Deg. F.	Washed or Unwashed	Value		Consump- tion, Lb.		Extraction Per Cent	Time, Hrs.
		Head	Tail	KCN	CaO		
800	Washed	\$15.10	\$1.60	0.55	17.26	89.41	144
900	Washed	15.20	1.80	0.48	16.87	88.16	144
1000	Unwashed	15.20	2.40	0.38	23.52	84.21	144
1100	Unwashed	15.15	3.40	0.38	22.99	77.56	144
1200	Unwashed	15.20	3.60	0.56	19.08	76.32	144

AGITATION OF FINE PORTION—SIZE, 80 MESH

Temp. of Roast Deg. F.	Washed or Unwashed	Value		Consump- tion, Lb.		Extraction Per Cent	Time, Hrs.
		Head	Tail	KCN	CaO		
800	Washed	15.10	2.00	0.30	9.12	86.76	16
800	Washed	15.10	1.50	0.50	8.66	90.06	48
800	Washed	15.10	1.50	0.30	9.06	90.06	120
900	Washed	15.10	1.40	0.30	9.06	90.74	120
900	Washed	15.20	2.50	*2.20	21.21	83.55	24
1000	Unwashed	15.20	3.30	0.31	17.38	78.29	24
1100	Unwashed	15.15	3.60	0.20	17.82	76.24	24
1200	Unwashed	15.20	3.55	0.20	16.84	76.65	24

*An unusual result: 2.10 lb. KCN and 20.92 lb. CaO consumed in first hour of treatment.

TUBE MILL TEST ON WHOLE ROASTED ORE

Solution: 2 lb. KCN and 20 lb. CaO per ton

Successive Portions	A	B	C	D	E	F	G	H	I
Hrs. tube milling....	1	1	1	1	1	1	1	1	1
Hrs. first standing....	1	1	1	1	1	1	1	1	1
Hrs. second standing....	1	1	2	3	4	4	4	4	4
Hrs. second agitation....	1	1	1	1	16	16	16	16	16
Hrs. third standing....	...	...	...	...	...	4	4	4	4
Hrs. third agitation....	...	...	...	...	...	1	20	20	20
Hrs. fourth standing....	...	...	...	...	...	...	...	4	4
Hrs. fourth agitation....	...	...	...	...	...	...	...	...	24
Hrs. total time.....	2	4	5	6	22	27	46	51	74
Value of original ore, \$	11.70	0	through out						
Value of tailings, \$	1.90	2.30	2.20	2.20	2.20	2.10	2.05	1.90	1.80
Extraction, %.....	83.76	80.35	81.20	81.20	81.20	82.05	82.48	83.76	84.62

ore contains only 2 per cent to 3 per cent sulphur, but is nevertheless very refractory, and it is possible that the physical effect of the roast will prove beneficial in the cyanide treatment. The Golden Reward company is doing pioneer work in this branch of ore treatment in the Black Hills, and the ultimate outcome is of great moment and interest to the district. Present indications are that roasting may become recognized as a valuable adjunct to cyaniding, and that the practice may be extended as a result of the success attending the work of the Golden Reward company.

The Trojan Mill.

Ore treatment similar to that at the Golden Reward is practiced at the Trojan mill. The company operates a number of mines and treats a mixture of oxidized and unoxidized ores, including some blue ore. The average value of the mill feed is about \$6 per ton, and the extraction averages 75 per cent.

Ore is gathered from the several mines by gasoline motor and delivered to three mill bins, which are so disposed and constructed as to discharge over grizzlies to the same gyratory crusher. The grizzly undersize and the crushed product are combined and elevated to the mill by a belt conveyor set at an angle of 16 deg. At the discharge end of the belt is a sampler similar to that used at the Golden Reward, consisting of buckets attached to a chain, designed to travel through the stream of ore falling from the belt. It will be apparent that the efficiency of this type of sampler depends on the delivery of a continuous, uniform stream of ore. Owing to the fact that in this case the feed to the belt is irregular, due to sticky ore and other causes, the sampler is unreliable, and its use has been discontinued. The mill results are checked by the value of bullion recovered and tailings discharged.

Roll Crushing in Solution.

The Trojan is one of two mills in the Black Hills where roll-crushing in cyanide solution is practiced, and its use here cannot be said to be satisfactory. The pulp passing the rolls flows to 7-ft. Chilean mills, where it is ground to pass ton-cap screen

having slots 0.021 in. wide. These Chilean mills have extra wide rings and dies which are very satisfactory on soft ore, but not particularly desirable for hard ore. Solution is added at the rate of 4 to 6 tons per ton of ore, the quantity varying with the character of ore under treatment, more being used for soft ores that produce considerable slime, and less for hard ores. The mills have a capacity of about 225 tons each per 24 hours, or a total capacity of 450 tons for the whole plant.

The ground pulp is elevated to two Dorr classifiers working in series, the sand discharged from the first being reclassified in the second. The separation yields about 55 per cent sand and 45 per cent slime. The sand is delivered to leaching tanks through the ordinary distributors, where it is leached with strong solution for three days and barren solution for two days, after which it is given a water wash and discharged in the usual manner. Extraction in the sand tanks is not as good as in the slime department. The sand tailings have an average value of \$1.60 per ton.

Slime Treatment.

The slime is thickened in a Dorr continuous thickener to a consistency of 1½ parts solution and 1 part solids, and then subjected to air agitation in two tanks in series. Several forms of central tube have been tried in the agitators, including long and short tubes, and a form comprised of a series of hollow truncated cones connected together. The object of the latter type of central tube was to permit discharge of the pulp at successively higher points as the tank filled with slime. Apparently, however, equally good results have been obtained without central tubes, and they have been removed entirely from some of the tanks.

Vacuum filtration is practised with a form of stationary leaf filter. A smooth, uniform cake of 1 in. thickness is readily formed in about 50 minutes. This is washed with barren solution for 70 minutes, and with water for 12 minutes before discharging. As at the Golden Reward, some difficulty is experienced in washing soluble gold from the cake in reasonable time. The washed slime tailing assays about \$1.10 per ton, which is 50 cents lower than the sand tailing.

Recovery of Water from Tailings.

All tailings are impounded by a sand dam, below which is a bed-rock concrete dam. In this connection it is interesting to note that the water, or weak solution, recovered at the dam contains gold to the value of 15 to 20 cents per ton, representing probably soluble gold remaining in the slime and sand tailings. It is planned to recover this solution after it has percolated through the sand dam, and precipitate it on charcoal, after which it will be pumped back to the mill for further use.

The gold solution from sand tanks and slime filter is precipitated on zinc shavings in the usual manner. The head solution assays \$2 per ton, and the tailing from 1 to 1½ cents. The strength of the mill solution is 1.5 lb. KCN, with protective alkalinity of 1.5 lb. CaO. Chemical consumption is 0.6 lb. KCN, and 8 lb. CaO per ton of ore. The gold precipitate is cleaned up, acid-treated and melted in the usual manner.

Proposed Changes and Improvements.

The Trojan mill will be remodeled this summer, and its capacity increased to 500 tons per day. Wet crushing at the rolls will be discontinued, and solution will be added to the roll discharge to convey it to the Chilean mills. The latter will be fitted with screen of larger openings, ton-cap 0.040-in. slot, and a short tube mill will be installed to regrind about 40 per cent of coarse sand. Experiments on Trojan ores indicate that grinding to minus 60-mesh gives the most efficient results. Finer grinding is without material benefit. Hence, it is planned to classify the Chilean discharge to give a sand containing practically all the plus 60-mesh particles for tube milling. The tube will work in closed circuit with the classifier, the overflow from which will then be elevated to a series of Dorr classifiers for separation into fine and coarse products. The present practice of adding dry lime at the crushers will be dis-

continued, and slaked lime will be added at the Chilean mills.

Steel-housed belt and bucket elevators are being built to raise the ground pulp to the Dorr classifiers. The steel housing is sectional, and the plates are riveted except at those points where the occasional removal of a plate is desired. In the latter case the plates are bolted in place, the bolt heads being inside and held in place by a strip of grooved metal. By this construction a plate can be readily removed without inconvenience arising from the displacement of the bolts. The boot pulley has outside bearings, and the shaft runs through split hardwood bearings placed in the housing to prevent leakage at that point. The boot is provided with an overflow just below the shaft, and with a discharge pipe at the bottom through which accumulations can be sluiced to a sump.

Counter-Current Washing to Be Adopted.

Additional thickeners and agitators will be installed, and a system of continuous counter-current washing and agitation will be employed. The details of this treatment have not been worked out, but it is planned to decant solution continuously for precipitation, and to agitate in fresh solution after thickening. By this scheme it is hoped to wash the valuable solution from the pulp before it reaches the filter, and to use the latter primarily as a dewatering machine.

A Lea V-notch recorder, the first to be used in the Black Hills, will be used as a gold solution meter. The Trojan management is of the opinion that this form of solution meter will prove superior to any form of enclosed meter or oscillating boxes, owing to the difficulty of preventing the deposition of scale in the former, and of properly standardizing the latter.

Dry Crushing at Wasp No. 2.

As an example of low cost treatment of a very low grade ore, Wasp No. 2 mill holds an unique position in the Black Hills, and indeed is almost without parallel in the realm of cyanidation'. The ore is coarse grained quartzite, sufficiently porous to permit ready solution of the gold and a high rate of leaching when crushed to pass a 1/4-in. mesh screen. According to the annual report for 1912, the ore averaged \$2.34 in gold and 0.64 oz. silver per ton. The extraction was 75.9 per cent of the gold and 30.3 per cent of the silver. Under these conditions, and with a daily capacity of 500 tons, the profit on the operation is about \$5,000 per month. Tonnage is a factor in the profitable operation of this mill; at a capacity of 300 tons per day, a net loss would be sustained, while at 400 tons per day expenses would be paid, but no net profit would result. The total mining and milling cost for 1912 was \$1.2337 per ton, of which 62.65 cents was for milling. The distribution of this item was as follows:

	Cents per ton
Labor	20.57
Supplies	6.00
Repairs	9.10
Coal	0.85
Cyanide	8.06
General	0.62
Stable	0.90
Assay Office	0.69
Superintendence	1.58
Lime	1.36
Clean-up	0.87
Zinc	3.64
Power	8.28
Tools	0.13
Total	62.65

Crushed Ore Not Classified.

The system of treatment is extremely simple as compared with that necessarily adopted where sand and slime must be separated. The ore is crushed in gyratories, first to 2 1/2 in. and

then to 1 1/2 in. The latter product is further reduced in roughing rolls, and passed over stationary, inclined rek-tang screen of 1/4-in. mesh, the oversize from which is returned to finishing rolls. Dry lime is added at the second gyratory, which is a practice that may be criticized on account of the coarse condition of the ore and the improbability of thorough mixing in the storage bins.

The crushing faces of the gyratories and rolls are made of manganese steel. Gyratory mantles last about one year, and concaves six months. Roll shells wear for 50 days. These figures relate to the treatment of 500 tons per day.

No screen analysis is available on the product passing the screen; but although it contains more or less fine sand, percolation and extraction proceed at a rapid rate. The ore is loaded into 32-ft. by 12-ft. leaching tanks of slightly over 400 tons capacity, by means of a system of conveyor belts. As the ore drops into the tank it is distributed by shoveling to avoid classification of coarse and fine particles. Twelve tank charges are treated every ten days.

The cycle of sand treatment is as follows: A 5-lb. cyanide solution is run onto the charge until the ore is covered. Contact is continued for 12 hours, after which the tank is drained and the contents treated with successive applications of a 2 1/2-lb. solution for from 48 to 56 hours. A water wash completes the treatment. Strong and weak solutions are precipitated separately on zinc shavings, the head and tailing values of the strong solution being \$3 and \$0.08, and of the weak solution \$1 and \$0.02, respectively. One ton of solution is precipitated per ton of ore treated. Chemical consumption per ton of ore is: KCN, 0.35 lb.; CaO, 5 lb.; Zn, 0.3 lb.

Cost of Unloading Tanks.

Owing to the situation with relation to the railroad, it is impossible to discharge the tanks by sluicing, and the tailings have to be stacked dry. Shovelers unload the sand through gates in the bottom of the tanks, filling cars which are then trammed to the dump. A system of belt conveyors might be used for this purpose were it not for the difficulty of operating in winter. The cost of unloading is about five cents per ton, a tank being discharged in seven hours by seven men, four of whom are engaged in shoveling and three in tramping.

The gold precipitate from the zinc boxes is recovered and treated in the usual manner, producing bullion of a total fineness of 900 to 930.

Owing to the uniform character of the ore and the simple system of treatment, it is considered unnecessary to use automatic sampling devices for arriving at the tonnage and value of ore treated. Each tank charge is pipe-sampled at various points, and the estimate of tonnage and value made in this way.

For courtesies extended and information furnished for these notes, acknowledgment is due to Messrs. Henry Schnitzel, F. R. Baldwin and M. E. Hiltner, of the Golden Reward company; H. S. Vincent and Charles Ellis, of the Trojan; and John Gray and C. F. Brenner, of the Wasp No. 2.

Absorption and Reaction Towers for Chemical Factories.

—By a mistake the name of the author, Mr. Rudolph Heinz, of Germany, was omitted in the publication of the article on page 359 of our June issue.

The Aluminum Industry Co., of Neuhausen, Switzerland, reports a net profit of \$717,000 for 1912, as against \$448,000 for 1911. It is stated that the low prices which prevailed for some time influenced the development of the industry by creating new uses and demands for the metal. A new agreement has since been entered into by the various aluminum makers.

Thermit marine repairs is the title of a very interesting and well-illustrated pamphlet of 30 pages of the Goldschmidt-Thermit Company, 90 West Street, New York, giving an account of recent progress made in marine repair work with the use of the thermit process with various examples from American practice.

¹An example of coarse-crushing and leaching in Mexico was described in this journal, May, 1913, p. 291.

The Purification of Blast Furnace Gases.

A Report of a Paper by C. Herwegh, Giving Data on the Use of the Feld Washer in French Blast Furnace Practice—Details of a Proposed American Installation

In our issue of July, 1912, we published an article by Mr. Walter Feld on his centrifugal gas washer and its application for the purification and cooling of blast furnace gas and producer gas and other purposes.

In this connection a paper by Mr. Camille Herwegh, read before the Mining and Metallurgical Section of the (French) Société Industrielle de l'Est is of considerable interest as it gives data from actual practice at the *Pompey blast furnace plant* which has been the first iron blast furnace plant to adopt the Feld washer on a large scale.

The author first gives data on the quantities of blast furnace gas available and its usual calorific value and the troubles due to the content of dust in the same. In their furnaces "de l'Est" the quantity varies from 8 to 18 grams per cubic meter of gas (3.52 to 7.92 grains per cubic foot). The obnoxious influence exercised by this dust is readily understood; it adheres to the walls of the recuperators and to the flues of the boilers, thus forming a coating which very greatly diminishes the efficiency of the apparatus. This demands frequent cleaning of the gas passages and of the various apparatus; the dust in the gas therefore produces a perceptible loss in fuel and a higher maintenance cost. If used in the engines, the dust packs in the valves, thus in a short time preventing their proper operation.

Methods of Purifying Gas

Mr. Herwegh then gives the following outline of different methods of purifying the gas.

"A great many systems were both proposed and tested for this purpose; the first ones were constructed of channels having many changes in direction and area, or of compartments with partition walls located in different directions. This was the 'dry' method of purification, and it is still in use for precipitating the heavy dust, which falls down easily, but it is not sufficient for all purposes. To obtain a more complete cleansing, columns of coke or gravel with a continuous flow of water were used, this being the beginning of 'wet' purification, but the results left much to be desired.

"The gases passed through this apparatus without proper contact with the water, and were purified until they contained about 3 to 5 grams of dust per cubic meter (1.32 to 2.2 grains per cubic foot) at the outlet. Afterwards coolers made their appearance; they at times had enormous dimensions, and used large quantities of water with a consequent high amount of motor power.

"Before we erected our purifying plant at Pompey we tried primary purification without the use of water or power. This apparatus, designed by M. Péard, mining engineer, was so constructed that the gas passed through pipes around which air was circulated—the air being continuously renewed by natural draft. It was provided with a dump for dust and for condensed water. This apparatus, which is still being used for cleansing the gas used under the boilers, necessitates large dimensions to enable it to purify 3,000,000 cubic meters (105,000,000 cu. ft.) per twenty-four hours.

"In the coolers using water circulation, the gases having a temperature of 80° C. (176° F.) at the inlet, are cooled down to 30° C. to 25° C. (86° to 77° F.), a certain quantity of dust is precipitated and a large portion of the water is condensed; the efficiency, however, is very low compared with the amount of water used—from 3 to 5 liters per cubic meter (0.09 to 0.15 quarts per cubic foot)—and the gas still contains from 2 to 3 grams of dust (0.88 to 1.32 grains per cubic foot) at the outlet.

"Ventilators were placed behind this apparatus, and here from 1 to 2 liters of water per cubic meter (0.03 to 0.06 quarts per cubic foot) were injected, and we were thus able to effect the desired degree of primary purification; that is down to 0.05 to 1 gram (0.22 to 0.44 grains per cubic foot) for the gas used in the recuperators and boilers. But these ventilators

absorb a great deal of power. For instance, a ventilator passing 50,000 cubic meters (1,765,000 cu. ft.) of gas per hour without water injection absorbs 50 hp, while the same ventilator absorbs 180 hp when 2 liters (0.06 quarts per cubic foot) are injected.

"There are a great number of patented washers designed to replace coolers and ventilators with water injection, but none of these have heretofore operated with much success.

Feld System

"This system effects the cleansing and cooling consecutively in such a manner that the primary cleansing is completed before the gases are completely cooled down.

"The apparatus consists of a washer (Fig. 1) built up of sections one above the other. The gas inlet and the outlet are indicated in the diagram; the gas passes upwards from one washing chamber to the other through gas ports.

"Each section is provided with a water-spraying device at-

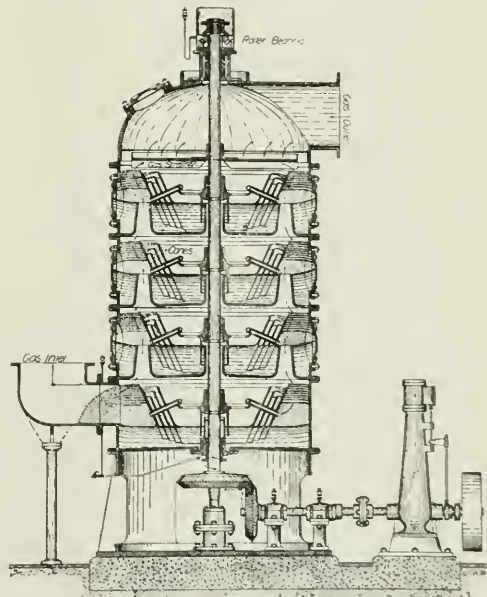


FIG. 1—FELD GAS WASHER

tached to the vertical shaft. This spraying device is composed of a series of concentric cones attached to a hub keyed to the shaft, the lower ends of the cones being submerged in water. By revolving the shaft, and hence the cones, the water is raised by centrifugal power along the inner sides of the cones and atomized at the upper edge.

"The upper edge of each cone is a little higher than the next outer one, thereby forming, according to the number of cones, a certain number of horizontal sprays of water. The upper portion of the outer cone, which is somewhat higher than the inner one, is perforated, the inner cones supplying water to the perforated surface of the outer one. This forms a series of cascades composed of excessively small drops of water through which the gas to be cleansed must pass. The water is violently thrown against the walls of the washer, and then returns under the revolving cones; this permits of using the same body of water as long as it may be desired. Theoretically, only such amount of water flows out of each section as may rise above the normal level. The washing is done in the lower sections while the cooling takes place in the upper ones.

"In order to decant only a small portion of the dust-bearing water, about three-quarters of the water is taken from the

third section from the top, without thus decreasing the cleansing, or the efficiency of the Feld washer is such that the wash water may be concentrated without detrimentally influencing the purification. It is also noted that hot water washes the gas much better than if cold water were used, and all sediment is thereby avoided.

"The upper part of the washer is provided with perforated plates, which, through their action, cool the purified gases above the cones. This cooling plays an important role in the condensation of the water vapor. In the later washers these perforated plates are replaced with advantage by another series of cones. The washer has a diameter of 2.9 meters (9 ft. 6 in.) and a height of 5.8 meters (19 ft.).

"You can readily see that this washer acts both as a purifier and as a cooler; the ventilators, which they have replaced as purifiers, now only serve to draw the gas through the apparatus and thus to overcome the resistance offered in the pipe connections and in the washer itself. I must state, however, that a small additional quantity of water must be injected into the ventilators to prevent the slow precipitation of dust which still remains in the gas at the outlet of the Feld washer and which dust would adhere to the inside of the ventilator.

"Washers of the same construction are used for the recovery of tar, tar oils, benzole, cyanogen, ammonia and hydrogen sulphide from gas.

European Installations

"The plant at Engers, Germany, tried the Feld washer in a small way in connection with a blast furnace; there was only one washer, but it was not connected with a ventilator, and only such amount of gas as happened to pass through the washer was purified.

"The lead mines at Mechernich-on-the-Rhine, where they work up a 4 per cent lead ore, also erected two washers to recover the lead dust which formerly accumulated in five immense galleries placed underground and leading to the stack. These galleries were built in sections of 400 meters (1316 ft.) each, or 2 kilometers (1.25 miles) in all, the chimney having a height of 130 meters (427.7 ft.). In spite of these galleries, the gases leaving the chimney inundated the surrounding country with their unhealthy dust contents, which were excessively rich—containing 80 per cent to 100 per cent of lead. Twice a year, consuming a period of three weeks each, the entire plant was shut down to permit of cleaning out these galleries by means of wheelbarrows and carts. You can readily understand how unhealthy this work was to the laborer, and the loss sustained by the company during these cleaning periods, as well as by the damage done by the gases escaping from the chimney.

"In order to prevent this loss, two Feld washers were erected, each with but two sections, working in series, a ventilator being attached to the second washer.

"During twenty-four hours this apparatus recovered six tons of dust, containing 80 per cent to 100 per cent of lead. From these tests in the lead mines, where the dust was entirely recovered, we were positive that this apparatus would work satisfactorily if applied in a large installation for the cleansing of blast furnace gases and for condensing the water vapor carried in the gas.

Power Consumed by the "Feld" Washer and Other Details

"An apparatus such as I am about to describe to you, having a capacity of 1,000,000 cubic meters (35,300,000 cu. ft.) of gas in twenty-four hours, consumes only from 15 hp to 20 hp. The shaft is suspended in a ball bearing. When the motor is stopped the shaft continues to revolve for about twenty minutes, the water continuing its circulation. The washer consumes about 3 liters of water per cubic meter of gas (0.09 quarts per cubic foot).

"After several months of operation the apparatus was stopped to be inspected and cleaned, if necessary, but no sediment was found and the washers were closed again. During a period of eighteen months the washers at Pompey were run continuously—day and night—and were never closed down for repairs.

"The degree of purification is from 0.4 to 1 gram of dust per cubic meter (0.17 to 0.44 grains per cubic foot). The gases leave the blast furnace carrying 120 grams of water vapor per cubic meter (52.8 grains per cubic foot) and leave the Feld washer with 2.8 grams (1.23 grains per cubic foot) above the theoretical point of saturation corresponding to the temperature.

Primary Purification Plant at Pompey

"When the gases leave the furnaces they first pass through the 'dry' purifiers; at the outlet the gases have a temperature of 150° C. (302° F.) and contain from 12 to 15 grams of dust (5.28 to 6.6 grains per cubic foot) and 120 grams of water vapor (52.8 grains per cubic foot) per cubic meter. They enter at the bottom of the first 'dry' dust catcher, this having a diameter of 3 meters (9.87 ft.) and a height of 9.5 meters (31.26 ft.) over the straight shell; they leave at the top through a pipe having a diameter of 2 meters (6.58 ft.) and enter the bottom of the second 'dry' dust catcher having a diameter of 4 meters (13.16 ft.), leaving at the top and entering a general collector 2.8 meters (9.21 ft.) in diameter. You can see from these dimensions that in the 'dry' dust catchers we have combined both a change in direction and a change in cross-section of conduits, so as to vary the velocity of the gases and to facilitate the precipitation of the dust. The gases do not contain more than from 3 to 5 grams of dust per cubic meter at the outlet (1.32 to 2.2 grains per cubic foot).

Wet Purification

"Contrary to the usual practice at several large plants, where only the gas required for gas engines is purified, the entire gas coming from the furnaces at Pompey receives primary purification, and the benefit thus realized in the increased efficiency of the boilers and coppers, by reducing the cost of cleaning and maintenance, is very important.

"The entire gas from the furnaces, which is collected in the general collector, is passed through the Feld purification plant. This plant consists of three washers, each washer being connected to an independent ventilator; the plant is erected within a building 12 by 16 meters by 15 meters high (39.5 x 52.6 x 49.4 ft.). The apparatus was supplied and erected by the well-known firm of "Fives Lille," which has acquired the French patents. The motors and shafting operating the washers, as well as the pumps, are located on the first or basement floor, the washers and ventilators being placed on the second floor. A traveling crane above the washers permits of quickly dismantling and removing any part of the apparatus. The ventilators force the gas either to the coppers or boilers, or to the final cleansing plant for the gas engines.

"A special main was provided to carry the raw gases direct from the collector to the ventilators, without passing through the washers. If this is used, the ventilators must be supplied with an additional amount of water for cleansing the gas if the washers are by-passed for repairs or any other purpose. This was a very important precaution in trying out a new system, but it has never been used.

"Below are given the average results taken from numerous tests made during a period of three months, or from December, 1910, to February, 1911:

"Quantity of dust per cubic meter of washed gas at 0° C. and 760 mm. pressure (32° F. and 30 in. pressure)—0.483 grams (0.21 grains per cubic foot).

Temperature of the washed gas above the water in the washer cooler—0.3° C. (43.34° F.).

"Quantity of water in the washed gas above the theoretical point of saturation, corresponding to the temperature—2.83 grams per cubic meter (1.245 grains per cu. ft.).

"Water consumed per cubic meter of gas at 0° C. and 760 mm. pressure—3.62 liters (27.05 gallons per 1000 cu. ft. of gas.)

"The impure gas contained 121.6 grams per cubic meter (53.5 grains per cubic foot) of water vapor at the inlet; if it had been 80 grams (35.2 grains per cubic foot), on which the guarantee was based, the amount of water would have decreased perceptibly.

"The three washers consumed 15 hp each. The three ventilators, which, in order to maintain a difference in pressure, consumed 35 hp each, would have consumed from 60 hp to 70 hp each if operated with a difference in pressure of 160 mm. (6.3 in.) between inlet and outlet. (The power consumed by shafts and belting has not been included in the above.)

Decantation

"The decantation process is also well worthy of description.

"The dust-laden water is carried by an underground canal to a concrete reservoir, from where an electric pump forces it up to a pointed wooden vat known as "spitzkasten" and used in coal washeries. The water enters at one end of this vat, and the openings for distributing the decanted water are at the other end. Broom straw, placed on supports at the water level, hold back the scum and facilitate decantation. The heavy-laden water accumulates at the bottom and is drawn off by means of goose-neck-shaped pipes—the pipes being provided with cocks at their lower end—into a concrete basin; the sediment being drawn from the bottom of this basin into cars and removed to a dump."

Proposed American Gas Washer Plant for Two Blast Furnaces Producing 5,400,000 Cu. Ft. of Gas per Hour

In connection with this paper the following design of a Feld gas washer plant of a capacity sufficient for two blast fur-

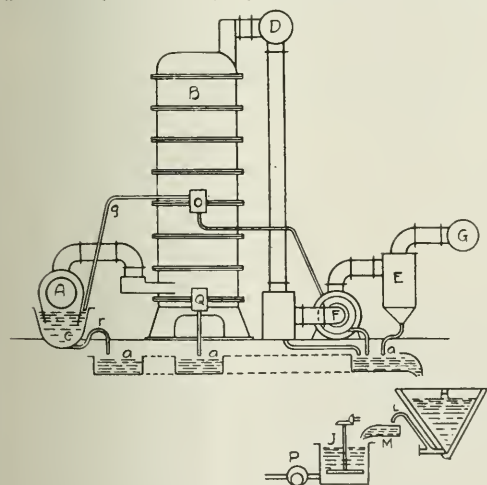


FIG. 2—GAS-WASHING PLANT

naces producing 5,400,000 cu. ft. of gas per hour is of special interest. For this information we are obliged to Captain **Fred H. Wagner**, chief engineer of the Bartlett Hayward Company, Baltimore, Md. This company owns the patents for the Feld washer in this country.

In the plant, as further described below, the necessity of arranging the apparatus in such manner that it occupies as little space as possible is of prime importance, at the same time bearing in mind the necessity of so arranging the apparatus that a general supervision and easy access to each individual piece of machinery is possible.

Design for Continuous Operation

Furthermore, in contradistinction to all of the usual plants seen in America, this one is arranged to avoid all periodic interferences with continuous operation.

As usually constructed at present, we find the plants subject to numerous cleaning periods for the gas mains as well as for the washing apparatus, that is, the removal of accumulated dust, which periods entail serious interruption to continuous operation. This condition has been completely overcome by the arrangement adopted for both the washers and the pipe connections; also the periodical cleaning of the settling basins,

entailing a considerable expenditure of labor and money, is completely obviated, due to the method adopted to automatically concentrate the mud in the clearing basins, and the continuous disposal of the concentrated mud by mechanical means, thus reducing the labor to a minimum.

The entire plant is placed at a sufficient elevation above the ground line to permit of the overflows from the various machines to empty into the clearing basins, the cleared water to run into a sewer and the overflows from the clearing basins to empty into the mud concentrators and from there to the mud pumps by gravity, all pumping inside the plant being thus avoided.

The apparatus is designed to primarily clean the gas from two furnaces, the quantity of gas from each furnace amounting to 2,700,000 cu. ft. per hour (76,487 cbm.), or 45,000 cu. ft. per minute (1275 cbm.) at 250° F. (120° C.), and, as shown in Fig. 2, it consists of:

- A = Inlet main from dry dust catcher.
- B = Vertical centrifugal washer.
- C = Hydraulic main.
- D = Collecting main.
- E = Water separator.
- F = Exhauster.
- G = Primary gas main.
- H = Cone basin, or clearing basin.
- J = Mud concentrating tank.
- P = Centrifugal mud pump.
- r = Mud overflow.
- a = Mud trough.
- O = Proportional overflow.
- Q = Washer overflow.
- L = Mud overflow from settling tank.
- M = Mud trough.
- g = Water to main.

The unpurified gas passes from the furnace through the down comer into the dry dust catcher (not shown) and from thence by means of the hydraulic main A into the washing plant. In order to avoid periodical cleaning of the gas main from the dust catchers to the washers B, the horizontal stretches of the main A are formed as a dip, the lower open edges of the main dipping into the water seal C. The dust, which may be separated from the gas by friction during passage through this main, is deposited in the trough C, which is always filled with warm running water, and the resultant mud in a concentrated form is automatically carried off through the syphon overflows r to the mud trough a, and from thence to the cone basin H, where the mud receives a further degree of concentration.

A reduction in the cross-section of the inlet pipe by the deposit of dust or mud on the inner surface and a consequent loss in pressure is thus avoided; the periodic interruption caused by necessary cleaning, as well as the usual disagreeable condition produced by the dust during the period when the usual type of inlet main is being cleaned is also avoided. Besides this, the hydraulic main provides a sufficient safety valve during a possible explosion in the furnace.

The gas is now admitted into washer B, where it is cooled and cleaned, the dust content being reduced to 0.13 or 0.20 grain per cubic foot, dependent upon conditions, and the gas being cooled to about 77° F., the gases entering the washer with a temperature of 250° F. and the water vapor content of 53 grains per cubic foot; it is possible, however, that the gases may enter the washer at a higher temperature, and the higher the entrance temperature the better will be the effect of dust washing.

At the inlet nozzle of the washer the hot, saturated gases come into intimate contact with the hot water in the lower washing chamber, whereby the temperature of the gas is brought to the equalizing temperature in this case to about 57° C. = 135° F., and the unit volume is thereby reduced. The washers are provided with three lower washing chambers, one intermediate or separating chamber and three upper or cooling chambers.

The three lower chambers are supplied with hot water, this water having been heated in the upper cooling chambers and the temperature of which changes with the total heat value of the gas, depending upon the water contents and the entrance temperature of the gas, this hot water being the active dust extracting medium.

In the upper or cooling chambers, the gas coming from the washing chambers with a temperature of about $57^{\circ}\text{C} = 135^{\circ}\text{F}$. is cooled down to $25^{\circ}\text{C} = 77^{\circ}\text{F}$. The final temperature is, of course, dependent upon the total heat value of the gas and upon the temperature and amount of cooling water used. The quantity of water required for cooling the gas is admitted into the top cooling chamber, the water extracting heat from the gas during its passage through the chambers; about a fifth to a sixth of this hot water is admitted into the lower or washing

The cooled and primarily cleaned gases now pass through the collecting main D and from thence down to the exhaustor F and water separator E, which in turn delivers it to the primary gas main G, and the gas is now ready for use as fuel in the stoves, furnaces or boilers.

The exhaustors are so constructed that it is possible to operate them with water injection, but normally only about 7.5 gallons of water coming from the overflow O would be run into the exhaustor per hour in order to clean the casing and revolving vanes; but the water connections to the exhaustor are of such diameter that the exhaustor can be supplied with a greater amount of water from time to time if such should be required, so that the machine can be thoroughly cleaned without shutting it down.

If it should be desired to clean the gas for gas engine fuel,

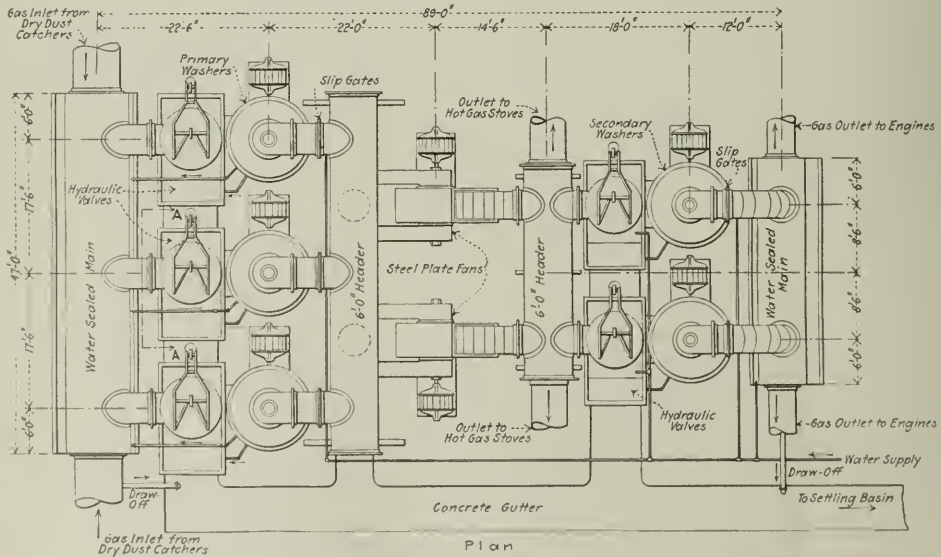
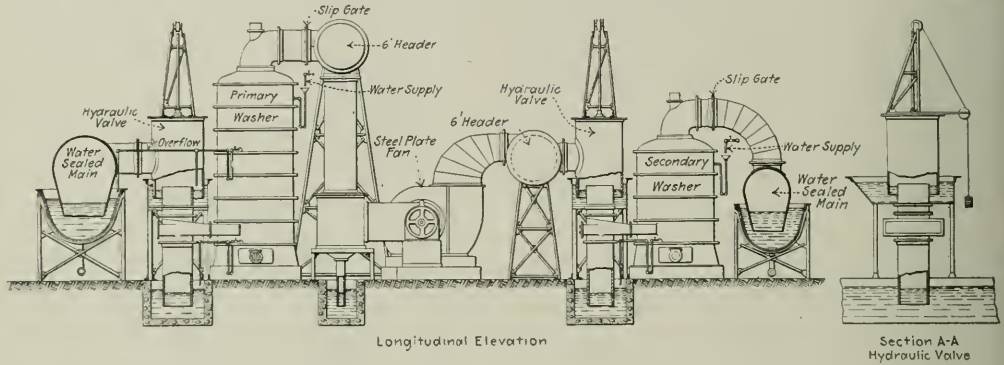


FIG. 3.—LAYOUT OF GAS-CLEANING PLANT FOR TWO BLAST FURNACES

chambers, where its temperature is increased from about $50^{\circ}\text{C} = 122^{\circ}\text{F}$. to $60^{\circ}\text{C} = 140^{\circ}\text{F}$.

The hot water coming from the overflow O under the cooling chambers, and which water contains but little dust, passes into trough C under the inlet main A and serves to produce a flow in the trough. This water finally passes into the cone basin H by means of the syphon r and mud trough a. The hot water which leaves the washer at Q contains the major portion of the dust washed out of the gas, or about 90 per cent, and it flows directly to the cone basin H.

it would have to be subjected to final or minute cleaning. To do this it would be necessary to place another washer in the system, but this washer would not be provided with cooling chambers, and the connections should be so arranged that the necessary amount of engine gas could be diverted from main G into the final washer.

As stated above, the entire mud-bearing water is conducted into the cone basin H for concentration. Experience has shown that these cone basins are sufficient for the purpose, but if further concentration should be required this can be added

to. The time required for clearing the water is dependent upon the nature of the dust and upon the temperature of the washing water, and therefore upon the temperature of the gases.

The mud-bearing water is distributed over the area of the basin by means of a distributing trough placed at one end; the mud, owing to the slow flow of the water through the basin, is deposited on the bottom, and the cleared water passes out on the other end and is conducted to the sewer.

The deposited and concentrated mud is automatically carried off by the overflows L into the collecting trough M, from whence it flows by gravity into the agitators and collecting tanks J. These agitators are supplied with centrifugal mud pumps P, and the outflow is so arranged to the pumps by gravity that they have no suction, their only requirement being to force the mud to the final depository. The collecting tanks J are connected with each other by means of overflows, the agitators being intended to prevent a settling of the mud during pumping periods.

Fig. 3 is a detailed drawing of the layout of a plant for two blast furnaces to be constructed by the Bartlett Hayward Company of Baltimore for an American blast-furnace plant.

Metering and Recording the Flow of Fluids* NEW METERS AND RECORDERS OF THE FLOW OF LIQUIDS THROUGH VENTURI TUBES, OR- FICES, OR CONDUITS, BASED ON THE IN- TEGRATION OF THE VELOCITY HEAD

By J. W. Ledoux

The flow of water and other liquids as well as gases follows the law of gravity. For steady flow this may be represented by the expression: "Velocity is proportional to the square root of the head-producing velocity," or, algebraically,

$$V = C\sqrt{H}$$

in which

V = the velocity, in any convenient unit,

H = the head producing such velocity,
and

C = a constant depending on the units of measurement and the character of space through which the fluid flows.

V may be expressed in feet per second, miles per hour, etc., and H , the effective head, may be expressed in feet or in pounds per square inch, etc.

The space may be a pipe of any length, a submerged orifice, an open inclined channel, a Venturi tube, or a nozzle. It is seen that this is the form of the simplest equation of a parabola $y^2 = cx$.

For practical purposes, the ideal form of meter would be a device, connected by small tubes with adjacent parts of a pipe through which fluid is flowing, which would measure automatically the square root of the head or pressure producing the velocity. Such a device would offer no resistance or obstruction to the flow through the pipe. It has been impossible, heretofore, to achieve this ideal, because the forces due to velocity head are so minute as to be difficult, if not impossible, to measure or integrate when the velocity is low.

Theoretically, the Venturi tube is a contraction in the pipe, formed so as to cause small frictional resistance to flow; and it is remarkable how slight this contraction may be in order to produce the same difference in velocity head as that caused by Pitot tubes without contraction in the pipe. In fact, the diameter of the contracted portion need not differ from that of the full section by more than 17.7 per cent in order to produce the full effect of two Pitot tubes, one pointing up and one pointing down stream, with their openings in the center of the pipe, or at the points of maximum velocity.

In practice, however, the Venturi tube is composed of a short pipe continuously connected with the main pipe by a conical section pointing down stream, the convergence of these two

sections being dependent on experimental determinations, the object being to obtain the highest flow with the least frictional or unrecoverable loss of head through the entire Venturi tube. These angles of convergence and divergence have been determined by Venturi and, in more modern times, by Weisbach, Eytelwein, Francis, Smith, Herschel, and possibly others, but at any rate it is pretty well settled that they should be somewhere between fairly wide limits, beyond which the results are certainly not as good.

However, they may be relatively the same for all Venturi tubes, whether the contraction is nearly as large as the pipe or very much smaller, the difference being that in the former case the entire Venturi tube would be short and in the latter long.

Of course, the purpose of a Venturi tube, the contracted section of which is small as compared to the full bore of the pipe, is to exaggerate the velocity head indications so that they can be observed and measured, especially for low flow.

To illustrate, consider the simple formula of a standard Venturi tube, having a coefficient of velocity = 0.977:

$$V = 7.84 \left(\frac{H}{(D/d)^4 - 1} \right)^{1/2}$$

in which the only two variables are V and H .

V = the velocity through the full section, in feet per second;
 H = the difference in piezometric heads between the up-stream section and the contracted section, in feet;

D = the diameter (or radius) of the full section;

d = the diameter (or radius) of the contracted section.

This formula may be written:

$$H = 0.0163 \{ (D/d)^4 - 1 \}$$

Now, assume a velocity in the main as low as 0.25 ft per second. Then the formula may be written

$$(D/d)^4 = 1 + 982H$$

and assume 0.01 ft. to be as low as it is possible to indicate H : then

$$D/d = \sqrt[4]{1082} = 1.86,$$

which means that when the velocity through the main is as low as 0.25 ft. per second, if it is desired to get an indication of the velocity head as great as 0.01 ft., the contracted section of the Venturi tube must be less than 54 per cent as large in diameter as the main.

However, with a tube having ratios of diameters equal to 3 to 1, $H = 0.0815$ ft. for $V = 0.25$ ft. per second. Even this result cannot be obtained with the best apparatus formerly in use, unless one of the essential features of the Venturi meter is seriously sacrificed, namely, the measurement of high velocities.

The best types of Venturi meter registers have formerly depended for their registration on some form of cam having a parabolic curve, fashioned around a drum or revolving about a center. Near zero, or the origin, the curve changes in direction rapidly, and at zero crosses one of the axes of reference at right angles and becomes parallel to the other.

It is probably this feature, more than any other, which has limited the minimum velocity measurable by a Venturi meter; and though the measurement of low velocities is attainable by a high ratio of tube diameters, there is thus encountered the other difficulty of excessive frictional loss through the Venturi tube for comparatively high velocities.

If it is even intended to measure velocities as high as 10 ft. per second, which is occasionally desirable, the frictional head through a Venturi tube having a ratio of 3 to 1 in diameters, or 9 to 1 in areas, would be 17 ft., obtainable from the formula $H_f = 0.0021 V^2$, in which H_f = the frictional head in feet and V = the velocity through throat, in feet per second.

Besides that, in any form of register depending on mercury as the medium of indicating the velocity head, in order to cover ranges of velocity of from 10 ft. per second down to zero, the apparatus would have to be more than 11 ft. high. In a tube having a ratio of 2 to 1 in diameters, or 4 to 1 in areas, with a velocity of 10 ft. per second through the main, the frictional head would be only 3.36 ft. This is still a material loss, but a velocity of 10 ft. per second is unusually high.

*Extracts from a paper published in the May, 1913, issue of the *Proceedings of the American Society of Civil Engineers*, page 1065.

It must be remembered that, as far as metering a fluid is concerned, any form of orifice or contraction in a pipe would be just as good as a Venturi tube. It is also highly probable that the coefficient of flow would remain as nearly constant through just as wide a range of velocities; but the great advantage of a Venturi tube is in its small frictional resistance to flowing fluid, and in this respect it is the best shape yet devised.

The writer will now describe a form of meter register particularly adaptable to the registration of flows through any form of orifice, such as by the use of Pitot tubes and Venturi tubes of small ratios and consequently minimum frictional heads. The fundamental principles were discovered by the writer in 1904 and were finally developed into the best practical working form in 1909. The basic idea is a float, shaped so that, when acted on by the head due to the velocity of flowing liquid, its movement will be proportional to that velocity.

The device is novel in form, and covers a principle in hydraulics, which, although rational and perfectly obvious on analysis, the writer believes has not heretofore been recognized. This may be stated as follows:

If a hollow body of any shape, with open bottom (Fig. 1), floats freely in a liquid having the same specific gravity and touches another liquid of greater specific gravity, and if the relative pressures without and within this hollow body are changed so as to make the internal pressure less than the external pressure, then the body will sink into the heavier liquid, and the amount of displacement will be exactly equal to the quantity of the heavier liquid which will pass up into the interior of the body and above the level of the heavier liquid outside of the body.

Let the hollow body, or float, be represented by $n n a a$, Fig. 1; assume it to be a figure of revolution, and call the total volume of the heavy liquid, $a a m m$, within the float M .

The principle may be understood by a consideration of the laws of equilibrium—the algebraic sum of the vertical forces must be equal to zero; the sum of the horizontal forces, and the sum of the moments must all be equal to zero.

We need consider only the first condition as affecting the problem. At the plane $b c b$ the heavier liquid is subject to the same unit pressure within and without the hollow body, but the resultant of external forces acting downward on the hollow body is measured by the weight of the heavier liquid, $m m c c$, and the total upward force acting on the body is the displacement of the annular portion of the floating body, $h c a b c a$.

For equilibrium, the algebraic sum of these two forces must be equal to zero, and, as one is positive and the other negative, they must be equal in amount; but, if that is the case, the entire quantity of liquid equivalent to the displacement of the hollow body, and no more, must be found in the interior of the hollow body, and hence the outer level of the surface of the heavy liquid does not change with variations of pressure out-

side and inside the float, as will be very easily understood.¹

If there is a difference of pressure, H , then the float will sink into the heavier liquid an amount represented by the annular volume, $a c b$, which will be called V ; and the net volume of liquid, $m m c c$, will also be equal to V , and h_2 will be proportional to H . As before stated, the level $b b$ will always be the same, no matter what the values of H , h_2 , and V may be. Let

x = the vertical movement of the float from a given datum;
 H = the Venturi head;
 h_2 = the corresponding mercury head;
 v = the velocity through the Venturi tube;
 y = the ordinate or variable radius of the inner portion of the float;
 h = the corresponding abscissas, measured from the bottom of the float;
 r = the constant radius of the outer cylindrical portion of the float;
 M = the total volume, $a a m m$, of the heavy liquid within the float;
 V_o = the volume of heavy liquid above cc ;
 and

V_i = the volume of the float below $bc cb$.

Assume the radius of the pipe, P , to be small and negligible. Take, by hypothesis, $v \propto x$, also for unity the coefficient, $x^2 = h_2$, for $v \propto \sqrt{H} \propto \sqrt{h_2}$.

Therefore, the cross-section of the float must conform to the following equation:²

$$y^2 = \frac{r^2}{2\sqrt{h} + \frac{1}{4}} \quad (1)$$

This is an equation of the fifth degree, as between the two variables, h and y . No attempt has been made to discuss this equation exhaustively or plot all the loci, for, though this would be interesting from a purely mathematical standpoint, it is evident that only that locus will satisfy the physical conditions wherein both h and y have positive, real values.

For $h=0$, $y=r$, and for $h=\infty$, $y=0$, thus it is seen that there are no values of h between 0 and ∞ which give infinite or imaginary values of y . As a consequence, the integrating apparatus operated by the sheave, S , is theoretically accurate from zero up to any practical limit that may be desired.

The shape of the float is such that its vertical movement is proportional to the square root of h_2 , or H (the mercury and water Venturi heads, respectively); and as the velocity through the Venturi tube is also proportional to the square root of H , the vertical movement is directly proportional to the velocity or quantity of water passing through the Venturi tube.

The same principle will hold true if the Venturi tube is replaced by any form of orifice, or if the pipes A and B be connected with Pitot tubes in a main; for, in any of these cases, the H values represent velocity heads, the same as they do for Venturi tubes, and the velocity heads are always proportional to the square of the velocity producing the head.

It is perfectly practical to have x less or greater than h_2 , for, if we assume $cx^2 = h_2$, equation (1) becomes:

$$y^2 = \frac{r^2}{2\sqrt{c}\sqrt{h} + c/4} \quad (2)$$

which is the more general form, in which we can give to c any value, consistent with practical limitations. For general use, however, equation (1) is satisfactory, particularly as there is no value of c which increases or diminishes the accuracy of the apparatus.

In these calculations, the radius of the pipe, P , within the float, is assumed to be so small that it does not affect the results. Even if no correction is made for this volume, the error at low velocities would be negligible, and, at maximum veloci-

¹If the float were not free to move vertically, that is, if it were restricted in any way, as with a variable lever arm, counterweight or spring then the movements of the liquids would not follow the law stated, but, as the difference of pressure became greater, the surface level of the heavy liquid would fall below a fixed level on the outside and rise above a fixed level on the inside.

²The proof of this equation which is given in the paper is here omitted.

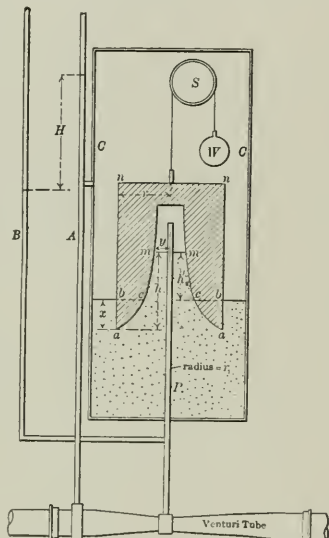


FIG. 1—METER REGISTERING THE FLOW OF LIQUIDS THROUGH VENTURI TUBE

ties, it would be less than half of 1 per cent, provided the pipe is made as small as possible. Its effect is to raise slightly the level of the heavy liquid, both inside and outside the float, by the same amount and aggregating the volume of displacement of the pipe P ; and, where it is desired to make this inside pipe P of an appreciably large outside diameter, allowance must be made for its disturbing influence.

The final rationally derived formulas for that purpose are as follows:

$$x^2 = h_2$$

$$h = x + x^2 + \frac{r_1^2 x^2}{y^2 + R^2 - r_1^2 - r^2}$$

$$x = \frac{(r^2 - y^2)(y^2 + R^2 - r_1^2 - r^2)}{2y^2(y^2 + R^2 - r_1^2 - r^2) - 2r_1^2(r^2 - y^2)}$$

in which the only variables are x and y .

h_2 = the mercury rise as before;

x = the vertical movement of the float;

and

y = the variable radius of the interior.

The constants are:

R = radius of cylinder containing float;

r = radius of outer surface of float;

and

r_1 = radius of pipe P .

To use these formulas practically, it is necessary to tabulate x and y , h_2 and finally h ; h representing the abscissas, as before, and y the ordinates. The necessity of this complication can be avoided by enlarging the outer cylinder at the location of the surface of the mercury.

Although the writer has devised many other forms and modifications, the form of float illustrated in Fig. 1, and covered by

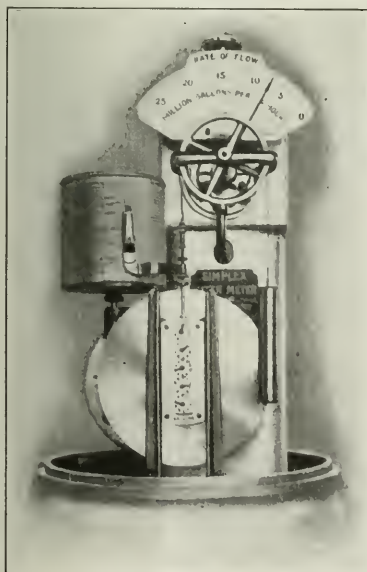


FIG. 2—TAP OF METER REGISTER WITH GLASS DOME REMOVED

equations (1) and (2), is believed to be by far the most practical, because it provides for maximum increments of volume and consequent accuracy at minimum velocities; and, with a float of a given radius, the minimum clearances are possible with the smallest cylindrical casing; also there are several other practical advantages of importance.

One of the first difficulties to overcome was to prevent air from accumulating at the upper interior of the float in Fig. 1.

Such an accumulation, of course, would lighten the float and therefore vitiate the results. Two check-valves opening upward are placed in the cap of the float. These are opened automatically when the velocity head reaches the maximum, and they can be opened at will, by a hand device, from the outside of the bottom of the pressure chamber, without otherwise dismantling the meter. This same device also automatically prevents the loss of mercury when the flow reaches an amount

beyond the maximum capacity of the meter register, as it equalizes the pressure within and without the float.

An additional device for this purpose has been designed to prevent loss of mercury due to a sudden break of either of the pipes leading from the meter register to the Venturi tube, but these are safety devices of not unusual novelty, and, therefore, a detailed description of them is unnecessary.

Fig. 2 shows an application of this meter register.

The float is made of hard rubber to conform to a metal template; but, to

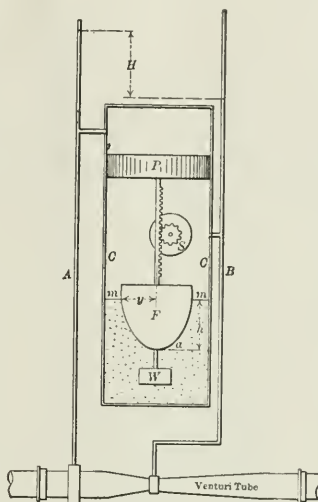


FIG. 3—ORIGINAL SIMPLE APPARATUS

make sure that it is correct and will perform according to the theory on which it is designed, each float is tested in a regular working machine by applying, under pressure, piezometric heads corresponding to the velocity heads of the Venturi tube or Pitot tube with which it is to be used, and noting from a standard rate-of-flow dial the corresponding velocities, which are compared with the theoretical velocities.

For heads, H , greater than $\frac{1}{2}$ ft. the coefficient of velocity through a standard Venturi tube is so nearly constant as to secure results within 2 per cent of the actual. However, with the apparatus above described, Venturi heads, H , of less than 0.01 ft. are readily integrated. The velocity coefficient through the Venturi tube, however, is materially different, amounting to as much as 7 per cent at low velocities, and the instrument is modified accordingly so as to produce accurate results down to less than 0.01 ft.

The data for this modification, as well as the method of making the correction, are intended to be the subject of a later paper.

Leading up to the final development of the apparatus illustrated in Fig. 1, it will be interesting to consider two of the earlier forms. Fig. 3 shows an extremely simple apparatus, which, at first sight, would appear to be superior to the device shown in Fig. 1. P is a piston, F is a shaped float. S is a sheave, and W is a displacement device, all within a casing. The upper portion of this casing is connected with a pipe from the full section of the Venturi tube, and the portion below the piston P is connected with a pipe, leading to the throat of the Venturi tube. The differential pressure or Venturi head, H , acts on the piston and forces the float, F , down into the mercury.

As before, the problem is to shape the float so that its vertical movement from a fixed line of reference will be proportional to the square root of H . The displacement device, W , is such that when there is no flow in the Venturi tube and the apparatus is in equilibrium, the bottom of the float, a , will be tangent to the upper surface of the mercury. The level of the

mercury, m , will change, of course, as the float descends. The equation of the float, though rather more difficult to deduce, is as follows:

$$y^2 = e - \frac{f}{\sqrt{h+g}}$$

e , f , and g being constants depending on the size of the piston, P , the assumed movement of the float, F , with the given Venturi head, H , and the relative weights of mercury and water.

The only radical difference in form between this formula and the one corresponding to Fig. 1 is in the constant e before the fraction. The curve, however, is still one of the fifth degree.

The objection to this apparatus is the friction of the piston P . This was recognized immediately, and, to overcome it, a diaphragm of an accordion or some very flexible type was substituted for the piston; this, however, introduced difficulties caused by the elasticity of the diaphragm and the necessity of modifying the shape of the float to provide for these variations, as the diaphragm could never be depended on to remain in a permanent position.

Another form is shown in Fig. 4. This is an entirely different principle, but the treatment of the problem is analogous to that of Fig. 1.

C represents a casing within which is contained a fixed displacing device F and a movable cylindrical vessel P containing, say, mercury buoyed up by the displacing volume W . The pipe leading from the full section of the Venturi tube connects as shown above the vessel, and the pipe leading from the throat of the Venturi tube connects within the displacing device F .

The differential pressure, or Venturi head, H , forces mercury from the outside of the displacing device F to the inside; but for equilibrium, the depth of the mercury in the vessel P must always be the same; that is, d must always be constant.

With these data it is found that the equation representing the inner curve of the displacing device is exactly the same as that of the float in Fig. 1, namely, equation (1) given before.

This device, in some respects, is superior to any of the others considered, and, in fact, is was the one of which a complete working model was first made and tested, and the results were good. The only objection is the large quantity of mercury required for the displacing device, W .

It is obvious that the vertical movement of the cylindrical vessel, P , may be guided by rollers, as shown, or in some other practical manner. The cross-section of the rod, I , need not be sufficiently large to make its variable displacement affect the results appreciably.

However, the form in Fig. 1 is that which has been adopted as the most practical; and of all the twelve other devices, each of which has been developed by the writer to a working design, that shown in Fig. 4, next to that in Fig. 1, is the most meritorious.

In all cases the registering mechanism outside of the casing may be exactly the same as shown. It comprises a rate-of-flow dial, a chart device to record the rate of flow, and a total-flow dial. The first requires only a graduated dial and a hand actuated by the shaft of the sheave, S . The second requires a

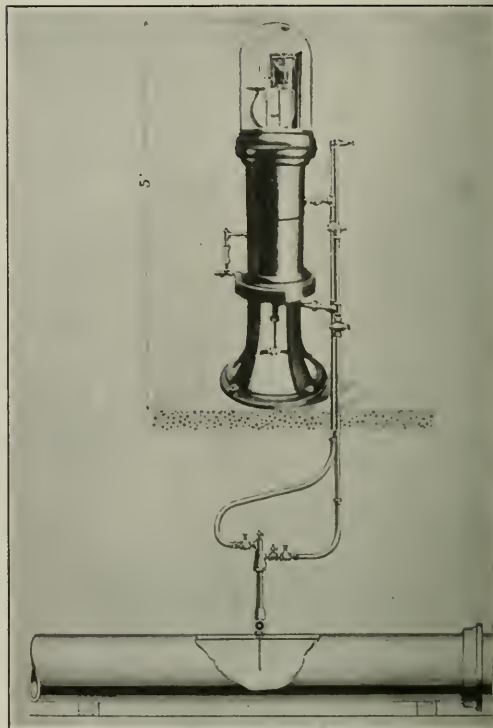


FIG. 5—STATIONARY METER WITH PITOT TUBE

pen reciprocating along the surface of a revolving figure having straight-line elements, such as a cylinder. The third requires a dial having hands which always revolve at a speed proportional to the angular position of the sheave, S .

This is readily accomplished by hanging from the sheave, S ,

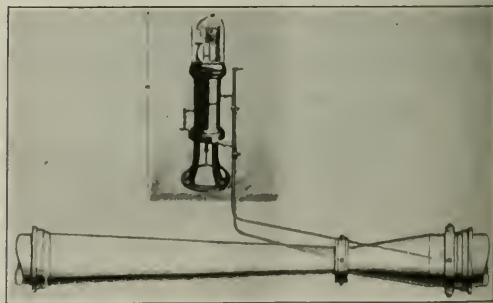


FIG. 6—STATIONARY METER WITH VENTURI TUBE

the dial which is connected with and operated by a traction wheel, the latter being rotated by a constantly revolving disk. When the sheave, S , sets at zero, the traction wheel stands at the center of the disk, and, therefore, has no rotating motion. As the flow increases from zero and the sheave, S , revolves,

the traction wheel moves away from the center of the disk and therefore revolves at a speed proportional to its resistance from the center of the disk and to the rate of flow through the Venturi tube.

There are other equally simple means of actuating the total-flow dial, but they are all on the same principle, and have been in use for several decades. The revolving flat disk is probably the oldest, but the revolving cone rivals it in antiquity. The Nicholson ship's log is a rather recent application of the latter device. It will be seen from Fig. 2 that the chart-recording cylinder and the disk are operated by the same clockwork.

Where it is intended to meter a liquid of a specific gravity different from that for which this meter is adjusted, as, for

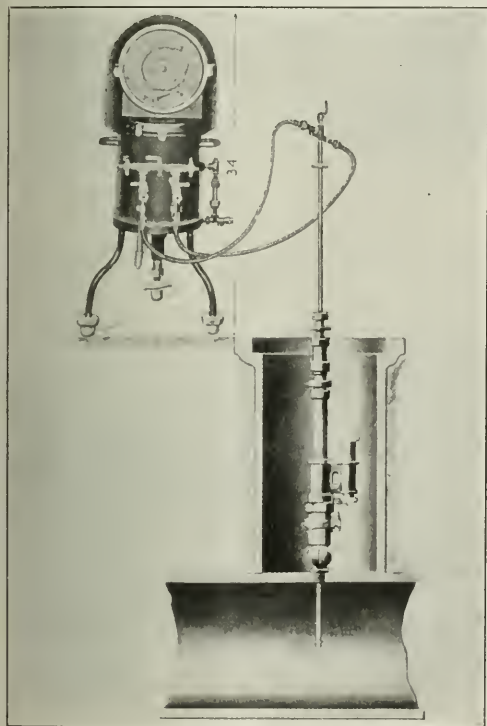


FIG. 7—PORTABLE PITOT RECORDER FOR MEASURING AND RECORDING FLOW OR LEAKAGE OF WATER MAINS, FLUMES, PENSTOCKS, ETC.

instance, if the meter has been adjusted for measuring water at 50 deg. Fahr., and it is desired to measure water having a temperature of 200 deg. Fahr., the results can be corrected according to the following principle:

In a fluid passing through a Venturi tube or orifice, the weight is directly, and the volume is inversely, proportional to the square root of the unit weight of that fluid, for:

Letting

W = the weight of fluid,

Q = the volume of the fluid,

H = the Venturi head,

and

w = the unit weight of fluid:

then

$$Q \propto \sqrt{H}$$

$$H \propto \frac{1}{w}$$

$$W \propto Q$$

From these equations we obtain

$$Q \propto \frac{1}{\sqrt{w}}$$

and

$$W \propto \sqrt{w}$$

The latter two equations are the stated relations.

TABLE 1.—TEST OF A METER REGISTER BASED ON VENTURI HEADS.

FLOAT NO. 46.				
C	V	V_1	D	P
0.010	0.195	0.194	+0.001	+0.52
0.013	0.226	0.225	+0.001	+0.45
0.023	0.300	0.298	+0.002	+0.67
0.036	0.381	0.379	+0.002	+0.53
0.054	0.470	0.467	+0.003	+0.64
0.078	0.571	0.562	+0.009	+1.60
0.112	0.677	0.675	+0.002	+0.30
0.154	0.800	0.797	+0.003	+0.38
0.208	0.931	0.930	+0.001	+0.11
0.270	1.079	1.080	-0.001	-0.09
0.368	1.240	1.240	0.000	0.00
0.487	1.429	1.428	0.001	0.00
0.638	1.635	1.635	0.000	0.00
0.833	1.872	1.874	-0.002	-0.11
1.092	2.150	2.133	+0.007	+0.33
1.426	2.460	2.460	0.000	0.00
1.873	2.820	2.825	-0.005	-0.18
2.455	3.250	3.235	+0.015	+0.47
3.250	3.750	3.720	+0.030	+0.81
4.320	4.321	4.280	+0.041	+0.96
4.830	4.570	4.540	+0.030	+0.66
5.800	5.010	4.970	+0.040	+0.81
7.270	5.638	5.600	+0.038	+0.68

Table 1 contains data relating to a typical test¹ of meter registers constructed on the principle described herein, in which

C = the Venturi head as measured, in feet;

V = the theoretical velocity through the full section of the Venturi tube, in feet per second;

V_1 = the velocity as indicated by the meter register;

D = the differences;

and

P = the percentages of error.

This meter register is in use in several important cases for recording and integrating the flow measured by Pitot tubes, and has also recently been adapted in portable form utilizing Pitot tubes for city main leakage survey purposes. This form is shown in Fig. 7. Quite a number of these instruments are already in service. (A typical test of one of these recorders is given in the paper, but not reproduced here.)

The Carnegie Institute of Technology has issued an instructive pamphlet entitled "Suggestions Concerning a Course in Engineering." Adaptability for engineering work consists in the possession of powers of observation and reasoning; interest in the discovery and application of nature's laws; a strong imagination, ingenuity and resourcefulness; keenness to see the application of mathematical principles to mechanical problems; and practical common sense combined with habits of care and accuracy. In addition to outlining the requirements for engineering in general, the pamphlet then gives the specific requirements for chemical, civil, mechanical, electrical, commercial, metallurgical, mining and sanitary engineering, and states the facilities offered for study in these courses at the Carnegie Institute of Technology, Pittsburgh.

Indian pig-iron is being imported into the United States in small quantities, as an experiment to see whether Bombay producers can compete in this market. About 200 tons has been sent to San Francisco, and more will follow if the iron is sold at satisfactory prices. The Indian iron is said to be of superior quality, and it is on this that the producers are relying to create a demand for their product. The cost of production in India is quite low, but the transportation and duty are high.

¹The paper contains in a table the results of a second test which are not reproduced here.

Evolution of Methods of Handling Slime. IV.

THE RAND, SOUTH AFRICA

By H. N. Spicer

The marvelous development of the resources of the Transvaal dates from about 1883, at which time the first stamp mill was erected to treat gold ore from the De Kaap goldfield. Many interesting comparisons could be made between condi-

tion of the launder and the surface of the slime. The operation was crude compared with modern methods of continuous thickening and decantation, but it shows from what insignificant beginnings the modern method of slime treatment has developed.

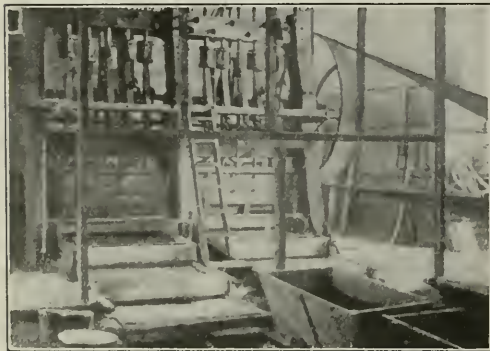


FIG. 1.—BROWN'S HILL 10-STAMP MILL, ERECTED IN 1883

tions existing then and now, showing the gradual development of metallurgical methods and the increasing magnitude of the industry. It will suffice, however, to give the reader an impression of the wonderful development on the Rand by showing in Figs. 1 and 2 views of the first 10-stamp mill erected in 1883, and one of the modern 600-stamp mills built in 1911. Comparison of the two will give an idea of the large scale of present operations; and this can be reinforced by statistics which show that in 1912 there were over 10,000 stamps in daily operation, and that the value of gold produced in that year was nearly \$200,000,000, recovered from about 25,500,000 tons of ore.

First Slime Treatment in 1894

The first cyanide plant on the Rand was built in 1890 at the Salisbury mine, but at that time and for several years thereafter only sand was treated by leaching. It was not until 1894 that any effectual attempt was made to treat slime. In that year Mr. W. Bettel did some interesting work at the Primrose mine, using an excess of lime to accelerate slime settlement. His experiments demonstrated the value of this treatment, and probably initiated an important step in cyanidation which made slime treatment a commercial success on the Rand. In this connection the views shown in Figs. 3 and 4 are of historic interest. Fig. 3 shows a bird's-eye view of the first slime treatment plant, at the Primrose; and Fig. 4 a closer view of

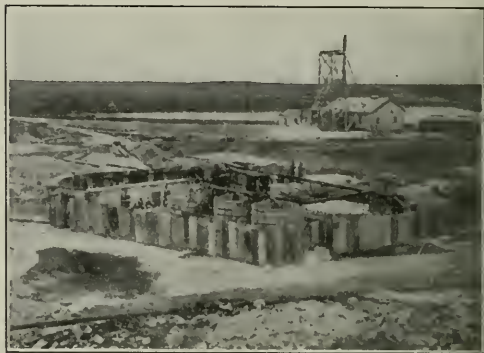


FIG. 3.—BIRDSEYE VIEW OF FIRST EXPERIMENTAL SLIME PLANT, PRIMROSE MINE, 1894

The practical application of the experimental work done by Mr. Bettel fell to the lot of Messrs. Williams and Butters, who initiated the intermittent decantation process which today is used in practically all the mills of the Rand.

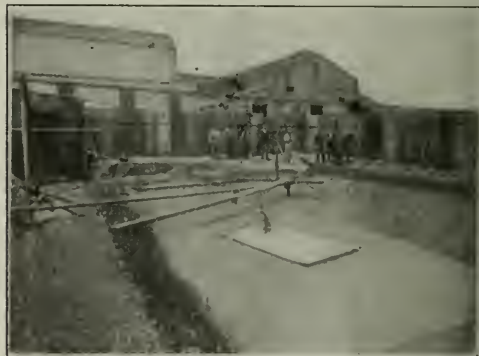


FIG. 4.—SLIME SETTLING BASIN AT PRIMROSE MILL, SETTLEMENT WITH LIME

Effect of Local Influences

For many years during the early history of the Rand, the development of the cyanide process was largely in the hands of local metallurgists, and was affected but little by the practice in other countries. And while it is true that the metallurgy of gold ores has been conducted there on a scale not equalled in magnitude elsewhere, nevertheless it has not kept pace, on the whole, in the adoption of methods and machines that are in daily use in other countries, and this is particularly true with

respect to the handling and the treatment of slime. During the last year or two some very marked changes have taken place in Rand metallurgical practice, and there is little doubt that more modern methods will be adopted in the



FIG. 2.—RANDFONTEIN CENTRAL 600-STAMP MILL

a settling basin in which the slime pulp was mixed with lime and allowed to settle. The supernatant clear water was decanted through a submerged launder, and a pump was used to remove the balance of the water remaining between the level

mills built in the future. The successful features of cyanidation as practiced in other countries are gradually finding acceptance, and will replace the less efficient local methods.



FIG. 5.—TUBE MILLS AT THE BRAKPAN MILL

Crushing in Water is General Practice

With one or two exceptions it is common practice on the Rand to perform the initial crushing and grinding in water. This entails the subsequent removal of water from both sand

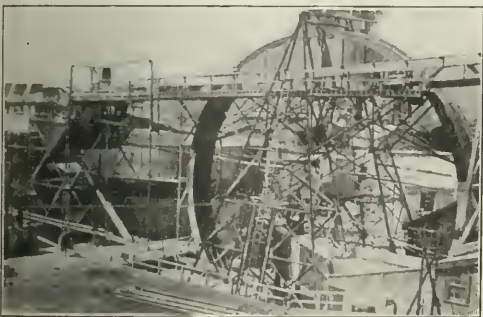


FIG. 6.—TAILING WHEEL ELEVATING PULP TO CLASSIFIERS AT NEW GOEH

and slime before cyanide solution can be applied, and introduces a step which is difficult of satisfactory accomplishment, particularly in treating slime by intermittent decantation. The first attempt at crushing in cyanide solution was made at the New Goeh in 1906, but for various reasons it was abandoned.

Heavy gravity stamps and tube mills are used as crushing machines throughout the Rand, and some very large installations of this type have been made. Formerly, as elsewhere, amalgamation followed stamp milling; but with the introduction of the tube mill at the Glen Deep in 1904, and its subsequent general adoption, the practice was changed, and today it is customary to amalgamate after tube milling. A marked change also has been made in the area of amalgamation surface used; and even in mills of recent construction, such as the Randfontein Central, more than half the amalgamation

area is not in service. Rand tube mill practice has become standardized, and the customary type of tube is 22 ft. long and 5 ft. 6 in. in diameter. Six of these are shown in a view of the unfinished Brakpan mill, Fig. 5.

A common method of elevating the mixed sand and slime pulp from the grinding machines to the classifiers is by the well known tailing wheel. The latest design is similar to a huge bicycle wheel, as shown in Fig. 6.

Cone Classification of Sand and Slime

Cones are generally used as sand-slime classifiers. From the view given in Fig. 7, which shows the system in use at the City & Suburban, the reader will gain an idea of the size of the cones. The system has its disadvantages in the neces-

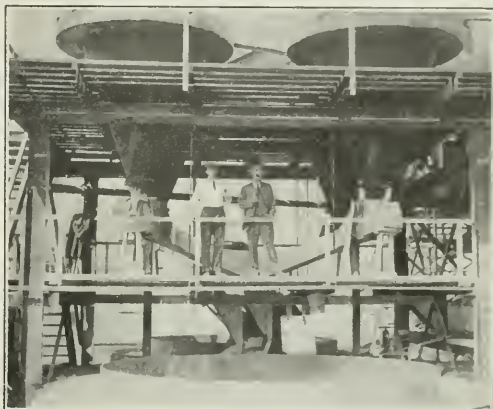


FIG. 7.—CONE CLASSIFIERS AND SAND FILTER TABLES AT THE CITY AND SUBURBAN

sary loss of head, and in the double treatment which must be adopted in order to remove the last portion of fine sand from the slime.

Rotary, vacuum filter-tables of local invention are used to dewater the sand. As shown in Fig. 7, they are placed below the cones and receive their underflow. These machines have a peripheral filtering surface about 30 in. wide, constructed in the usual manner and connected with a suitable vacuum pump. They do excellent work, reducing the moisture in the sand to an average of 13 per cent. The cost of operation, however, is considerable, as a 20-ft. table consumes about 10 hp for vacuum and rotation. The cost of maintenance also is high, as the filter cloth has to be removed and cleaned once in 24 hours, due to the fact that the classified sand still contains



FIG. 8.—SAND LEACHING VATS AT RANDFONTEIN CENTRAL

enough slime to clog the pores of the filter. As a consequence it is necessary to have at least two tables in any installation.

The necessity for obtaining such a low percentage of mois-

ture in the sand is a consequence of crushing in water. If the ore were originally crushed in cyanide solution, the necessity would not exist, and the filter-tables could be dispensed with.

The dewatered sand is mixed with cyanide solution and either pumped or raised by a wheel to collecting tanks, from

weakest point in Rand metallurgical practice. Owing to the fact that the ore is crushed in water, the necessity arises for preliminary thickening by settlement, and the decantation of the water before cyanide solution can be added. In order to aid settlement as much as possible, and at the same time neu-



FIG. 9.—SAND LEACHING VATS AT ROBINSON DEEP, WITH BELT CONVEYORS

which the solution overflows through gates. It is customary to give the sand a double treatment: either in superimposed sets of vats, as shown in Fig. 8, or in separate sets as in Fig. 9. In the first case the sand is shoveled from the upper to the lower vats after the first treatment; in the second it is transferred by means of belt conveyors, as shown in the figure. With but few exceptions the collectors and leaching vats are unloaded by hand.

Variable Quantity of Slime Produced

The quantity of slime produced in the fine crushing of Rand ores depends largely on the depth at which the ore is mined. Ore from the outcrop or shallow mines will produce as much as 35 per cent true slime, whereas the hard ore from the deep mines produces none at all, the so-called slime being only very fine sand. Pyrite is found in some of the deep-level ores in quantities as high as $2\frac{1}{2}$ per cent, but it offers no obstacle to extraction if it is ground very fine. Owing to



FIG. 11.—ROBINSON CYANIDE PLANT FOR DECANTATION OF SLIME

tralize the acidity of the ore, lime is added at the stamp mills. It is imperative that the slime be dewatered to a minimum percentage of moisture, as otherwise large quantities of cyanide solution would have to be wasted continually.

Intermittent Decantation

The cyanidation of slime by intermittent decantation requires large areas of collecting and settling tanks. These are characteristic of all Rand cyanide mills, and form a large part of every plant. Typical views of such installations are shown in Figs. 10 and 11.

The size of slime tanks usually found on the Rand is from 50 ft. to 70 ft. diameter and 12 ft. to 15 ft. deep. The bottom is slightly conical to facilitate discharge of the settled slime.

The diluted slime pulp overflowing from the cone classifiers enters the collecting tanks at one or several points near the center, flowing into wells which restrict undue agitation of



FIG. 10.—SLIME VATS AT VILLAGE DEEP



FIG. 12.—SLIME TREATMENT VATS AT THE BRAKPAN MINES

the wide difference in the classes of ore, there is considerable variation in the rate of slime settlement at the different mills.

Until within the last two years there had been but little real change in the method of treating slime since the day when Butters and Williams introduced intermittent decantation. This system offers great disadvantages and is perhaps the

the charge and prevent disturbance in the settling area. The inflow is so regulated that a perfectly clear overflow is secured at the periphery of the tank. When the overflow will no longer run clear, the feed is diverted to another collector, and settlement is allowed to proceed. The time allowed for this operation depends on the nature of the slime and varies

between wide limits. As soon as settlement is complete the supernatant water is decanted through adjustable pipes.

In the foreground of Fig. 12 are shown some slime collectors arranged for loading through central wells. The rear tank at the left is being filled, and clear water is overflowing at the periphery. The corresponding tank at the right is

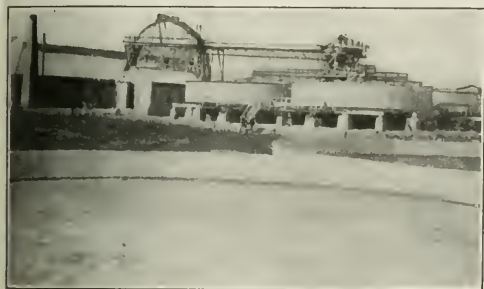


FIG. 13.—AGITATION IN SLIME VATS AT ROODEPORT UNITED

being decanted. Incidentally, the view shows one of the methods adopted for stacking sand tailing.

As mentioned before, the rate of settlement and the density of the settled slime vary widely with the nature of the ore. Speaking broadly, it is found that slime produced from the outcrop or shallow mines will settle to about 50 per cent moisture, while that from deep mines will settle to 42 per cent or 45 per cent moisture. At some of the plants advantage is taken of the fact that the rate of settlement increases with the temperature, and various methods are adopted to heat the pulp.

Following decantation of the water, the settled slime is sluiced from the collector with cyanide solution, and transferred by centrifugal pumps to the first treatment vat in which the operation of settling and decantation is repeated. The strength of solution varies from 0.0005 per cent to 0.02 per cent KCN, and the ratio of solution to solids is 4 or 5 to 1.

Agitation Not Generally Practiced

As a whole, the slime from Rand ores requires but little agitation, and until recently no special provision has been made for that operation. Dissolution of the gold occurs during the



FIG. 14.—PACHUCA AGITATORS AT FRENCH BOB'S

transfer of pulp from the collectors to the treatment vats. This, however, is frequently insufficient to dissolve the maximum amount of gold, and is commonly supplemented by circulating the charge in the vat. This operation is shown in Fig. 13, which shows the swirling motion imparted to the contents of the vat.

The time available for this operation seldom exceeds three or four hours, as the vat is needed for the next lot of collected slime. In addition to the fact that the time is insufficient to accomplish the purpose, the shape of the vat is ill adapted to keeping slime in suspension, and thereby effectually aid dissolution of the gold. Realizing the deficiency of the method, and the advantage to be gained in some cases by genuine agitation, some companies have adopted the Brown or Pachuca agitator, as shown in Fig. 14.

Returning to the decantation process: After the charge is agitated, or rather mixed, the slime is allowed to settle and the clear solution decanted. The thick slime is then sluiced from the vat with barren solution, and again transferred by centrifugal pumps to a second treatment vat. But little circulation is required in this vat, as most of the gold has been dissolved and removed; and when the transfer and mixing is complete, settlement and decantation proceed as before. The only solution precipitated is that decanted from the first treatment. Ordinarily this is clarified by passing through sand filters, but lately plate and frame presses have been adopted for this purpose.

Usually this ends the treatment of the slime, but in some instances a third dilution is required to remove the last portion of dissolved gold. In that event the settled slime is again transferred with solution to a third vat where the same operation is repeated. Finally the impoverished slime is sluiced

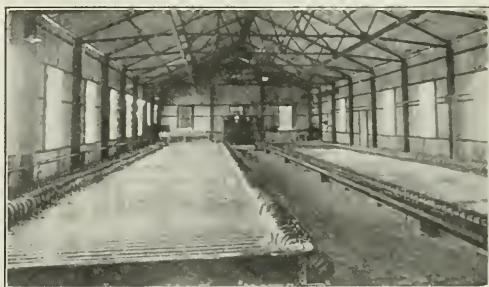


FIG. 15.—BUTTERS' VACUUM SLIME FILTER AT THE BRAKPAN MINES

from the vat with the least water possible, and impounded in dams from which considerable water is recovered and pumped back to the mill. The solutions decanted from the second and third dilutions are variously used as wash solutions in sand treatment.

It will not be surprising to the reader to learn that attempts have been made from time to time to supplant this cumbersome method of slime treatment. As far back as 1803 filter pressing was introduced for this purpose. Continuous decantation in cones also was tried, with the idea of reducing the value of the slime before filter pressing. These attempts proved abortive, however, largely on account of the mechanical difficulties encountered in the use of cones, for at that time no better device than the cone was known for continuous decantation. Had the modern continuous thickener been known then, it is possible that the whole history of slime treatment on the Rand would have been changed. Some of these modern machines have been used recently and probably will be more extensively adopted.

Vacuum Filtration in Newer Mills

It was not until 1910 that the first vacuum slime filter was installed on the Rand, despite the fact that the system had long been in successful operation in other countries. The first installation aroused some pessimistic and skeptical comment, but the success of the process was such that today it is the greatest rival of intermittent decantation on the Rand. Eight of these plants are now in operation, treating an aggregate of 6500 tons of slime per day, and there is every indication that

the process has become an integral part of Rand slime treatment. Fig. 15 shows a view of the Butters filter at the Brakpan mines, handling 1000 tons of slime per day.

Notes on Chemistry and Metallurgy in Great Britain

(From Our Special Correspondent)

Iron and Steel Institute

The annual meeting of the Iron and Steel Institute was held on May 1st and 2nd at the Institute of Mechanical Engineers, under the presidency of Mr. A. Cooper.

The Bessemer Medal was awarded to M. Adolphe Greiner.

The first paper read was on "The Production of Sound Steel by Lateral Compression of the Ingot while its Centre is Liquid." The author, Mr. Benjamin Talbot, advocates the advantage of using aluminium and similar deoxidizers to reduce the amount of segregation and prevent blow-holes in the outer portions of the ingots; and he also employs a process of lateral compression after the ingots have been in the soaking pit for a definite time and before central solidification occurs.

The discussion was opened by Sir Robert Hadfield, who asked if the author had taken the casting temperature into account, what would be the effect of a greater percentage of phosphorus and if traverse tests and tests to destruction had been taken.

M. Adolphe Greiner remarked that whereas the author indicated that the object of his paper was to give a comparison between his own method and the result of the present-day practice, he did not really give any such comparison, and he therefore asked him to do so.

Mr. H. Savage referred to trials made at Seraing, the results of which indicated inferior strength of rails produced from laterally compressed steel as compared with those from ordinary metal.

Mr. R. T. Scott remarked that sections of a very large number of ingots compressed by the Harnet process, when sliced down the middle and planed to the exact centre, did not show any line of demarcation or segregation like those which Mr. Talbot had found in his ingots.

Mr. F. W. Harbord said he had examined specimens of rails and ingots from steel made by the author's process, both with and without aluminium, and it seemed that it was necessary in order to get a sound outside to have some deoxidizing agent. The difficulties from the practical point of view were very great, and it remained to be seen how far they could be overcome in every-day practice. It appeared to him that there would be always a danger that if the ingots were squeezed too soon they would bleed, or that the segregate would squeeze out too near the surface; and, on the other hand, if the squeezing were done too late the piping would not be closed up.

Mr. J. M. Gledhill said that his firm (Sir W. G. Armstrong, Whitworth & Co.) had continued to carry out compression in all their ingots ever since the introduction of compressed steel. He believed that Mr. Talbot was on the right track, but the difference of casting temperature gave rise to the greatest possible trouble. He had always found that one of the greatest advantages of compressed steel ingots was the entire absence of blow-holes on the surface.

Mr. F. W. Paul said that the bleeding effect seemed to him to be one of the most serious difficulties in practically carrying out the new process. He had frequently analyzed metal which exuded out; and even with a soft ingot, containing an average of 0.17 carbon, segregation had taken place up to as much as 0.6 carbon, and the portion which had bled out was the highly segregated portion. Another important matter which was mentioned by Mr. Talbot was that the process depended on a time-table, and this, he thought, would prove the crux of the whole question; because the physical properties of molten steel varied very much, and what was known

as a lively steel took much longer to cool than a steel which was termed dead.

Mr. E. H. Saniter said that the liberation of nitrogen by aluminium, mentioned in the paper, appeared to him to be a novel idea, and it seemed to him that if aluminium disengaged gas from the steel it was likely to make blow-holes rather than solid metal.

Mr. C. J. Bagley said it was well known that when pigs were broken from the sow some liquid metal escaped which was always found to contain more phosphorus and sulphur than any other part of the pig or sow. This showed that the segregate was squeezed out from the centre, which was the last part to solidify. This was the worst position for the segregates, but by Mr. Talbot's process they were taken from the centre to a position between the centre and the outside, where they were harmless.

The president remarked that the author spoke of experiments made with a 25-inch ingot. That was rather a large size, and there were not many mills in this country which could deal with it. He would like to ask Mr. Talbot whether he had arrived at any conclusion as to how much smaller ingots could be made for use with his process. It was clear that an ingot of 10 or 12 inches could not be successfully worked, because it would not be possible to maintain the temperature requisite to roll out the piping.

Mr. Benjamin Talbot, in his reply, said that with regard to the question of casting temperature he had found the use of a pyrometer unsatisfactory, and the temperature he employed was the practical temperature at which a clean ingot was cast. With regard to high phosphorus steel he presumed that reference was intended more especially to phosphorus in American steel, which for the last year or two was for Bessemer steel 0.10; but they had only to look at the present records of the American rail trade to see that it was reverting as quickly as it could to the basic open-hearth process. This process could not be adapted to every kind of ingot. There must be uniform conditions and, therefore, necessarily a certain size, and he had started on a size of ingot which worked successfully.

The next paper taken was by Mr. Andrew Lamberton on "A New Form of Electrically-driven two-high Continuous-running Reversing Mill," but the discussion did not afford any points of more than ordinary interest.

Mr. P. Longmuir next read a paper entitled "Studies in the Cold Flow of Steel," in which he also dealt more particularly with the effects of silicon, aluminium, and high carbon, and said that he found that the effect of cold working on the tensile strength of steels ranging between 0.1 and 0.89 per cent of carbon does not show any great variation. A sound uniform hot-rolled section is necessary for a successful flow; and segregated cores may give rise to cupping, which may also follow from excessive reduction at any one drawing. The discussion, in which Drs. Rogers and Rosenhain, Mr. Saniter and Sir Robert Hadfield took part, dealt with the question of stresses in drawing and its effects on crystalline structures.

Professor J. von Ehrenwerth then read a paper on "The Economy of Dry Blast," which, he contended, saved fuel caused less strain on the blowing engines, produced a higher temperature in the melting zone, with more regular working of the furnace. In the discussion Mr. Greville Jones agreed that the dry blast was advantageous for ferro-silicon and ferro-manganese.

Mr. A. K. Reese said that each particular furnace had its own maximum temperature of best work, and moisture in the blast caused fluctuations in the melting zone which would not be caused by the dry blast.

Mr. Hutchinson did not think that dry blast would be economical in the Cleveland district, where they had a fairly dry atmosphere.

The next paper read was by Mr. W. H. Hatfield on the "Influence of Sulphur on the Stability of Iron Carbide in the Presence of Silicon." The author concludes that sulphur augments the stability of iron carbide at high temperatures.

possibly in consequence of a small quantity of the sulphur remaining associated with the carbide crystals, but the action is of a chemical nature; and Levy's suggestion of mechanical action by sulphide films is not supported; the effect of sulphur is largely neutralized by silicon and by manganese.

In the discussion Dr. Rogers suggested that the addition of sulphur in certain other forms would perhaps have resulted in the melting point of such additions not being reached as it had been in the author's experiments.

Dr. Rosenhain, Mr. H. I. Coe and Mr. Hatfield also took part in the discussion, which concluded the first day's papers.

SECOND DAY

The proceedings of the second day began with the presentation of the Andrew Carnegie gold medal to Dr. J. Newton Friend.

The first paper was on "Some Fundamental Faults of the Present Day Furnaces and Their Remedies," by Mr. **Alleyne Reynolds**, who advocated the production of the reducing atmosphere in the Siemens steel furnace and the utilization of the heat of the waste gases for heating the air. He claimed that a saving of about 25 per cent of fuel would be effected.

Mr. A. D. Ellis, who opened the discussion, said that the author's furnace reduced the quantity of scale and slagging and would be most useful for heating high carbon steels and for melting bronzes.

The next paper was on "The Tenacity, Deformation and Fractures of Soft Steel at High Temperatures," by Dr. W. Rosenhain and Mr. J. C. W. Humphrey who came to the conclusion that the behavior of such steel at high temperatures resembles that of an aggregate of crystals contained in a viscous fluid; while when cooled below Ar_3 the crystals are weakened and yield before the cementite does, and consequently the strength of the material is then dependent on the size of the crystal increasing as the quantity of cement diminishes. On the allotropic theory of the hardening of steel the authors only say that one important transformation takes place in iron at about 900° C. and the lesser change at the temperature of Ar_2 , and in view of their results they are unable to support Benedicks' theory of the nature of beta iron.

The discussion, which was of a most spirited nature, was opened by Prof. J. O. Arnold, who said that that they were actually assembled to bury that poor, battered, threadbare old myth hard beta iron, and by the irony of fate the grave-diggers were named Rosenhain and Humfrey, while the officiating clergy were Drs. Carpenter and Stead. They had before them two papers: one of comparatively small importance by Dr. Carpenter, and one of profound importance by Dr. Rosenhain and Mr. Humfrey.

With regard to Dr. Carpenter's paper, he would suggest that the memoir should have appeared under the joint names of Drs. Carpenter and Stead: the voice was Henry's but the hand was the hand of John. Dr. Carpenter's paper might be summed up in saying that in his almost pure iron the absorption curve presented only the point of Ac₃, while the recalescence curve presented both the points Ar₃ and Ar₂. But Dr. Carpenter asserted that the latter was only the end of the former because some groups of gamma molecules had been rendered metastable by investing membranes $\approx$ to 3 molecules thick, resulting from traces of impurities. Ile, in accordance with Dr. Benedicks, said that at Ar₂ the metastable gamma molecules became unstable and changed into the alpha molecules with a slight evolution of heat: consequently there was no such thing as beta iron, but only alpha and gamma. Unfortunately Dr. Carpenter's theory was as thin as his hypothetical membranes, and the application of a small bevelled straight edge showed that Ac₂ extended along a considerable amplitude in all Dr. Carpenter's inverse rate heating curves.

Professor Arnold proceeded to find fault with Dr. Carpenter's statement that the analysis made by Professors Ilicks and O'Shea of the pure electrolytic iron had never been published. That statement was quite unfounded. It was published in the commemoration volume of the University Col-

lege of Sheffield, to which the speaker referred in his British Association lecture in 1970, and also appeared in the discussion on Dr. Benedicks' paper before the Iron and Steel Institute in 1912, to both of which Dr. Carpenter had made reference. After further technical details Professor Arnold said it was perhaps fortunate for Dr. Carpenter that his paper had fallen to the ground, as otherwise he would have to assert that the magnetic change point at Ac₂, or about 760 °C, was really at Ac₃, about 910° C. Equally, he would have laid to rest that mythical constituent martensite, which was supposed to be a highly crystalline solid solution of iron and carbide in beta iron. It was obvious that he could not possibly have a solid solution of carbide of iron in an allotropic modification of iron which, according to him, did not exist. He would also have to admit that 13 per cent of carbide of iron was soluble in the alpha range of temperature and that such solid solution was a micrographically amorphous and quartz-hard substance called hardenite, discovered by Sorby a lifetime ago.

Having thus mildly admonished Dr. Carpenter, Professor Arnold turned his artillery on Dr. Rosenhain and Mr. Humfrey. It would not be possible to do justice to Professor Arnold's criticisms if they were condensed, and your correspondent must therefore rest content for the present with saying that point by point he steadily and mercilessly demolished the authors' arguments and rendered their position untenable.

Engineering Imports and Exports

The returns issued by the Board of Trade for the four months ended April 30th continue to show very satisfactory features. As compared with the corresponding period of last year the imports of iron and steel, including manufactures, showed an increase of £1,552,062, and totalled £5,230,759; while exports amounted to £18,341,167, showing an increase of £3,333,944. In other metals, including manufactures, the imports touched £11,026,974, an increase of £1,117,153, and exports £4,532,727, an increase of £693,970. The imports of electrical good totalled £500,145, and showed a decrease of £24,667; exports went up to £1,916,285, a rise of £473,008. In machinery imports were £2,512,131, an increase of £207,593, and exports expanded to £11,034,657 and increased by £1,564,421. Imports of new ships with a total of £11,508 showed a decrease of £80, while exports put at £2,383,054 increased £666,121.

Taking the month of April alone, imports of iron and steel increased £405,000 and the exports went up by £2,351,651. Other metals increased £560,403 in imports, and improved £327,219 in exports. Imports of electrical goods were £12,255 higher and exports were £123,400 better. The imports of machinery increased £60,184 and the exports by £786,828. Imports of ships went up £7,878, while exports decreased £107,640.

Market Prices

May, 1913

Tin opened high £232, dropped to £230 in a few days, rose on the 7th to £231¹/₂, and dropped away from this point to £221 by the 15th. Then recovered to £224 (10th) and has since, after a drop to £218, kept about the £221 level, closing lower at £212.10.

Copper has been rather unsteady, rising from £67.5 to £69.5 in the first week, appreciating another 15/- in the second, dropping a pound in the third, then recovering a little, and closing £66.15.

Hæmatite has been steady throughout the month, but suffers slightly at the close 77 1/2.

Scotch Pig has declined in the same way as Cleveland, opening at 73 3/8, it closes at 66 1/8.

Cleveland remained steady at 67.0 till mid-month, then dropped to 68.6, recovered about 20th to 70.0, but since declined and sharing in the general depression closes at 69.6.

Lead has made a very steady rise throughout the month, opening £18 10 and closing £20.5 0.

Aluminium, per ton	£ 5 d.
	10 0 0.

Alum, lump, loose, per ton.....	£ s. d. 6. 0. 0
Antimony, black sulphide powder, per ton.....	20. 0. 0
Borax (crystal), per ton.....	17. 10. 0
Copper ore, 10 to 28%, per unit.....	12/4½ to 12,10½
Copper sulphate, per ton.....	22. 15. 0
Carbolic acid, liquid, per gal.....	1. 4
Caustic soda, 70%, per ton.....	9. 12. 6
Ebonite rod, per lb.....	4. 6
Hydrochloric acid, cwt.....	5. 0
India rubber, Para, fine, lb.....	3. 8½
Mica, in original cases, medium, lb.....	3/6 to 6. 0
Petroleum, Russian spot.....	9¼
Quicksilver, bottle.....	7. 10. 0
Sal-Ammoniac, lump, 1st delivered, U. K., per ton.....	44. 0. 0
Sulphate of ammonia, f. o. b. Liverpool, ton.....	13. 0. 0
Sulphur, recovered, ton.....	5. 5. 0
Shellac, per cwt.....	4. 5. 0
Platinum, per oz.....	9. 5. 0
Tin ore, 70%, per ton.....	142 to 147. 0. 0
Zinc, Vieille Montagne.....	28. 12. 6

Differences on the Month

Higher.	£ s. d.	Lower.	£ s. d.
Lead	1. 15. 0	Copper	15. 0.
India rubber	4	Tin	13. 10. 0
		Haematite	1. 3
		Scotch Pig	6. 9
		Cleveland	6. 9
		Aluminum	5. 0. 0
		Copper sulphate	12. 6
		Sulphate of ammonia ..	17. 6
		Tin ore	3. 0. 0

Recent Chemical and Metallurgical Patents

Gold and Silver

Precipitation of Cyanide Solutions.—In the method of precipitating cyanide solutions by means of zinc dust, Mr. CHARLES W. MERRILL, of Berkeley, Cal., has discovered that it is desirable to maintain reducing conditions not only to the entrance of the press, but also throughout the precipitation and filtration in the press. Owing to the fact that some solutions filter freely, and that in some cases the supply of solution to the filter is insufficient to keep the press entirely full, the upper portion of the press is apt to contain air or other oxidizing gases. The effect of this air is to oxidize the mixture or the solid component, and thus decrease the efficiency of precipitation and the cost of the precipitant.

In presses which discharge their filtrate from the bottom, it is not always possible to maintain reducing conditions in the press, and Mr. Merrill proposes to adopt a discharge at or near the top of the press, so that the filter will always be full of the mixture, and thereby avoid the accumulation of air in the press. Any filter or press from which the atmosphere is completely excluded and which is capable of being maintained full of the mixture during the operation of filtration, may be used in connection with this process. (1,063,567, June 3, 1913.)

Alloy of Zinc as Precipitant.—The preparation and use of an alloy of zinc, lead, magnesium, aluminium, or other similar combination, is the basis of several patents granted to Mr. CHARLES W. MERRILL, of Berkeley, Cal. The inventor claims that such an alloy is a more efficient precipitant than zinc alone. In order to prepare the alloy in a finely divided state, it is mixed with an abrasive before cooling, whereupon it can be readily reduced to powder by any form of comminuting apparatus. The use of the abrasive is essential to overcome the natural malleability of the metals which ordinarily prevents their comminution.

Another form of application of the precipitant is to make use of it as a liner and balls for a small tube mill, through which the solution passes. The resulting attrition is sufficient to liberate the metal and allow it to flow with the solution to the

collecting press. The alloy has a galvanic action which makes it a more efficient precipitant than one metal alone. (1,063,568-69-70, June 3, 1913.)

Zinc, Lead and Copper

Coating Metals with Lead.—While there has been comparatively little difficulty in coating metals with tin and zinc, it has been found almost impracticable to coat such metals with a uniform covering of lead by immersing them in a bath of molten lead. The difficulty lies in properly fluxing the surfaces of the metal at the relatively low temperature of molten lead. According to an invention of Mr. SAMUEL EDWARDS, of Pittsburgh, Pa., a suitable flux which will permit of such coating, is composed of the following ingredients: 100 lb. hydrochloric acid in which is dissolved 2½ lb. of zinc, and 100 lb. hydrochloric acid in which is dissolved 1 lb. aluminium and ½ lb. sal ammoniac. The two solutions are mixed. The articles to be coated are immersed in the bath thus formed, and while their surfaces are still wet with the flux, they are dipped in a bath of pure lead heated to a temperature of about 600 deg. F. By the use of this flux it is possible to make as firm and adherent a coating of lead as with zinc or tin. (1,063,248, June 3, 1913.)

Recovery of Zinc and Copper from Brass Waste.—In the ordinary method of melting brass turnings or waste, zinc is largely lost and copper alone recovered. According to an invention of Mr. HARVEY M. BURKEY, of Newark, N. J., brass turnings or other forms of brass waste are mixed with finely divided fuel, such as anthracite coal, and subjected to an oxidizing blast in a converter of the Huntington-Hieberlein type. The temperature and volume of air are so regulated that the zinc is oxidized, volatilized and collected in a suitable system of bags, while the copper remains behind as a sinter which can be charged directly into a reverberatory furnace and refined. (1,061,447, May 13, 1913.)

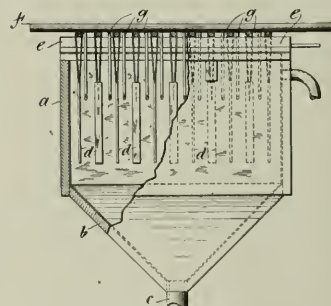


FIG. 1.—METHOD OF CIRCULATING ELECTROLYTE

Roasting Copper Ores.—The conversion of copper oxides or carbonates into sulphates in order to facilitate their treatment by hydrometallurgical methods, is the basis of a patent granted to Mr. UTLEY WEDGE, of Ardmore, Pa. The inventor employs a furnace of the superimposed-hearth type, and mixes copper oxide or carbonate ores with a sulphide such as pyrite. This mixture is treated under such conditions of oxidation and temperature as will convert the copper into sulphate, the iron being first changed into sulphate and then reacting with the copper. In case pyrites is the form of sulphide used, a temperature of 800 to 1000 deg. F. is required. Some sulphur dioxide and trioxide will be formed, and in order to utilize these gases the furnace is designed for a downward draft of evolved gases, so that the material is subjected to the action of the gases from the time it enters until it leaves the furnace. This tends to increase the conversion of copper oxide or carbonate into sulphate. The product is then suitable for leaching and recovery of the copper by any method of precipitation. (1,063,629, June 3, 1913.)

Electrolysis of Copper at High Current Densities.—In the electrolytic refining of copper and the recovery of anode slime, it has been impossible to use currents of high density on account of the uneven deposition of the copper and the consequent development of short-circuits. Agitation or circulation of the electrolyte would avoid the short-circuiting, but would cause

contamination of the cathode with anode slime. In order to overcome these difficulties, and permit electrolysis at high current densities without contaminating the cathode deposit, Mr. KENNETH GUTERMAN, of New York, N. Y., proposes a method of circulating the electrolyte which will continuously remove the liquor to a filter, where the anode slime will be collected, and return a clean electrolyte to the cell. Fig. 1 shows a side elevation of an electrolytic vat, partly in section, with the arrangement of electrodes and feed pipes.

Electrodes are shown at *d*, suspended in the usual manner

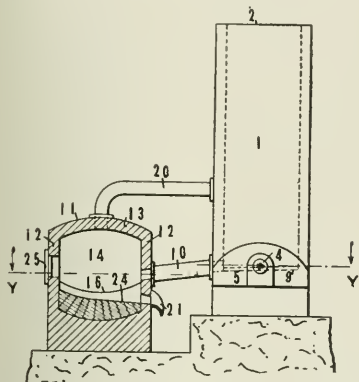


FIG. 2.—FURNACE FOR MELTING AND REFINING COPPER

from bus-bars *c*. The electrolyte is supplied through the pipe *f*, and thence through smaller pipes *g*, the latter being curved at their lower end so as to direct the stream across the electrodes and remove the anode slime. At the same time the electrolyte is constantly flowing out of the vat at *c*, and passing to a filter where the slime is collected, after which the liquor is again returned to the vat through the supply pipes. By means of this process a high rate of circulation is maintained between the electrodes where it is required, so that concentration at the cathode surface is constant. The slime is removed as fast as formed, and a smooth, dense cathode deposit is obtained. The process facilitates the treatment of impure material, and increases the capacity of the plant. (1,062,966, May 27, 1913.)

Melting and Refining Copper.—A process for melting and refining copper has been patented by Mr. DAVID W. BLAIR, of New York, N. Y., consisting in melting the copper in a shaft furnace under oxidizing conditions and then finishing the refining process in a reverberatory furnace which is connected with the vertical furnace. Fig. 2 shows an elevation of the vertical furnace, and a section of the reverberatory. In carrying out the process, the furnace *i* is charged with cathode copper, which

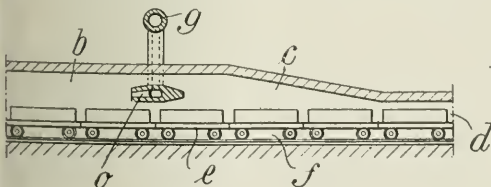


FIG. 3.—TUNNEL FURNACE

melts in the heat generated at the fuel burner 4. Conditions are maintained so that the melted copper delivered to the refining furnace 14 will contain some slightly oxidized copper. A fuel burner situated at one end of the refining furnace maintains the metal in a molten condition. The refining process consists in spreading a reducing agent, such as carbon, over the surface of the bath, and then poling the charge in the usual manner until the oxide of copper is reduced.

By insuring the presence of some oxygen in the copper passing to the refining furnace, a means of controlling the refining process is secured. For example, should the copper in the refining furnace be overpoled, oxidized copper from the melting furnace could be added to correct the condition. The hearth 24, of the refining furnace is formed of wedge-shaped blocks of refractory material, the form of which prevents copper from forcing the blocks out of place, for should any metal find its way beneath the blocks, its pressure would be resisted by the inverted arch structure of the hearth. (1,058,941, April 15, 1913.)

Metallurgical Furnaces

Tunnel Furnace.—An improvement in tunnel furnaces has been patented by Mr. ARTHUR RAMEN, of Olympia, Sweden. The object of the construction shown in Figs. 3 and 4 is to introduce into the combustion chamber the preheated air from the inner end of the cooling chamber, and mix the air with the gas or fuel used in combustion. According to the invention, there is arranged between the cooling chamber *c* and the combustion chamber *b*, a chamber *o*, formed of brick, placed across the furnace, and open to the combustion chamber. Above and beneath the chamber *o*, the combustion and cooling chambers are in communication with each other. Gas is introduced under pressure into the chamber *o* from the conduit *g*, and passes into the combustion chamber between two air currents, thus causing by injector action an abundant supply of air from the inner end of the cooling chamber. At the same time an intimate mixture of air and gas is formed, causing complete combustion and a high temperature. (1,062,606, May 27, 1913.)

Rabble for Mechanical Roaster.—To facilitate the attachment and removal of rabbles from the stirring arms of mechanical roasting furnaces, Mr. MAURICE VAN MARCKE DE LUMMEN, of Cologne, Germany, has designed a special form of

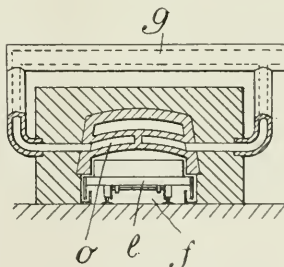


FIG. 4.—TUNNEL FURNACE

stirring device. The lower face of each arm rises obliquely in the direction of rotation, and has an obliquely rising guide for each rabble. The latter, which is loosely inserted in the guide, is thus forced against the under face of the arm by the counter pressure exerted by the roasting material, and is thus safely and firmly secured in position. In order to remove a rabble it is necessary only to withdraw it by means of a hook which can engage with a notch or opening in the rabble. By this arrangement, any rabble can be replaced without disturbing the other rabbles on that arm. (1,061,303, May 13, 1913.)

Combination Blast Furnace and Converter for Copper.—According to an invention of Mr. HOWARD F. WIERUM, of Upper Montclair, N. J., it is proposed to design a blast furnace so that in addition to the regular production of copper matte, the converting process can be carried on, producing either metallic copper or white metal. To this end the furnace is provided with additional tuyeres for the introduction of air and silica into the matte formed in the crucible of the furnace. The advantages claimed by the inventor are the further oxidation of sulphur and iron, the conservation of the heat produced by this oxidation, and the enrichment of the sulphurous gases given off from the furnace.

One form of the invention is shown in Fig. 5, which embodies a blast furnace of the usual form, with ordinary low-pressure air delivered through pipes 12 and tuyeres 15. For the further oxidation of the matte, additional high-pressure air is provided from a supply pipe 18 and tuyeres 16. The crucible 3 is of refractory material, and has a tap for metal at 4, and a

recess and spout for slag at 5 and 6. The sole plate 7 is supplied with cooling water through pipes 11. Receivers 19 are provided for silica which is injected into the matte through the high-pressure tuyeres.

Assuming the furnace to have been charged in the usual manner, but with less carbonaceous fuel than is customary, and an accumulation of matte in the crucible, the converting tuyeres are used, and the matte subjected to a bessemerizing action. Silicious flux also is introduced through the converting tuyeres to slag off the iron. Flue dust also may be thus introduced beneath the molten charge. The smelting and converting operations may be successive, continuous or simultaneous. (1,063,486, June 3, 1913.)

Smelting and Refining Furnace.

Another form of furnace designed to effect the operations of smelting and converting in the same receiver, has been patented by Mr. Edward Fink, of Milwaukee, Wis. As shown in Fig. 6, it comprises a rotary roasting or preheating barrel *a*, and a rotary smelting furnace *b*, both mounted and operated in the usual manner. The smelting furnace is further provided with three sets of tuyeres *w*, one set of which is shown. Blast for the tuyeres is supplied by blower *z*, through the wind box *u* and pipes *v*. Fuel is supplied through the pipe *m* at the opposite end of the furnace.

In operation the ore is fed through the preheating or roasting furnace *a*, being discharged into a hopper *k*, and thence fed into the smelting furnace. As the furnace rotates, the tuyeres pass through the molten bath, and air is admitted to the charge. Rapid desulphurization and smelting takes place, and ultimately the barrel becomes filled with slag and matte. When this stage is reached, the furnace is stopped with all tuyeres above the level of the bath, and the matte is allowed to settle. Fuel is supplied through the burner, and the bath kept fluid so that

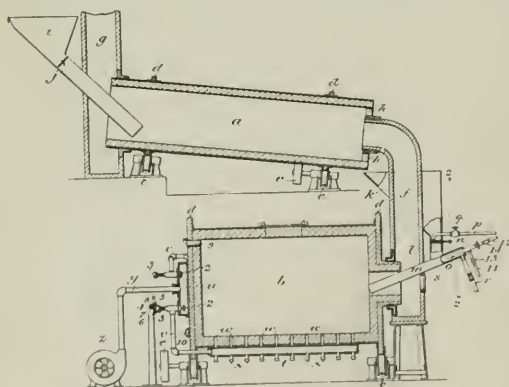


FIG. 6.—COPPER SMELTING AND REFINING FURNACE

settlement of the matte can be complete. The slag is then drawn off through tap holes provided for the purpose at one end of the barrel, and the refining operation begins. During this part of the process the barrel is not rotated, but is rather oscillated so that the tuyeres do not pass through the metal formed. When the matte has been converted into metal, the latter is drawn from the furnace through suitable taps. The slag from the refining operation is used to form a bath for

the next charge of ore. In this manner, the same furnace is made to serve for smelting and refining. (1,061,279, May 13, 1913.)

Chemical Engineering

Electrolysis of Black Liquors of Soda-Pulp Mills.—In order to purify the black liquors obtained in pulp mills from the boiling of wood with lye, Mr. Nils A. Langlet, of Gottenborg, Sweden, proposes to use electrolysis in such a manner as to precipitate the ulmic compounds in a form in which they may be easily separated by filtration. One of the conditions necessary to obtain a filterable precipitate is to increase the salt concentration of the liquor. For this purpose salts such as sodium chloride, sulphate or carbonate are added to the liquor, and the precipitation carried out at a temperature of not less than 65 deg. C. The inventor prefers to use a cell with porous partition between the electrode chambers, and to supply the liquor to the anode chamber only, while the cathode chamber is filled with regenerated soda lye or sodium chloride solution. This solution may be used until the ratio between the caustic soda formed in the said chamber and the remaining quantity of sodium chloride is suitable for boiling cellulose material, whereupon it can be used for compensating losses of alkali in the process. The invention differs from others of a similar nature in producing a filterable precipitate. (1,062,016, May 20, 1913.)

Synopsis of Recent Chemical and Metallurgical Literature

Iron and Steel

Microstructure of Sintered Iron-Bearing Materials.—In the May *Bulletin* of the American Institute of Mining Engineers, Mr. B. G. Kluhn, of Birdsboro, Pa., presents the results of a study of the microstructure of sintered iron-bearing substances, and compares the value of various processes for sintering such materials. The article is illustrated with microphotographs which show the effect of sintering by different methods. The object of the investigation was to determine the relation of the structure of these materials to their reducibility in the blast furnace. A notable fact demonstrated by the study is that the reducibility of the sintered product is inversely proportional to the degree of fusion which occurs in the sintering process.

Thirteen products were examined, formed of different iron-bearing materials and produced by different sintering methods. Those to which the greatest degree of heat has been applied in sintering show clearly the formation of iron silicate as a glass which coats the particles of sinter and prevents the reducing action of the gases in the blast furnace. Thus the agglomerating process which requires the least fuel for effective binding of the particles is not only the most economical in its own operation, but also produces the ideal material for the blast furnace.

The products examined and illustrated are arranged in the order of the degree of complete fusion to which they were subjected in order to effectually bind their particles together. This arrangement also presents the materials in reverse order of their desirability as a furnace feed, and is as follows: Heating-furnace cinder; puddle cinder; Heberlein agglomerate of flue dust; Heberlein agglomerate from pyrites cinder; rotary kiln nodules from red hematite ore; rotary kiln nodules from flue dust by the process of the American Sintering Co.; flue dust and pyrites cinder by Dwight & Lloyd process; sinter of Cornwall ore by Dwight & Lloyd process; Norwegian magnetic concentrate by Dwight & Lloyd process; magnetic concentrate by Dwight & Lloyd process; granular hematite ore by Dwight & Lloyd process; flue dust and magnetic concentrate by Dwight & Lloyd process; flue dust by Dwight & Lloyd process.

From the observations made on the tests the author arrives at the following conclusions:

1. The permeability of the cell wall of a sintered product varies inversely as the degree of fusion to which it has been subjected.

2. In a product of complete fusion, the silica present combines with its equivalent of iron oxide to form a perfect glass, which, from its greater fluidity, envelops and seals the remaining iron oxide from the action of gases.

3. Conversely of the foregoing conclusion, in the product of the lowest degree of fusion, the iron oxide and slag forming materials as a unit are bonded together by incipient fusion, leaving the predominant iron oxide free and vulnerable to the action of the gases in the highest degree attainable in solid products.

The above salient facts show the Dwight & Lloyd products, when properly made, to possess those properties which distinguish them from the products of other sintering processes or agglomerating methods by freedom from those constituents to which scouring action in the blast furnace is attributed.

Gold and Silver.

Weight of Tube Mill Pebble Loads.—The following tabulation has been prepared by MR. W. A. CALDECOTT, and published in the Feb. 1913 issue of the *Journal of the Chem. Met. & Min. Soc. of S. Africa*:

		Internal Diameter of Tube Mill Lining									
Pebble Load in 22 ft. Tube Mill		54 in.	55 in.	56 in.	57 in.	58 in.	59 in.	60 in.	61 in.	62 in.	63 in.
2 in. above axis of mill	14.21	14.65	15.10	15.55	16.02	16.48	16.96	17.44	17.92	18.42	
1 in. above axis of mill	13.81	14.25	14.69	15.14	15.59	16.05	16.51	16.98	17.46	17.94	
10 in. above axis of mill	13.41	13.84	14.27	14.71	15.15	15.60	16.06	16.52	16.99	17.47	
9 in. above axis of mill	13.01	13.43	13.85	14.28	14.71	15.16	15.60	16.06	16.52	16.98	
8 in. above axis of mill	12.60	13.01	13.42	13.84	14.27	14.70	15.14	15.59	16.04	16.49	
7 in. above axis of mill	12.18	12.58	12.99	13.40	13.82	14.24	14.67	15.11	15.56	16.01	
6 in. above axis of mill	11.76	12.15	12.55	12.95	13.36	13.78	14.20	14.63	15.07	15.51	
5 in. above axis of mill	11.33	11.72	12.11	12.50	12.90	13.31	13.73	14.15	14.58	15.01	
4 in. above axis of mill	10.90	11.28	11.66	12.05	12.44	12.84	13.25	13.66	14.08	14.51	
3 in. above axis of mill	10.47	10.79	11.21	11.59	11.98	12.37	12.77	13.17	13.59	14.00	
2 in. above axis of mill	10.04	10.40	10.76	11.14	11.51	11.90	12.29	12.68	13.09	13.50	
1 in. above axis of mill	9.60	9.96	10.31	10.68	11.05	11.42	11.80	12.19	12.59	12.99	
Level with axis of mill (i.e., half full)	9.18	9.53	9.88	10.23	10.60	10.96	11.34	11.72	12.11	12.50	
1 in. below axis of mill	8.77	9.10	9.45	9.79	10.14	10.51	10.88	11.25	11.63	12.01	
2 in. below axis of mill	8.33	8.66	9.00	9.33	9.69	10.03	10.39	10.76	11.13	11.50	
3 in. below axis of mill	7.90	8.27	8.55	8.88	9.22	9.56	9.91	10.27	10.63	11.00	
4 in. below axis of mill	7.47	7.78	8.10	8.42	8.76	9.09	9.43	9.78	10.14	10.49	
5 in. below axis of mill	7.04	7.34	7.65	7.97	8.30	8.62	8.95	9.29	9.64	9.99	
6 in. below axis of mill	6.61	6.91	7.21	7.52	7.84	8.15	8.48	8.81	9.15	9.49	
7 in. below axis of mill	6.19	6.48	6.77	7.07	7.38	7.69	8.01	8.33	8.66	8.99	
8 in. below axis of mill	5.77	6.05	6.34	6.63	6.93	7.23	7.54	7.85	8.18	8.50	
9 in. below axis of mill	5.36	5.63	5.91	6.19	6.49	6.79	7.08	7.38	7.70	8.02	
10 in. below axis of mill	4.96	5.22	5.49	5.76	6.05	6.33	6.62	6.92	7.23	7.53	
11 in. below axis of mill	4.56	4.81	5.07	5.33	5.61	5.88	6.17	6.46	6.76	7.06	
12 in. below axis of mill	4.16	4.41	4.66	4.92	5.18	5.45	5.72	6.00	6.30	6.58	

The above table is calculated as follows:

Assume R = Internal radius of lined tube mill in inches.

V = Height in inches from top of pebble load to lining.

L = Length of tube mill in feet.

F = Weight in lb. of one cubic foot of pebbles.

Then the general formula for the pebble load in tons is:

$$0.0001 LF \{ 0.1091 R^2 - 0.0463 V \sqrt{V(2R - 0.608 V)} \}$$

The pebble load in tons of a tube mill 22 ft. long, and with pebbles at 105 lb. per cubic foot, is:

$$0.0252 R^2 - 0.0107 V \sqrt{V(2R - 0.608)}.$$

The above formulae are derived from Molesworth's formula for the area of a segment, namely:

$$\frac{4V}{3} \sqrt{(0.626 V)^2 + C^2},$$

when V is the height of the segment, and C is the semichord.

When the upper surface of the pebbles is below the axis of the tube mill, find the weight of the pebble load whose upper surface would be the same distance above the axis, and deduct this weight from the weight of the pebble load required to entirely fill the tube mill.

Recovery of Black Sand and Floating Minerals.—An article relating to recovery of black sand on amalgamating plates in Rand mills is published by MR. J. M. NEILL, in the March, 1913, issue of the *Journal of the Chem. Met. & Min. Society of South Africa*. The general practice on the Rand is to collect and treat separately all the black sand that happens to adhere to the amalgamated copper plates. This sand consists mainly of iron sulphide containing large quantities of gold, with small quantities of iridium, osmium and in some

cases platinum. It generally comes down on the plates near the discharge from the mortar boxes, and is seldom found below the top 2 ft. of the plate. In the opinion of the author much more black sand could be collected if provision were made to prevent its escape from the plates. The failure of all the sand to become attached to the plate is due probably to flotation of small particles, by reason of attachment of small bubbles of air, or of films of air which prevent the mineral from overcoming the surface tension of the water and sinking to the plate. In order to accomplish this purpose the author devised a light, flexible, floating apron of canvas and cork, which extends across the plate at a point about half way down and under which the pulp may flow freely. Air bubbles attached to mineral particles are disturbed and liberated and when the surface tension of the water is overcome the particle sinks and is attached to the plate.

The value of the device in effecting an additional recovery of black sand which ordinarily would find its way into the cyanide pulp together with floured mercury is shown by the records of four months' operation with and without the de-

vice on the plates. During each period of four months 70,000 tons of ore were treated and all conditions of temperature, alkalinity, etc., were the same. At no time during either period was the mill allowed to run with acid water. The black sand was collected by the usual methods at the regular times of dressing the plates.

The first period, when the device was not used, yielded 1 ton of black sand for each 6188 tons of ore crushed. The value of the black sand actually recovered was 647 oz. fine gold per ton.

The second period, during which the device was used, yielded 1 ton of black sand for each 5000 tons of ore treated and the value actually recovered was 800 oz. fine gold per ton. Subsequently, when the methods of operation were better perfected, values of 1200 oz. to 1500 oz. per ton were recovered.

During the first period the proportion of black sand amalgam in total amalgam was 33.00 per cent; during the second period it rose to 43.41 per cent. The actual increase in mill recovery was 5.88 per cent of the value of the screen sample.

The separate recovery and treatment of this product by amalgamation offers great advantages over allowing it to pass to the cyanide department, as the percentage recovery is higher.

Consolidated Langlaagte Mill, South Africa.—In the *South African Mining Journal* for May 17, 1913, the Consolidated Langlaagte mill is spoken of as "the last word in Rand milling and cyaniding practice." The following data are given for the first three months of this year:

The aperture of the battery screen varied from 0.375 to 0.5 in., and the falling weight of the stamps averaged 1758 lb. The stamp duties were as follows:

Tons per stamp
per 24 hours.

January	18.230
February	19.545
March	20.135

Grading analyses of the final pulp sent to the cyanide plant were:

	+60 per cent	+90 per cent	+200 per cent	-200 per cent
January	0.69	14.33	23.58	61.40
February	0.27	10.04	22.83	66.86
March	0.12	10.33	20.03	69.52

By means of the double cone system of classification, 39.5 per cent of sand and 60.5 per cent of slime were produced. The actual percentages of gold recovered were:

	January per cent	February per cent	March per cent
By amalgamation	71.982	70.474	72.880
From sand	12.118	12.667	12.221
From slime	12.423	13.607	11.502
Total	96.523	96.748	96.603
Value of residue per ton milled, dwt.	0.221	0.219	0.231

A general outline of the process is as follows: The ore is screened over grizzlies, washed in trommels and sorted on belts. The undersize of the trommels passes to the sand pumps and the coarse ore is broken in jaw crushers preliminary to stamping. One hundred stamps of 1900 lb. weight fall through a height of $8\frac{1}{2}$ in, ninety-eight times per minute. The battery pulp is classified in a double set of cones, the underflow of the large cones passing to the tube mills, of which there are ten, 16 ft. 6 in. long by 6 ft. diameter. The overflow of the primary cone is classified into coarse and fine sand products and flows to the pumps for those products. The tube mill discharge is amalgamated on forty stationary plates, each 7 ft. by 5 ft., four to each tube, and finally elevated to the classifying cones for separation into fine and coarse products for cyaniding. The sand is treated by leaching and the slime by intermittent decantation. Precipitation is by zinc shavings.

Zinc

Formation of Zinc Ferrite in Roasting Blende.—The tendency of oxides of aluminium, chromium, calcium and zinc to combine with Fe_2O_3 at high temperatures has been proved due to the function of Fe_2O_3 acting as an acid, forming in the case of zinc, ZnOFe_2O_3 , zinc ferrite. As the formation of this compound has an effect on processes for extracting zinc from its minerals, a study of the formation of ferrites was undertaken by Mr. G. S. Brooks, of Depue, Ill., who publishes a paper on the subject in the May, 1913, *Bulletin* of the American Institute of Mining Engineers.

In examining with a microscope the many products of roasting kilns it is impossible to find individual grains of zinc ferrite as formed synthetically. A partial ferritization is noticed, however, in which one end or side of a grain seems to be transformed into ferrite. This, however, is not true of Western marmatites, the product of which seems always uniform.

A method of chemical analysis of roasted products to determine the percentage of ferritized zinc is to use a solution which will dissolve unchanged zinc oxide, and leave any zinc that has been altered to ferrite. A suitable solution, and one that gives results that are consistent, contains 200 gr. NH_4Cl , 500 cc. NH_4OH and 750 cc. water. By digesting 0.5 gr. of the sample in 50 cc. of this solution for an hour, the unchanged zinc oxide will be dissolved, and can be estimated in the usual manner.

In considering the formation of zinc ferrites on the hearths of roasting kilns, two distinct ores must be considered: one a mechanical mixture of iron and zinc minerals, and the other a chemical combination of the two. In the former case contact of the grains plays a more important part than in the latter.

In the case of mechanical mixtures of blende and marcasite, the author is of the opinion that the temperature of the roasting furnace is insufficient to cause incipient fusion of the two minerals to any great extent. An ore very high in marcasite will be more sticky, and it is likely that the zinc-iron matte thus formed will yield ferrites. In such ores the fine particles may, at the time of rabbling, flash to a temperature sufficiently high to cause fusion.

As to the progress of formation of ferrites during the roast: It is generally agreed that zinc sulphides in roasting form intermediate products such as normal and basic sulphates. It appears probable that the tendency of the oxides of iron and zinc to form ferrites begins with the zinc as sulphate, or with both metals as oxides, or with the desulphurization of any zinc-iron matter present. The author has found that the two roasted oxides of iron and zinc, when mixed in a very fine condition, do form ferrites to an appreciable extent, notwithstanding the observations of some earlier investigators to the contrary. Zinc sulphate and iron oxide, when roasted together in a laboratory assay muffle at a low heat, were found to yield a high percentage of ferrite.

In regard to the composition of ferrites, the author is of the opinion that the kiln products contain zinc ferrite of greatly varying composition, ranging from a normal ferrite at one end of the series to an extremely basic one at the other, so that its formula at all times cannot be accurately written. Furthermore, no definite relation between the iron-zinc ratio of the green ore to this composition has been discovered.

In discussing the surface factor in the formation of ferrites, the author calls attention to the fact that with the perfect molecular contact of the isomorphous iron in the Western or British Columbia blendes, the size of grain has little bearing on the tendency to form ferrites; but that in ordinary ores, and for equal weights, the portion which will pass 60-mesh screen offers such a surface as will tend to form ferrites more easily than will the coarser pieces. In addition to the area of contact, the frequency with which the ore is rabbled will have a decided bearing on the formation of ferrites.

Rise in temperature increases the tendency for the combination of iron and zinc materials, and it has been observed that the higher temperatures give ferrites toward the more basic end of the series.

The foregoing observations seem to indicate:

1. The possibility of some control over ferritization by reducing the contact surfaces.
2. That highly oxidizing atmospheres in the kiln hearths may permit quicker roasting and lower temperatures, both of which lessen the tendency to combine. This, however, must always allow for the exigencies of acid manufacture with its strong SO_2 gas requirements.
3. That frequent rabbling may aid, when not excessive, to the point of grinding ore grains into fines.
4. That the combination as a ferrite on the hearths may occur with the two minerals as oxides, or as a combined matte, or from the reaction of ZnSO_4 and Fe_2O_3 .
5. That true marmatites yield a high proportion of ferrite under all conditions of commercial roasting.
6. That the formation temperature in practice is somewhat below 1000° C.

The average tenor of zinc and lead sulphide concentrates in different states is shown by the following table, compiled from statistics of the United States Geological Survey for 1912. A similar table for 1911 appeared in this journal, July, 1912, page 423.

	Per Cent. Metal Crude Ore Lead Conc.	Per Cent. Lead in Lead Conc.	Per Cent. Zinc in Zinc Conc.	Value Lead Conc.	Per Ton Zinc Conc.
Oklahoma—					
Miami District.....	1.7	2.6	79.6	51.8	\$54.52 \$39.80
Quapaw District.....	...	1.1	79.5	57.8	50.00 48.36
Missouri—					
Soft ground.....	0.27	2.14	79.5	58.3	54.43 49.99
Sheet ground.....	0.36	1.26	79.5	58.6	55.83 50.84
Illinois—					
.....	0.4	3.2	79.5	35.4	55.38 27.30
Arkansas—					
.....	0.1	2.7	79.5	49.0	55.90 39.63
Kansas—					
.....	6.3	1.6	78.3	55.4	52.88 47.30
Wisconsin—					
.....	0.2	2.9	76.5	38.4	53.57 28.34

The Adaptability of Electric Welding

By George Hills

Among modern money and time-saving processes, welding by electricity and gases are in the front rank and among the several welding processes now in vogue; the electric arc method is now of particular interest, due to its comparative novelty and its wonderful flexibility.

It is but some nine years ago since I installed, to the best of my knowledge, the first electric arc welding system in America.

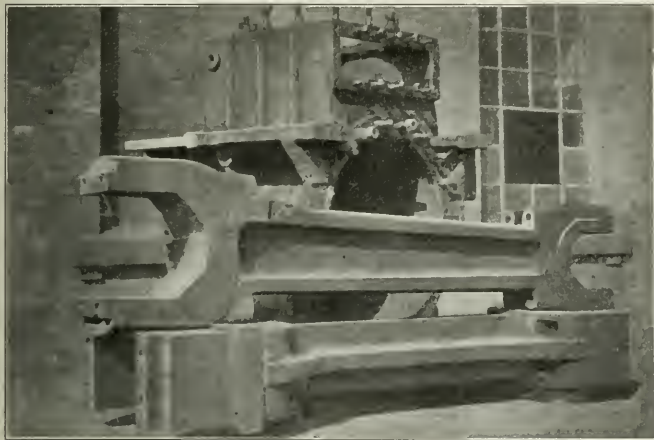


FIG. 1. CONNECTING RODS OF LARGE STATIONARY ENGINE AND CYLINDER WELDED ELECTRICALLY.

This was at the works of the Midvale Steel Company. Scullin-Gallaghers, Commonwealth and other steel companies soon adopted the system for the purpose of patching up and redeeming defective steel castings and forgings, and with what success may be gathered by the fact that each of them trebled their capacity in this direction within a year. Manufacturers of many other lines were quick to see the possibilities of electric welding, and the writer of this article was besieged with questions from all corners of the globe regarding the adaptability of the system.

I have personally engineered the sale of arc welding apparatus during the past thirty months approximating a quarter million dollars and distributed the system amongst steam railroads, electric railways, ship builders, steel foundries, steel mills, tank builders, sheet metal workers, car builders, pipe benders and fabricators, etc. Yet the method is comparatively new, and surprisingly little known. Those who do know the value of electric welding are indeed reaping a rich harvest. Consumers are specifying the welded article in preference to riveted work.

Prices are usually based on gas welding which is from 50 per cent to 75 per cent more expensive than the electric process.

Electric arc welding has been used extensively for the past twenty-five years in England and in Continental Europe for the purpose of uniting iron and steel forming pipe shapes, expansion joints, tanks, reservoirs, galvanizing kettles, etc., and no inconsiderable amount of the product has been imported into this country.

The earliest days of electric arc welding appear to have been about 1881, when this process was employed by one DeMentens for lead burning; a patent was granted to one Bernados in 1887 for iron and steel welding by this method,

which was substantially the same as the present day method although not so economical. The method consists primarily in drawing an arc between the metal to be welded and an electrode, which may consist of a carbon for some classes of work or a metal stick for work of another character, such as flue and locomotive fire-box welding or other work on a vertical wall, or overhead.

In pipe welding either method may be employed with equal success, except where cutting is necessary, for which purpose the carbon electrode is used.

In welding a nozzle to pipe, the hole may be cut to suit, using approximately 400 amperes, the nozzle then fitted and welded with the metallic electrode using approximately 150 amperes. Reinforcement to any desired extent may be added and oftentimes such a fillet is built up to two or three times the thickness of the wall of the pipe.

The metal may be worked and hammered while in a plastic state, thus insuring close grain, equal to drop forgings.

An electric street railway company which has been using electric arc welding on its breakages gives the following interesting data:

Gear case lugs 7 to 10 minutes; electric power 4 to 6 kw.

Armature shafts (broken) 2 inch, 40 to 60 minutes; power 20-30 kw.

Dowel pin holes 5 to 12 minutes; power 4 to 8 kw.

Broken motor cases 2½ to 3½ hours; 75 to 90 kw.

Broken lugs on a compressor cover doors and grease hinges; 2 to 5 minutes. 1 to 3 kw.

Broken truck frames 30 to 60 minutes; the power required varies between 20 and 35 kilowatt.

Worn bolt holes in motors and trucks 5 to 10 minutes; power 3 to 5 kw.

Enlarged and elongated holes in brake levers 2 to 4 minutes; the power required varies from 1½ to 3 kilowatt.

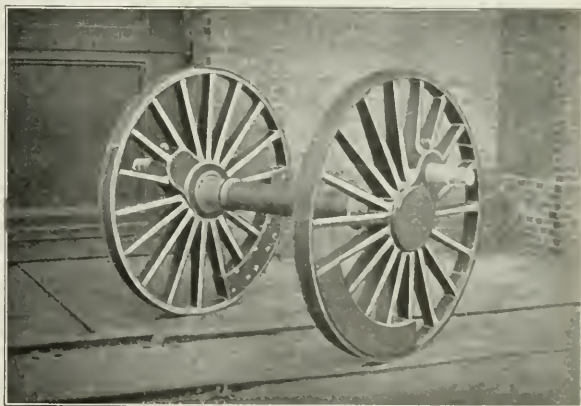


FIG. 2—BROKEN DRIVING WHEEL (SHOWING CRACKS)

Armature shafts, worn in journals 2 inch, 2 to 3 hours; 60 to 90 kw.

Armature shafts, worn in key-ways 10 to 15 minutes; 7 to 12 kw.

Armature shafts, worn thread 20 to 30 minutes; 10 to 15 kw.

Air brake armature shafts (broken) 20 to 30 minutes; 10 to 20 kw.

Leaking axle boxes 5 to 15 minutes; 3 to 7 kw.

The following data is compiled from actual practice and may be taken as average:

Carbon Electrode Cutting and Welding					
Size.	Time burning out and preparing.	Time to weld.	Amp.	Volts.	Kw.-hrs.
2" x 2"	3 minutes	10 min.	400	65	5.4
3" x 3"	7 minutes	20 min.	400	65	12
4" x 4"	10 minutes each side	25 min.	400	65	34
	1 hr. 15 min. to complete welding and cutting.				
5" x 5"	15 minutes each side	40 min.	450	65	58½
	2 hrs. complete welding and cutting.				
6" x 6"	18 minutes each side	40 min.	500	65	81
	2½ hrs. complete welding and cutting.				

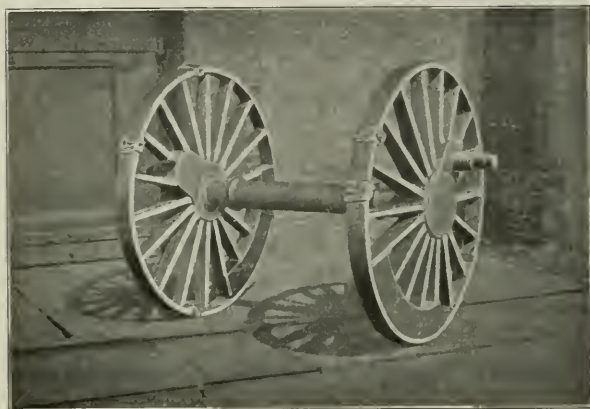


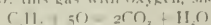
FIG. 3. WELDED DRIVING WHEEL.

Metallic Electrode Welding

Size, Inches.	Time (minutes).	Amp.	Volts.	Electrodes.	Kw.-hrs.
1/8	6 1 min. 40 sec.	25	65	1/16	0.04
1/16	6 1 min. 25 sec.	40	65	3/32	0.06
1/8	4 1 min.	70	65	3/32	0.1
1/4	4 3 min.	100	65	1/8	0.33
3/8	6 12 min. reinforced	135	65	5/32	1.75
3/8	6 8 min. no reinforcement	100	65	5/32	0.80
1/2	6 12 min. reinforced	135	65	5/32	1.75
1/2	6 11 min. no reinforcement	135	65	5/32	1.70
3/4	6 31 min. reinforced	140	65	5/32	4.60
3/4	6 25 min. not reinforced	140	65	5/32	3.65
1"	6 44 min. reinforced	140	65	5/32	6.70
1"	6 40 min. not reinforced	140	65	5/32	6.10

The electric arc method of welding is obviously superior to any method which employs a mixture of carboniferous gas and oxygen, such as oxy-acetylene. In the case of the electric arc the heat is generated by resistance and there is no carbon or oxygen present to combine with the metal at high temperature, whereas, with systems of welding by use of gases, the flame is caused by the liberation of heat for acetylene.

Acetylene is an endothermic compound; that is, its formation is attended by the absorption of or storing up of heat, in contra-distinction to those exothermic bodies which evolve heat in their formation, and the intense heat of the flame is caused by the combining of this gas with oxygen, shown by formula:



The adjustment of the proportions of acetylene and oxygen must be very fine, or either the oxygen or acetylene will be in excess. In the former case the superheated steel would be oxidized while in the latter case it would be carbonized. In

either case the strength of the metal would be seriously impaired.

The British Association of Civil Engineers investigated this subject recently, and a paper was read before them by Messrs. T. E. Stanton and J. R. Pannell, and I take the liberty of quoting below from the synopsis of the same, as it bears out the results of my own investigations.

"The strength of welded joints has been investigated by the National Physical Laboratory of Great Britain, at the request of Sir John Wolfe Barry, and the results show that with good workmanship the joint has somewhat better than 80 per cent efficiency, with 1¼-inch bars, the only size tested.

"The experiments were made under rather unusual conditions, for the 167 welded joints received for test were furnished by sixteen firms invited to submit them as samples of the methods of working which they preferred. The bars were subjected to tensile tests and to alternating stresses by the

Wohler method. In the latter the machine was run at 2200 r.p.m., after experiments at that speed and at 200 r.p.m. had shown that there was no difference in the fatigue strength of the same material.

"The experiments were described in detail in a paper which Messrs. T. E. Stanton and J. R. Pannell presented on December 12 before the Institution of Civil Engineers. They showed that the mean tensile strength of hand-welded iron bars was 89.3 per cent of the strength of the original bar, and that the percentage in the case of the hand-welded steel was 81.6, with electrically welded iron 89.2, and with electrically welded steel 93.4. Joints made by the oxy-acetylene process were submitted by two makers, but the results obtained with them were not comparable with those obtained by hand or electrically.

Fig. 1 shows connecting rods of large stationary engine and cylinder-welded. Plate 2 shows driving wheel broken. Plate 3 shows driving wheel welded. These are typical repairs.

Butte-Anaconda Electric Railway

While electric traction problems and the competition between the single-phase and the direct-current systems, important as they are, are somewhat out of the sphere of interest to the readers of this journal, the electrification of the Butte, Anaconda and Pacific Railway is of such exceptional interest as to deserve special mention at this place.

This new electric railway is of interest not only from a metallurgical standpoint, on account of its connecting the Washoe smelter with the Butte mines in a new and very efficient way, but also from a purely electric engineering standpoint, as it is the first installation in this country utilizing the 2400-volt direct-current system. The construction work necessary to effect the change from steam to electric equipment is now practically completed.

The adoption of the 2400-volt direct-current system for this railway was determined after a comprehensive study of local conditions and requirements. The traffic demands are unusually severe and consist principally of hauling long trains conveying copper ore over heavy mountain grades. In comparison with other existing systems, the 2400-volt direct-current system was considered best suited for service of this character.

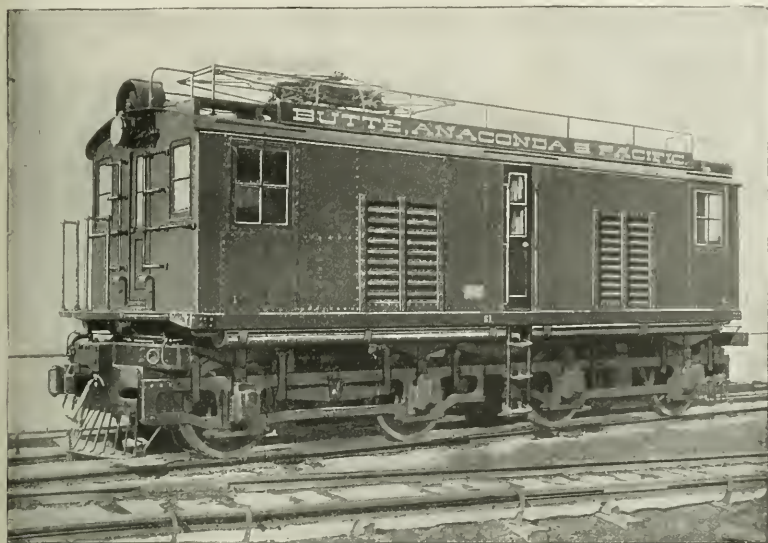
The Butte-Anaconda section, which has now been equipped, comprises 30 miles of main-line single track and numerous sidings, yards and smelter tracks, aggregating a total of about 90 miles on a single-track basis. The haulage of copper ore from the Butte mines to the smelters at Anaconda, together with all mine supplies, lumber, etc., moving in both directions, amounts to practically 5,000,000 tons of freight per year. Complete freight trains weighing 3,400 tons are made up of fifty loaded steel ore cars and will be handled against a ruling grade of 0.3 per cent by a locomotive consisting of two of the units illustrated. Single units will be used for making up trains in the yards and for "spotting" cars.

The initial equipment consists of seventeen locomotive units, fifteen for freight and two for passenger service. Each unit weighs approximately 80 tons. The two units for forming the freight locomotives in each case will be coupled together and operated in multiple unit. The combination freight loco-

for practical furnace work, that, at high temperatures, the transmission of heat is equivalent to the fourth power of the absolute difference in temperature. This indicates the necessity for a flame of the highest possible temperature in order to work economically, and this flame is best obtained by gas firing with preheating of gas and air.

In the correct assumption that the application of the regenerative system to the heating furnace would mean economic advantage, a number of practical experiments were made, none of which, however, led to the desired result or secured the wished-for advantage over the recuperative furnace, and the claimed advantage over the recuperative furnace, which naturally was the strongest competitor of the regenerative furnace, was not proved.

The types tried were constructed on the principle of reversing the flow of the combustion gases issuing at the charging end of the furnace, and diverting them alternately into heat accumulators. The reasons for the non-success of this type are manifest. Since the flame or used gas



80-TON 2400-VOLT D. C. LOCOMOTIVE FOR BUTTE-ANACONDA RAILWAY

tives will haul the usual trains of 3400 tons at a maximum speed of 15 miles per hour against the ruling grade and at 21 miles per hour on level tangent track.

The passenger locomotives are of the same design as the freight locomotives, but they are geared for a maximum speed of 45 miles per hour on level tangent track. A schedule of eight passenger trains per day, four each way, is maintained, the average train being composed of a locomotive and three standard steam-road passenger coaches.

All the locomotive equipment, as well as the substation apparatus and overhead line material, was designed and built by the General Electric Company.

New Regenerative Heating Furnace

A new regenerative heating furnace, invented by Mr. Friedrich Siemens, of Berlin, Germany, has lately come into extended use in several of the largest rolling mills of Germany and Austria. While it has been described in *Stahl und Eisen*, 1912, No. 37, and in the *Iron and Coal Trade Review* (London) 1912, Nov. 8, it is little known in this country. As the furnace is now to be introduced, however, into this country by Dr. Karl Georg Frank, 60 West Street, New York City, as Mr. Siemens' representative, the following description based on the above mentioned publications should be of interest.

The main point to be kept in view when designing a heating furnace is not so much to develop inside the furnace all the heat contained in the fuel, but rather to transmit to the material to be heated the maximum possible amount of heat, even though part of the total heat is intentionally allowed to go to waste.

The transmission of heat depends, as is well known, on three factors—conduction, convection and radiation. The first two of these increase in approximately direct proportion to the difference in temperature, whereas—according to the Stephen-Boltzmann formula—radiation increases as the fourth power of the absolute temperature. Hence, radiation will finally predominate to such an extent that one can assume, with sufficient accuracy

was obliged in part to flow over cold ingots, the temperature was lowered too far to enable the accumulators to be heated strongly; and furthermore, the highest temperature in the accumulators was at the point opposite to the air exit, so that, in order to maintain the counterflow principle, it was necessary to construct long intermediate passages.

Finally, a valve capable of diverting the hot gases was required; and it has been found impossible up to the present to overcome, anything like completely, the practical difficulties presented by such valves, exposed as they are to great heat. This type of furnace is, in principle, nothing more than the double-reversing system, of which so much was heard in the early days of the regenerative system, at the time when the

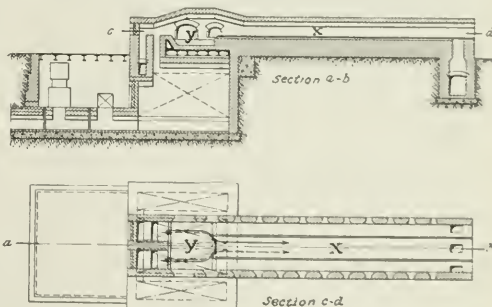


FIG. 1.—REGENERATIVE GAS FURNACE

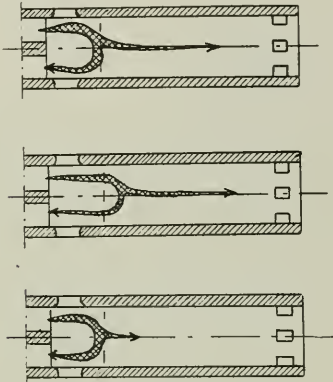
change of direction of the flame was still regarded as a drawback.

An entirely novel solution of the problem is afforded by the heating furnace illustrated in Fig. 1. In this type the flame is divided in the open chamber of the furnace, the one portion flowing right through the heating chamber, while the other turns back, horseshoe fashion, and escapes through ports situ-

ated close to the admission ports. After reversal, the disposition of the flame is just the same as described.

At first it was frequently objected that the flame would not take the prescribed course; and this objection is tenable so long as a regenerative flame is not in question, because a flame whose molecules are not highly heated and therefore does not contain any store of energy in itself, must necessarily terminate in a chaos of soot directly the mechanical intermingling ceases at the point where the flame is to divide, and where, therefore, for a short time there are no differences of pressure.

The highly-heated regenerative gas flame, on the contrary,



FIGS. 2, 3, 4.—ADJUSTMENT OF FLAME

overcomes this dead point with ease; the flame does not lose its heat for a moment at the dividing point, and the two divisions burn clearly and perfectly. The reversible part of the flame therefore comes in contact solely with the hot charge, and is thus well adapted to strongly heat the regenerative chambers; or, in other words, these chambers can be raised to the maximum temperature with a comparatively small amount of flame.

Like all regenerative furnaces, these work with natural in-draft of the air and with an outward pressure in the heating chambers, so that no air can enter when the discharging door is opened. Fig. 1 shows a furnace of this kind in longitudinal and cross section, *x* being the delivery or welding portion and *y* the preheating of the "push" portion of the bed.

This furnace is intended to be operated with cold producer gas or blast-furnace gas, and is consequently provided with four adjacently situated regenerative chambers. The arrangement of the air and gas valves is the same as in the ordinary Siemens furnace, except that the flame ports are not situated at both ends of the furnace, but adjacently at the delivery end.

The flame, therefore, travels for a time in the direction of the plain-line arrow, and after reversal, in the direction of the dotted arrow. This is the usual arrangement; but there is nothing to prevent the horseshoe portion of the flame from being assigned a vertical position, and similarly the flame may be led over the welding hearth in a direction parallel to, but alternately in opposite direction to that portion of the flame which flows through the "pushing" hearth *y*.

It is also possible to arrange for a transverse flame, the one portion of which shall be diverted to the "pushing" hearth. There is a further modification with a large central burner and two smaller ones on the right and left of same, both these lateral burners leading to the common heat accumulator. This pattern works with great success.

For metallurgical "push" furnaces, however, the arrangement of the flame as in Fig. 1, with cold or blast-furnace gas, will undoubtedly remain the most usual type.

Accurate measurements and heat balances for these furnaces are now being compiled; but it has already been ascertained

that, given proper adjustment of the valves and dampers, the temperature of the escaping gases both at the charging end and the heat-accumulator end of the furnace is about 300 deg. C., and that the mean temperature of the air of combustion is very near 1100 deg. C.

The "push" furnaces previously in use do not in the least fulfil the divergent requirements which must be imposed upon them in respect of controllability. The only method of control hitherto available was the imperfect one of admitting an insufficient, correct, or excessive amount of air, in order to be able to adapt the furnace in any way to meet the various requirements existing when hard or soft, cold or warm, thick ingots or thin slabs are to be heated.

If large pieces of soft steel are in question, the heat on the welding hearth, *x*, may be raised to the highest welding temperature without objections, whereas it is not desired to bring the "push" hearth *y* too quickly to a temperature that will cause an appreciable loss of metal. In these circumstances, with the new regenerative "push" furnace, the flame would be adjusted as shown in Fig. 2, and the thickness of the indicating (arrow) line approximately corresponds to that of the flame and its radiation.

If hard steel is to be heated, then an accurately regulated temperature must be provided on the welding hearth, *x*, together with gradual uniform preheating. In such case the course of the flame should be approximately in accordance with Fig. 3.

If the material is charged warm, it is true that a high temperature will be required on the welding hearth, *x*, but only a gentle heating up on the "push" hearth *y*, since otherwise there would be an unnecessary waste of metal. In this case the regulating valves and dampers must be adjusted to give a flame as in Fig. 4.

In this way it is possible to insure a uniform output, in one and the same furnace, and under the most favorable conditions, no matter whether the charge be entered cold or warm. When thin billets or slabs, which heat up quickly, are treated, the possibility of obtaining a very high temperature on the welding hearth *x* will be utilized, and the furnace made as short as possible, in order to save room and reduce the loss of metal. In this and similar ways, the distribution of heat and the tem-

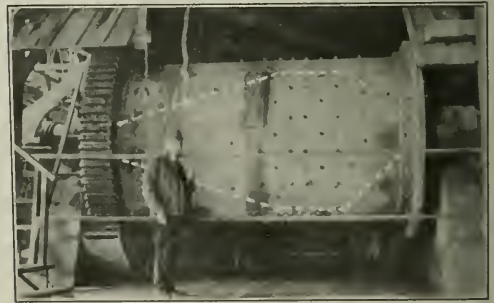


FIG. 1.—TUBE MILL CHANGED INTO CONICAL MILL

peratures can be accurately adjusted, with this furnace, to the requirements of the moment.

This regenerative gas-fired "push" furnace fulfils modern requirements in so far as the present practice is to heat up ingots, blooms, or billets to higher temperatures than was formerly the case, and because of the tendency in large steel works to make use of blast-furnace gas, either alone or in conjunction with producer gas or coke-oven gas, in the rolling-mill furnace.

In the steel works of Germany and Austria alone, 42 of the above-described "push" furnaces have been started or are in course of erection during the last two years, their aggregate capacity being three million tons per annum.

Conversion of a Tube Mill into a Conical Mill

The adjoining illustrations of the conversion of a cylindrical tube mill into a Hardinge conical mill at the plant of the Federal Mining and Smelting Company, at Wallace, Idaho, is interesting in more than one respect.

This company had six Huntingdon mills in operation, but decided to install some other kind of mill and for this purpose started a preliminary competition. This competition was be-



FIG. 2.—VIEW OF LINING

tween a 7 ft. x 12 ft. cylindrical tube mill and an 8-ft. Hardinge conical mill. The large power consumption of the cylindrical tube mill as well as the impossibility of starting it under full load with a 100-hp. motor resulted in the desire to convert it into a Hardinge conical mill by inserting a special conical lining. As such a conversion constituted a direct infringement of the Hardinge patents, the Hardinge Conical Mill Company furnished the user with a special license to continue the use of the mill as reconstructed.

Fig. 1 (p. 422) shows the outside of this tube mill and the dotted lines indicate the conical lining. This mill is now putting through a little less tonnage with approximately twice the horsepower than the standard Hardinge conical mills.

Fig. 2 is a photograph of the lining. It is of particular interest to note that when Mr. Hardinge invented and developed the conical mill, the first big mill he actually built was such a

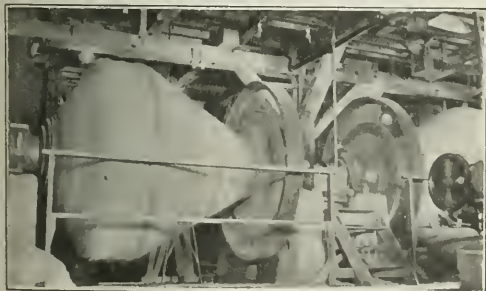


FIG. 3.—TWO CONICAL MILLS IN OPERATION

tube mill provided with a conical lining.

Fig. 3 shows two of the six standard 8-ft. Hardinge conical mills which have been installed by the Federal Mining and Smelting Company after the preliminary competition and which are now operating in conjunction with the converted tube mill.

Personal

Mr. **Gelasio Caetani**, metallurgical engineer of San Francisco, was in Denver recently on his return from a trip abroad.

Mr. **Pedro P. Calvo**, of Bogota, is at present in New York to study recent chemical and electrochemical developments

with a view of introducing some of them in the Republic of Columbia, which is rich in waterpowers and natural resources.

Mr. **Edwin M. Chance**, for four years chemist of the Philadelphia and Reading Coal and Iron Company, has resigned his position to become consulting chemist to a number of the anthracite and bituminous coal companies, and is now located at 61 South Pennsylvania Avenue, Wilkes-Barre, Pa.

Dr. **Regis Chauvenet** was recently elected president emeritus of the Colorado School of Mines, and Mr. **W. G. Haldane** was appointed acting president. Dr. Chauvenet was president of the school about ten years ago, and Mr. Haldane has been assistant professor of metallurgy. Other changes made in the faculty included the appointment of Mr. **Harry J. Wolf** as professor of mining.

Mr. **John A. Davis**, of the Denver branch of the Bureau of Mines, is engaged in an extensive study of western milling methods, and will be in the field for several months.

Prof. **Stephen L. Goodale**, of the department of ore dressing and metallurgy, University of Pittsburgh, has been visiting in Colorado.

Mr. **Franklin Guiterman**, of the American Smelting & Refining Co., New York, made an inspection of the company's Colorado smelters in June.

Mr. **H. W. Hardinge**, having just returned to New York from an extended trip through the Northwest and British Columbia, is to sail shortly for a two or three months trip to Russia and other countries.

Dr. **J. W. Holmes**, director of the Bureau of Mines, was a guest at a luncheon of the American Mining Congress in Denver in June.

Dr. **Charles L. Parsons**, of the United States Bureau of Mines, has completed an investigation of the uranium and vanadium deposits of southwestern Colorado and southeastern Utah. He was accompanied on the trip by Mr. **K. L. Kithil** of the Denver office of the Bureau.

Mr. **Edward S. Wiard**, metallurgical engineer of Denver, Colo., is engaged in testing ores for the Vindicator Consolidated Mining Co., Cripple Creek, and designing a plant for their concentration.

Prof. **Paul Walden**, of Riga, Russia, is the subject of a brief biographical sketch, written by Dr. **G. F. Kunz**, under the title "The President of the Ninth International Congress of Applied Chemistry," in the June issue of the *Popular Science Monthly*.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject-matter and in chronological order.

Compiled by **Byrnes, Townsend & Brickenstein**, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (Concluded).

Refining Metals

312,814, Feb. 24, 1885, **Henry R. Cassel**, of New York, N. Y.

Relates to apparatus and process for electrolytically refining metals. The apparatus comprises a decomposition tank, a silver precipitation tank and a device to circulate the electrolyte. The anode consists of the metal to be refined and is contained in a bag of horse-hair or asbestos; within the bag is a frame of metal not attacked by the electrolysis. The cathode is surrounded with a bag of a dense, porous, non-conducting substance which permits the passage of the electric current and of hydrogen, by dialysis, but does not permit the passage of dissolved metals from the anode. On passing the current copper and silver in the anode dissolve; any gold present collects as an insoluble residue in the anode bag. Any soluble particles falling from the anode collect on the metal frame and are there dissolved. The acid copper sulfate solution filters through the anode bag into the outer cell where it is precipitated by metallic copper. The solution then runs

off into the precipitating-tank where any remaining silver is precipitated by metallic copper, the solution then pumped into a supply tank from which it flows to the cathode bag or compartment where the copper is deposited and the remaining acid solution overflows to the decomposition tank.

315,265, April 7, 1885, Moses G. Farmer, of New York, N. Y.

Relates to apparatus for refining copper by electrolysis and consists in arranging the deposition vats in rows with spaces between the rows; tracks are placed in the spaces, and a car moved over the tracks to remove electrodes and to pump the electrolyte from a vat to an empty tank for cleaning, etc.

322,169, July 14, 1885, Moses G. Farmer, of New York, N. Y.

Relates to apparatus for refining copper, and consists of a vat in which the ingots are placed so as to divide the cell into a plurality of compartments by fitting tightly between the sides. The metal is dissolved from one side of one plate and deposited upon the adjacent side of the next plate. Against the cathode side of the plates or ingots is placed a grid or lattice-work of wood, etc., to divide the deposited copper into definite sized units, while the anode side of the plate is being dissolved away. When the operation is completed the cathode may be readily divided into sections along the division lines.

322,170, July 14, 1885, Moses G. Farmer, New York, N. Y.

Relates to a process of refining copper and contains a reference to his companion application which became patent 322,169. The impure copper in the form of plates is placed in inclined grooves in the sides of a vat, the bottoms of the plates resting on transverse bars affording space for the accumulation of the impurities in the metal. The plates so disposed form a plurality of compartments, and are so connected that the underside of a plate is an anode, while the upper side of the facing plate is the cathode. Air pipes supply air to the solution in between the plates, keeping the solution in circulation and assisting in removing the impurities. By this process one side of the plate is gradually dissolved off, while the other side is gradually built up with pure copper.

377,487, Feb. 7, 1888, Edward S. Hayden, of Waterbury, Conn.

Relates to a process for refining crude copper. The metal in plates of suitable size is fitted in grooves in the sides of a vat, which do not reach to the bottom, leaving a space below for the accumulation of impurities. The plates are so arranged that one side is the anode and the other side the cathode, so that a plate of impure metal will be dissolved on one side while pure metal is being deposited on its other side. The impure metal is preferably rolled before refining. The electrolyte is gently circulated through the cell, so as not to disturb the impurities which collect at the bottom.

Book Reviews

Electric Furnaces in the Iron and Steel Industry. By W. Rodenhauser and I. Schoenawa. Translated from the second German edition, with additions, by C. H. Vom Baur, E.F. 5 3/4 x 9 in. (15 x 23 cm.); 420 pages; 133 illustrations. Price \$3.50. New York: John Wiley & Sons, London: Chapman & Hall, Limited.

The first German edition of this work was reviewed in our issue of April, 1911. Mr. Vom Baur has made his translation from advance sheets of the second German edition, and has added here and there such items as were particularly needed by the English and American reader. The record of 60 arc furnaces and 40 induction furnaces in operation or under construction will come as a rude shock to those purblind American steel makers who seem to regard electric steel furnaces as an expensive fad, who do not see their possibilities, and who think eventually to save money by letting them alone. The old old story is repeating itself, and American industries are again trailing after the European—five years or so after them. The joke would not be so hard on us, if we did not usually pretend or profess to be leading the world. We may be leading the world in making profits, but in general this is not by virtue of

our enterprise and superlatively good practice, but simply by reason of our abundant natural resources and high tariff, and in spite of very evident moss-backed conservatism and frequent lack of initiative in adopting technical improvements.

To return to the book, Mr. Vom Baur has made an excellent translation and some useful additions. In particular, the chapter on electric furnace ore reduction to pig iron has been brought "up to the minute," with extensive assistance from Prof. D. A. Lyon. The typographic arrangement is not clearly defined as regards headings and sub-headings; some improvement in this regard should be made in the next edition. Everybody interested in the electric manufacture of pig iron and steel will need this book, for it is the best source of information on that subject now in print.

Handbuch der Mineralchemie. With contributions from fifty-nine special contributors and with the support of the Academy of Sciences of Vienna, edited by Hofrat Dr. C. Doelter, professor of the University of Vienna. To be complete in four volumes. First volume in 6 parts: vol. I, part 1. (p. 1 to 160), marks 6.50; vol. I, part 2 (p. 161 to 320), marks 6.50; vol. I, part 3 (p. 321 to 480), marks 6.50; vol. I, part 4 (p. 481 to 640), marks 6.50; vol. I, part 5 (p. 641 to 800), marks 6.50; vol. I, part 6 (p. 801 to 1008 and index and title page), marks 9.10. With numerous illustrations, tables and diagrams. Dresden and Leipzig: Theodor Steinkopff.

The first volume of this great new German work is now complete. The parts of the next volumes are following each other in regular rapid succession. The whole work which is to comprise four volumes is expected to be complete in 1914. If from the contents of the first volume, we may by extrapolation draw a conclusion as to the value of the whole work, there seems to be little doubt that this work will become classical in its field.

"In the beginning of its evolution, mineralogy was developed mainly by chemists. But when mineralogy grew to become an independent science and great successes were obtained by physico-mathematical methods, this school became preponderant and the intimate relation between mineralogy and chemistry got lost. Partially this relation was reestablished by the cultivation of mineral synthesis by chemists and by E. Mitscherlich's discovery of isomorphism. . . . Quite recently the application of the principles of physical chemistry to problems of mineralogy has again produced a more intimate connection of mineralogy with chemistry and has opened new and important vistas. . . . This handbook is intended to connect more closely chemistry and mineralogy and to summarize all that chemistry has found out concerning minerals so that the mineralogist as well as the chemist may familiarize himself with anything that has been done in the field common to both."

It may be said right here that the work is really of much broader interest and appeals to many readers outside of the purely mineralogical and chemical fields. Many metallurgical and chemical engineers should find it a source of instruction and inspiration. To mention a single example from the first volume, C. Doelter's extended chapters on silicates and J. H. L. Vogt's concise and illuminating chapter on slags should be of great interest and value to metallurgical engineers.

That fifty-nine specialists, besides the editor-in-chief, have united to write this work, makes its strength. The field is now too big to be covered authoritatively by a single man. Great credit is due to Dr. Doelter, however, for uniting the contributions of his fifty-nine collaborators into a harmonious whole. In this difficult task he has been remarkably successful.

The contents of the first volume are as follows: General introduction. Carbon: diamond, graphite. Carbonates (general, sodium carbonates, magnesium carbonate, calcium carbonate, and other carbonates), carbides, silicon, silicates, glass, glazes, enamels, slags.

The work is undoubtedly at present the most complete and exact summary of our knowledge of the subject and should be in every private or public library which claims in any way to be up-to-date.

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Denver Meeting of the American Electrochemical Society

In this issue we publish the preliminary program of the meeting of the American Electrochemical Society, to be held in Denver, Boulder, and Golden, from Sept. 9 to 11, with two additional days in Colorado Springs, on Pike's Peak, and in Cripple Creek. It promises to be a charming and thoroughly enjoyable affair and the old and ever-young Electrochemical Society spirit which has distinguished so many brilliant former meetings will have an opportunity to manifest itself under new and particularly attractive and congenial conditions. To the Easterner it is an ever new revelation to enjoy the sublime grandeur of Colorado's natural beauties and the big-hearted Western hospitality, while the special train which is being arranged, if possible, right from New York, should be an additional inducement to insure a large representative attendance from the East.

But, then, there are deeper dormant potentialities in this convention. It will be the first Western meeting of the American Electrochemical Society. It will be the first official professional getting-together of Western metallurgists—proven pioneers and empire builders—and of Eastern electrochemists—again proven pioneers with a vision of extending their influence westward. But it will not be a raid into the enemy's country nor a clashing of commercial interests. Whatever electrochemistry can do for Western metallurgy will be welcomed and accepted there with the true spirit of Western hospitality.

It may be expected that there will be a good program of instructive papers on the relation of electrochemistry to metallurgy. But however instructive and interesting the papers will be, they alone will not exhaust the possibilities of the Denver meeting. It is in the direct personal contact between man and man in open discussion in the professional sessions and in quiet heart-to-heart talk that the most good can be accomplished. The Denver meeting should give ample opportunities in this direction and the sphere of subjects of mutual interest for discussion is wide. There is electric smelting, electrolytic processes as an adjunct to leaching, electromagnetic and electrostatic separation, electroamalgamation and electrocyanidation, and an almost unlimited number of special possibilities.

Taking everything together, the Denver meeting promises to become not only a truly charming affair, but a new milestone in the career of the Society and in the forward march of electrochemistry.

Investigating the Smelter Trust

Representatives of the federal Department of Justice are in the West studying the relation between the mining and smelting industries and investigating alleged unfair treatment of the miner by the smelter. The investigation, we understand, is of a general nature, but more particularly

with reference to the operations of the American Smelting & Refining Company, commonly known in Colorado and the West as the "smelter trust." If an investigation is needed, the government is late by several years in seeking the reasons for the unfeigned hatred and unconcealed distrust of the miner for the smelter; but at that it is ahead of the several western states which have agitated the question for years without accomplishing anything.

The government will find that the miner believes he does not get a square deal from the smelter, and the reasons assigned will be many and various. Much of the information volunteered will be non-essential, incompetent and immaterial, coming from miners who have long harbored a prejudice against the smelter, but who, in the last analysis, will be found lacking in a knowledge of the essential requirements of smelting and in ability to drive a fair bargain with the smelter, or to prepare and grade their ore so as to get the greatest net return. In this respect it will be necessary for the investigators to exercise good judgment as to the reliability and competence of their sources of information, and to discount mere grumbling dissatisfaction and deep-seated, ignorant prejudice.

It is highly improbable that any serious fault will be found with the methods of weighing, sampling, and arriving at the gross value of the ore. The miner has every legitimate opportunity to watch the sampling of the ore, either personally or through an agent; and he has a similar privilege in assaying his sample to determine the metallic content of his shipment. Neither is it conceivable that any reflection can be cast on the integrity of the assayers selected as umpires, nor any suspicion entertained that the smelter has ever used unfair means or exerted pressure to influence the umpires' results. In fact, it is difficult to see where any fault could be found along these lines.

The crux of the whole matter lies in smelting contracts and the basis on which final settlement is made, and the government, if it has a legitimate business to do so, may well direct its main efforts to ascertain whether these contracts are fair, equitable, impartial and clear, or the reverse; and whether the prices, penalties, bonuses, charges and other rates are justifiable. It is a fact that smelter contracts are needlessly cluttered with vague and involved terms. Direct, plain statement is a lost art in these documents. The miner is easily befuddled in trying to comprehend them, and frequently does not find the jokers until too late to eliminate them.

In fact, we have the principal psychological elements of a horse trade if we reverse the position of buyer and seller in the analogy. The novice buying a horse from a typical, shrewd trader, and the plain miner selling ore to the smelter, occupy similar positions of uncertainty, distrust, and inability to cope with the superior knowledge of the other party to the transaction. The average smelting contract is a mystic maze in which many a miner finds it difficult to find his way. Nor does it contribute to his composure to find that a neighboring miner may have a different contract on similar ore with the same company, and which is equally unintelligible to him.

The extent to which the smelter may control the market for the metals, particularly lead, is another point that may occupy the attention. The ability to make arbitrary prices,

for reasons best known to the smelter, may possibly work a hardship on the miner, and certainly adds to his feeling of dissatisfaction and distrust.

The investigation probably will require several months, and its result will be awaited with interest. It is by no means a foregone conclusion that the government will be able to establish a case against the smelter, nor is the inquiry being conducted with prejudice or bias. In the end it may be, and probably will be, without great profit, but it should shed some light on a subject which has agitated the Western mining industry for several years.

Flue-Dust Explosions at Smelting Plants

The occurrence of two disasters by explosion and fire at Utah lead smelting plants has directed attention to a contingency which apparently did not exist prior to the use of the latest method of sintering fine ore. At the Tooele smelter of the International Smelting & Refining Company an explosion of flue dust caused serious damage to the roaster flue, and at the Midvale plant of the United States Smelting Company a spontaneous fire destroyed a large number of bags. Both disasters involved material loss and were quite extraordinary. In the case of the flue-dust explosion the usual provision of explosion-doors was insufficient to prevent wrecking the flue, while the destruction in the bag-house fire was of much greater magnitude than that caused by incipient fires which sometimes occur in such plants.

In each case the trouble is ascribed to the presence of finely divided elemental sulphur in the gases from the sinter-roasters. This material, settling in the flue-dust and bag accretions, apparently gives rise to dust explosions when the conditions of temperature and oxidation are favorable, similar to the progressive explosions of other finely divided combustibles, such as coal dust. That similar explosions have not occurred before probably is due to the fact that in earlier forms of roasters the sulphur was eliminated as some form of oxide instead of a sublimed element, the latter apparently resulting from the present method of roasting. Both the plants mentioned employ the Dwight & Lloyd method of sinter-roasting, a characteristic feature of which is that a comparatively thin layer of ore is quickly agglomerated into a porous mass, highly suitable for smelting. The mixture is exposed to heat for only a short time, and the quantity of air available for oxidation is limited to that which can be drawn through the charge during its passage over the wind-boxes.

Under such conditions it is likely that more sulphur would be sublimed, and less oxidized, than in the old forms of hearth roasting furnaces, or in other forms of agglomerating furnaces. Thus, in the Huntington-Iieberlein pot system of sinter-roasting, the charge is deeper, the air supply more copious, and the progressive sintering slower than in the Dwight & Lloyd method. There is a greater probability of sulphur being eliminated as oxide. It appears, however, that even with the Dwight & Lloyd system the tendency to distill sulphur into the flue gases varies with the physical condition of the roaster charge; and the likelihood of resulting explosions also is modified by these conditions. It has been observed that a very wet charge results

in the distillation of more sulphur than comes from a reasonably dry mixture. Some moisture is desirable in the mixture, in order to handle it properly, and indeed it is customary to add some water at times; but such excessive moisture as occurs in frozen ore apparently is detrimental.

The problem is being investigated, and probably will be well in hand before other disasters can occur. The suggestion has been made that the tendency to distill sulphur might be turned to advantage by creating conditions most favorable for this action, and recovering the sulphur thus formed. This would have a distinct bearing on the smelter fume question. At present, means are being adopted to cool the gases in the flues and further to regulate automatically the temperature of the gases entering the bag house by admitting a variable quantity of air with them.

Stimulus to By-Product Coking

What amounts to an artificial stimulus to the by-product coke industry is furnished by the Connellsville coke industry, in demanding prices for its beehive coke, which include large allowances for interest and exhaustion, based upon high valuations for Connellsville coal acreage. For several months the Connellsville operators have been engaged in an effort to secure \$2.50 for coke for delivery in the second half of this year, at a time when the price of pig iron would indicate a much lower price for coke, based upon the relations between pig iron and coke prices which have obtained in the past. The operators represent that the average price of coke has not furnished them a proper return upon the value of their properties. Whether the transactions were justified or not, Connellsville coal acreages have sold as high as \$1500 to \$2000 per acre, and on a 20-year life for the operation, at \$1500 an acre for the coal and an allowance for the investment in development, the interest, exhaustion and amortization charges amount to more than 50 cents per ton of coke produced, if full production is realized throughout the period. Average coke prices in the past few years have not carried such a margin beyond the bare factory cost, and the fact has been brought home with particular force to some operators, who bonded their properties on the basis of the cost of the coal acreage. Acreage values once assumed, the computation of cost of carrying the investment is simple, as the seam produces quite uniformly about 7000 tons of coke per acre.

The high values set upon Connellsville coal were based upon the theory that from no other coal could be produced a coke nearly as satisfactory for iron smelting as Connellsville. Within limitations, this assumption has been proved fairly correct, having regard only to the beehive process, but the introduction of the by-product process has entirely changed the situation.

The point is not that, *per se*, the by-product oven will produce as good coke as the beehive oven, with a considerably inferior coal. There are other important elements, growing out of the fact that the beehive oven is located at the pit mouth, while the by-product oven is located at the point of consumption. This difference gives rise to certain important factors. In the first place, the beehive coke must be shipped and the handling requires it to be of stronger and larger structure than the by-product. In the

second place, the beehive oven must be located upon a coal acreage certain to afford a satisfactory coal for a period of years; no chances can be taken. The by-product oven, on the other hand, can draw coal from anywhere, within the limits of freight cost. It can mix various coals and can change the mixture as often as desired. The owners can pick up small or large coal tracts in this place or that place as the market affords them favorable opportunity. Coal freights per ton average sufficiently less than coke freights to make the coal freight the less, per unit of coke, despite the larger tonnage moved. In the third place blast furnace designs can be modified so as to use with economy a coke of softer structure than the best of the Connellsville beehive coke. These considerations are not always taken into account, in addition to the well-known advantages of the by-product process, in its larger coke yield, and its yield of gas and other by-products.

As soon as high values for Connellsville coal acreage appeared, steel manufacturers recognized that the fact largely wiped out the disadvantage of the much greater capital investment involved in by-product oven construction as compared with beehive, for the beehive oven cannot be built unless an attendant supply of coal has been secured, while with the by-product plant this protection is less necessary, but in any event an equal coal acreage would cost much less money.

The placing of high values upon Connellsville coke thus stimulates the construction of by-product ovens. From what has occurred, taken in conjunction with current conditions and trends, the future can be forecasted with considerable accuracy. In 1900 the proportion of by-product to total coke made was 5 per cent; in 1905, 11 per cent; in 1912, 25 per cent, and in 1914, through the erection of plants now completed or in course of erection the proportion will not be far from 35 per cent. The growth in by-product coke production has been greater than the growth in total coke consumption, whereby it was as far back as in 1907 that the maximum production of beehive coke was reached, the 1912 production showing a decline of more than 6 per cent from this tonnage. In 1880 the Connellsville region produced 66 per cent of the total coke made in the United States. In 1890 the proportion had fallen to 56 per cent, chiefly through the rapid development in Alabama, but on account of the advantages of Lake Superior ore such outlying districts as Alabama and Tennessee did not keep up with the growth in pig iron production, and Connellsville coke maintained its proportion of about 55 per cent of the total through 1906. Then occurred a drop to about 45 per cent., which has been the average from 1907 to 1912, inclusive. This drop has been due chiefly to the growth of by-product coking. The production of Connellsville coke has been practically stationary, amounting in 1906 and in 1912 to 20,000,000 tons. At this rate, which certainly will not increase, the Connellsville region proper will be exhausted in about 20 years. The probability is strong that the remaining Connellsville coal will not all be made into coke in the region, but that shipments of Connellsville coal to by-product ovens, which have already commenced, will increase year by year. Few new beehive ovens will be built, as their life would be short, and old ovens will be abandoned, the remaining coal being shipped.

Readers' Views and Comments

Smelting Zinc in an Electric Furnace

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In a paper by Mr. Francis Louvri r, published in this magazine, November, 1912, on "Causes of the Practical Non-Success of Electric Furnaces in Treating Zinc Ores," he ascribes lack of success to the amount of blue powder produced in them, and attributes its formation to the oxidation of the reduced zinc by carbonic acid. He does not assume a greater production of carbonic acid than in ordinary practice, but believes that its more injurious effect is due to the rapidity with which it escapes from contact with hot carbon, which in part is due to the very rapid reduction of the zinc, and in part to a defect in the position of the electrodes.

It is not my purpose to review this article, or to discuss all of the questions raised. I believe the same object is often obtained in an argument by presenting a different point of view, as by proving that the other man was deceived by appearances. I shall suggest that the production of an excess of blue powder may not be due to an excess of oxidation; that even if it is, the carbonic acid formed by reduction may not be the oxidizing agent; that a more easily remedied defect than the position of the electrodes may be responsible for the rapid cooling of the zinc. I shall raise questions I do not expect to answer, but possibly others may supply, from their own experience, facts which will make explanation easier.

Some thirty-five years ago, I announced that a product closely resembling blue powder could be produced from distilled zinc, by methods of cooling, in no way oxidizing in their effect. I believe this is now generally accepted to be a fact. But the oxidation of some of the zinc in standard and proposed zinc furnaces is almost universally stated in current publications to be the reason why they produce blue powder. The intentional production of a large percentage of blue powder in the de Saulles furnace, for use in the arts, has some bearing on this question. I went even further at the time, and convinced myself by repeated experiment that the oxidation on a sample of ordinary blue powder produced in regular practice, which had been kept several months in my laboratory, was not sufficient to prevent the coalescence of its separate particles into a single liquid mass, under proper conditions of heat, without the application of pressure while heated, and without consolidation by compression before heating. The experiment was a very simple one, made in a Berzelius arsenic tube heated in a Bunsen flame. The powder melted readily, and had a brilliant lustre when in contact with the glass. It readily oxidized on top and was pitted with particles of oxide cinder and ore.

It may be admitted that any oxidation of the zinc has a retarding effect on its uniting as a single liquid mass, but I believe that it cannot be contended either that it is the sole or even the principal cause of the production of blue powder.

On the other hand, the fact should not be lost sight of that there are occurrences of zinc oxide in the products of every zinc furnace, other than the superficial coating on blue powder. A zinc smelter is not likely to confuse them even in thought. The skimmings of the ladles and the molds, and the scrapings from the condensers and the mouths of the retorts, contain zinc oxide which is regularly returned for a subsequent treatment. The passing in and out of the tool, in the drawing of the metal, introduces much air into the condensers; and when prolongs are not on the condensers, unless the workman is careful to keep the openings of the condensers nearly closed much air is drawn in when the temperature of the condenser falls, as it must under conditions of firing. Every smelter knows, and every user of blue powder must learn, that what goes by this name is far from uniform in composition. It

ranges, in the percentage of zinc which is present in the metallic state, from over 90 per cent. to less than 80 per cent., and it deteriorates rapidly by oxidation. It is to be regretted that more analyses have not been published, giving precise details, of the ores from which it was produced, of the manner in which it was collected, and of the conditions under which it was stored, and that this knowledge is not more accessible and widely diffused. Until this is done by some competent authority, it is necessary to insist on the great differences which are found between different samples of this product, and on the importance of distinguishing between them.

I cannot reach any conclusion as to what products Mr. Louvri r had in mind, when he ascribed the lack of success of electric zinc smelting, to their production.

If the phrase "excessive oxidation" is substituted for "production of blue powder" in his explanation, and this has been proved, and is not merely assumed from the formation of blue powder, then the source of this oxygen may profitably be looked for. I will say in passing, that it is improbable that a molecule of CO_2 can come in contact with carbon at the temperature of zinc reduction and escape being reduced to CO . These reactions are indeed successive, for CO_2 cannot react until it is formed. The time required for this molecular reaction occurring over a wide range of temperature must be very brief. It will be more profitable to look first into the two questions of temperature and contact, before recurring to this question of required time.

Two ideas thrown out by Mr. Louvri r in other connections afford valuable suggestions. The very rapid reduction (and distillation) of zinc, by the reaction $\text{ZnO} + \text{CO} = \text{Zn} + \text{CO}_2$, which is strongly endothermic, may exhaust the high-temperature heat and lower the temperature of the products, to such an extent that there is not enough high-temperature heat left to bring the surrounding carbon, gradually approaching the hottest zone, up to the temperature at which it will react promptly with CO_2 to reduce it. On the other hand, preponderance of evidence leads us to believe that CO must be present in considerable amount to reduce ZnO in the presence of CO_2 . If this be true there must be a very considerable reduction of CO_2 in the hottest zone of an electric furnace, and also in any cooler zone, where zinc is still reduced. This effect of rapidity of reaction is otherwise improbable, for the amount of CO_2 produced by reduction is directly proportional to the zinc reduced, and the more rapidly it is evolved, the less will be the amount of heat dissipated and lost. If reduction is effected by carbon, only CO would be formed, and there would be no CO_2 to be reduced. The effect of driving a furnace rapidly is to diminish the amount of heat lost by radiation, etc., other things being equal.

His other suggestion is, that the vertical position of one of the electrodes, which requires a certain free space, however slight, to move in, affords a line of least resistance for the escape of the evolved gases. The greater activity of reduction at the point of greatest heat, namely, at the end of the electrode, causes the charge to settle more rapidly above this point, and in this way diminishes the depth of the charge through which the gases must rise in order to emerge.

It is difficult to imagine an arrangement of electrodes which would secure a greater uniformity in the depth of the charge, but the idea advanced suggests the formation, within a uniformly filled furnace, of gas-filled cavities, due to greater chemical activity at these points. If the charge in the furnace is softened under the influence of heat, even to a very slight degree, these cavities would not be immediately filled by solid material dropping into them. The connection of such cavities would establish open passage ways throughout the charge, for the escape of gases, and their contact with hot carbon would

be reduced to a minimum. Thus the effect of an insufficient time exposure would be reached by poor conditions of contact.

With a charge consisting of ore mixed with carbon, which becomes more fusible as the carbon is removed by gasification, it is difficult to see how the formation of such cavities can be avoided, with the resulting irregular escape of the gases. I do not understand why experimenters in electric zinc smelting do not simplify their sufficiently difficult problem by experimenting with sublimed zinc oxide, which is practically infusible.

Although Mr. Louvrier appears to criticise all forms of electric furnaces, his specific references are to those of the arc type. To bring together my own criticisms of the data furnished, I will say in conclusion, that I think he overestimates the effect of oxidation. If indeed it occurs, and has the effect attributed to it, the source of the oxygen will probably be found to be air admitted around the electrodes, at the very point where it will do the most harm. In fact, it is very difficult to maintain electrodes passing through furnace walls, without allowing some air to pass in around them.

I think he has misjudged the effect of more rapid distillation, which should be to diminish, relatively, the loss of heat by radiation, etc., and consequently facilitate condensation of the zinc within the range of temperature through which this is possible. I think he has overlooked the time required to allow the liquid globules to come together after they are formed, and I think by fixing his attention on the time element in the reduction of CO_2 , to the exclusion of the elements of temperature and intimate contact, he has not noticed that the same effect may be produced in very much less time, under more favorable circumstances. Apparently he has not considered the effect of the local softening of the charge under the heat conditions of an electric furnace, in facilitating the irregular passage of the gases through it, and on the rapid cooling of the gaseous products.

But in spite of these somewhat minute criticisms, his paper gives evidence of good thinking, and of a desire to weigh carefully such information as has been given out by experimenters with electric furnaces. A great advance in general knowledge would result if more articles like his were written and published.

F. L. CLERC.

Estes Park, Colorado.

The Pinch Effect and the Pinch-Effect Electric Furnace

To the Editor of Metallurgical and Chemical Engineering:

Sir: On a short recent visit to this country, Dr. Carl Hering, the well-known author on the "Pinch Phenomenon," read a short lecture on this subject and demonstrated a small model of his furnace, designed on the pinch-effect principle. The lecture was held before a special gathering of the Faraday Society in the Northampton Institute in London and was evidently much appreciated by the members present. In his clear, lucid manner, Dr. Hering gave a brief but instructive résumé of his researches and the furnace as an outcome thereof.

The model itself was, owing to the fact that it must be as portable as possible, of the simplest nature, only intended for short demonstrations with lead as a charge; but the underlying principle of Dr. Hering's invention could be easily studied by means of it, and it seemed to work perfectly well under the prevailing conditions.

As a special feature, Dr. Hering drew our attention to the effect of two auxiliary channels emanating from the main electrodes, converging toward the center of the bath. It could be clearly seen how, contrary to expectations, the metal was readily sucked into these latter channels, while it was constantly ejected on the periphery of the main channels. Dr. Hering said that he reserved his comments regarding this point for the future; it was evident, however, that this added most effectively to the circulation of the bath.

Various designs were shown on slides and drawings, demonstrating different methods of application of the main principle.

On the whole, the furnace appeared to be very effective

along the lines demonstrated; it would, of course, be difficult to judge how it would behave as a commercial tool when used on liquid steel on a large scale and for a prolonged run.

One difficulty seems to be in the starting up from cold; but where liquid metal is obtainable for the start this seems to be of minor importance. Another is the application of two or more openings or channels near the bottom; every steel man is by nature somewhat shy of any holes or recesses in the bottom of his furnace, when he has a charge of several tons of expensive steel, superheated and highly liquid, on the hearth. But Dr. Hering assured that there was little danger in breaking through. No figures regarding the power consumption per ton of treated metal were given, but it was understood that some larger furnaces were under preparation with which such figures would be obtained.

During the course of the lecture, Dr. Hering stated that steel had been refined down to 0.006 per cent of sulphur. The amount of sulphur in the metal charged was not stated, but this is of less importance. Of far more importance, however, is the question how such far going desulphurization is effected in a furnace, where the slag blanket is heated secondarily by means of the underlying metal. This question was raised in the discussion; Dr. Hering attributed it to be caused through the "washing" action of the highly agitated metal. But even so, the result would be remarkable.

As well known, it is easy enough to lower the percentage of sulphur down to, say, 0.02 or 0.03 per cent, but to bring it down to 0.006 is considerably more difficult. As already pointed out by the author some four years ago, and since often enough corroborated in technical journals, one has a good indicator of a thorough desulphurization when calcium carbide is found to form in the slag; i.e., when the slag itself has been hot enough to show distinct traces of carbide, one may be fairly sure that the possibilities for the sulphur to enter into the slag as calcium sulphide were present.

Now that a rapid agitation of the steel-bath will facilitate the contact between the sulphur in the metal and the superheated slag is quite clear; another question is, however, how far would it be possible to drive the elimination of the sulphur when the slag is heated indirectly from the superheated metal bath, and what would be the metallurgical explanation of the reaction? This question might lend itself to discussion, since the mere statement that such and such has been observed is not always sufficient for a full understanding, and the subject is doubtless of vital interest.

The writer has had opportunity to observe that desulphurization slags from furnaces, where the slag was only heated from beneath, showed distinct traces of calcium carbide, and in these cases the elimination of the sulphur was very thorough, down to 0.008 per cent or thereabout; but as far as the writer is aware, this was partly due to the fact that such substances as would cause a kind of thermit reaction were added to the slag at the end of the melt, whereby the local temperature of the slag blanket was raised, thus altering the heat gradient between the metal and the slag in such direction as to obtain the result mentioned.

The quantity and nature of the material used were such as not to materially increase the cost of the operation; but this is not the point of issue, the question is—*is it metallurgically possible to effect a thorough elimination of the sulphur when the metal is hotter than the slag?*

It should be well understood that it is not the object of these lines to discuss the merits and disadvantages of any particular furnace principle, but the general question of refining with a metal hotter than the slag or vice versa is of sufficient importance to deserve a clearing up.

Reverting to the "pinch effect," it may be noteworthy that such effects may also, under circumstances, be observed in arc furnaces as will be seen from the following observations made by the author.

In an open-type reduction furnace, where only one top electrode and a bottom electrode were used, with single-phase current at 25 periods with 3500 amp, a slag layer about 7 in.

deep over the metal could be made to show the influence of the pinch effect in a marked degree. If the slag and reaction zone, for certain reasons, was made decidedly acid by adding a larger amount of quartz sand than usual, and then was so treated as to materially free it from metallic oxide, the ammeter began to swing over a large range.

At first it was thought that this was due to a temporary, violent reaction in the bath, but on closer observation it was found that the resistance of the slag was very much increased, which was natural, and that the current, instead of being more equally distributed over the whole bath, was concentrated to a narrow path under the electrode. At times it could be clearly seen how this path was contracted and the current interrupted for a moment, after which it again closed for a few minutes, when the phenomenon was repeated. During the whole time the voltage in the furnace was about 30 per cent higher than usual. The effect was clearly due to the "pinch" phenomenon.

By adding a shovel or two of metal oxide, thus increasing the conductivity of the charge and slag, the surging ceased, in spite of the more violent reaction. Also, by altering the nature of the slag through the addition of lime, all fluctuation was avoided.

This would indicate that a slag rich in SiO_2 is a less good "second-class" conductor. A true lime silicate, or a slag richer in CaO , seems to be a better second-class conductor, while, of course, the presence of free metallic oxides, such as iron oxide, etc., will sufficiently increase the conductivity to stop the pinch effect. But as this would mean a slag richer in metal, and also an unstable condition on account of the metal being constantly reduced, it seemed advisable not to employ a too acid slag, though its merits may otherwise justify such course.

JOH. HÄRDÉN.

Luton, England.

American Institute of Chemical Engineers Report of Boston Meeting

The fifth semi-annual meeting of the American Institute of Chemical Engineers was held in Boston, Mass., from June 25 to 28. The sessions were held in the Engineers Club. The meeting was opened by an address of welcome by Mayor Fitzgerald who spoke of the need of the services of technical men in solving municipal problems. Mr. Frederick P. Fish also addressed the Institute on behalf of the Massachusetts Institute of Technology.

The report of the secretary showed an increase of membership of 17 during the past half year, the total membership being 105, while 13 applications had not been acted upon.

The Patent Committee offered a resolution to President Wilson asking him to appoint a committee of experts to examine the patent system and recommend needed legislation. This resolution is printed at the end of this report.

A resolution was adopted indorsing the action of Secretary Lane in recommending Thos. E. Ewing for the position of Commissioner of Patents.

A resolution was also adopted addressed to the Senate, urging the confirmation of the appointment of Thos. E. Ewing as Commissioner of Patents and the hearty support of the new commissioner in the great work he has to do.

The paper by Wm. M. Booth of Syracuse, N. Y., on The Effect of Climate on Plant Location proved very interesting, the effect of temperature, humidity, rainfall, etc., being discussed.

Mr. Perry Barker of A. D. Little, Inc., in a paper on Distribution of Heat in the Operation of Steam Boilers, gave a number of interesting cases in which losses were reduced and efficiency increased.

In the afternoon two parties were formed to visit the U. S. Arsenal at Watertown and the laboratory and experimental paper mill of Arthur D. Little, Inc. The latter proved of special interest as all the machinery of a modern paper mill had been reproduced in miniature so that experimental runs of paper could be made.

The second party visited the works of the Hood Rubber Co., which is an up-to-date, well-managed plant producing rubber boots, shoes and automobile tires. All parts of the process were shown from the washing and rolling of the crude rubber to the completion of the finished product. The plant was clean and working conditions for the employees excellent. Some of the rooms were provided with ozonizers to remove odors and purify the air.

After dinner at the Engineers Club the papers of the evening were presented as after dinner addresses.

President T. B. Wagner addressed the Institute on Efficiency in Chemical Industry, speaking particularly of the many notable advances which had been made in the manufacture of corn products. Beginning as a very wasteful industry one by-product after another had been utilized until now 99½ per cent of the raw material is converted into useful products. While the price of corn has risen five fold the cost of starch has fallen three fold. This very interesting address is published in full elsewhere in this issue.

Dr. Louis J. Matos presented a very interesting paper on General Efficiency in Dye Houses and Bleach Works, while Dr. Grosvenor in his paper on Relation of the Manufacturer to the Patent System showed the very great influence which the United States Patent System has exerted in building up our industries.

All day Thursday was spent at Lawrence, Mass., where the mill of the Russell Paper Co. and the wool scouring, spinning, weaving and bleaching department, as well as the cotton mill of the Pacific Mills were visited. The enormous size of these mills is indicated by the fact that the woolen mill turns out about 12,000 pieces of 50 yd. each per week, while the capacity of the cotton print works is 700,000 yd. per day. The sample cards alone of this mill consume 1,000,000 yd. of cloth. The floor space of the print works is 1,500,000 sq. ft.

The bleaching as well as the printing of the cotton cloth was inspected, as well as the etching of the designs on the copper rolls used for printing the cloth. The working conditions in this mill seemed ideal, light and air being provided in abundance and cleanliness.

After dinner at the Engineers Club the following papers were read. Low and Mixed-Pressure Turbines was read by John G. Callan of Arthur D. Little, Inc. Very great interest was shown in the demonstration of the possibility of using the exhaust steam in low pressure turbines so as to increase the power developed by 50 per cent. The author pointed out that in spite of the great efficiency of the low-pressure turbine, it was more economical to use the exhaust steam for heating purposes as is generally possible in chemical plants.

Legal Control of Dangers to Health in Factories by Dr. Chas. F. McKenna was then read. Practical suggestions were made of laws which should be enacted for the purpose of remedying existing evils. As a part of this paper a standard legislative bill was presented which has been passed by the legislatures of fifteen states with reference to occupational diseases. This bill is published at the end of this report.

In the paper by Dr. F. W. Frerichs on Import Duties on Chemicals and Their Influence on Chemical Industry the danger to a number of existing industries from unwise legislation was pointed out.

On Friday morning at 9 o'clock the members of the Institute and their guests embarked on the ocean-going tugboat *Pallos* and were conveyed to the pier of the New England Gas and Coke Co. at Everett. The admirable equipment for unloading coal was first inspected. This consists of seven towers equipped with scoops of the clam type which are capable of taking 5000 tons of coal from the hold of a steamer in two hours. A train of cars is filled and emptied automatically distributing the coal to the bins or storage piles as desired. In addition to the 600,000 gross tons of coal carbonized at this plant annually 1,000,000 tons of steam coal are handled. The coke ovens were then inspected as well as the ammonia recovery plant and the gas purifying apparatus. All of the gas used by Boston is supplied by this plant.

The party then reembarked on the towboat for the trip to Marblehead. As the day was warm the ocean trip proved very enjoyable. Marblehead was reached at about 1 o'clock. Dinner was taken at the New England Yacht Club. President Wagner presided at the dinner and introduced various speakers between the courses. He first called upon the first president, Dr. Samuel P. Sadtler who spoke of the growth of the Institute since its organization, having increased its membership 500 per cent. Mr. Henry Howard, chairman of the Local Committee, was then called upon, after which Mr. John V. N. Dorr, of Denver, Col., who had come East to attend the meeting, spoke of conditions in the West, especially the mining and metallurgical industry. He suggested the possibility of utilizing in widely different industries apparatus which had been developed in special metallurgical industries. Mr. John C. Hebben, of Providence, closed the program with a number of very amusing anecdotes.

The party then adjourned to the spacious verandas of the club where cigars and the beautiful view of the harbor were enjoyed. The return to Boston was made on the towboat ending a very enjoyable trip.

After a light supper at the Engineers Club the following papers were read: Depreciation and Obsolescence by Richard K. Meade, of Baltimore, Md. In this paper the author discussed in a very thorough manner the allowance which should be made for wear and tear as well as obsolescence in chemical plants, especially cement mills.

Professor Wm. P. Mason, in a paper entitled A Peculiar Case of Lake Pollution, showed how typhoid bacteria were carried 10 miles, the entire length of one of the needle lakes of New York State, and produced a very severe epidemic of this disease.

A paper by J. C. Olsen and A. H. Callaghan on The Drying of Linseed Oil with Red Lead and White Lead was then read. The authors had determined the amount of water and carbon dioxide given off during the drying of red lead and white lead paint. They found that with the same absorption of oxygen very much smaller amounts of water and carbon dioxide were given off by the red lead paint than by the white lead paint or pure linseed oil. This was suggested as an explanation of the superiority of the red lead paint as a protective coating for iron and steel.

New York was chosen as the place for the annual meeting which will be held during the first half of December.

After the usual resolutions of thanks, the Institute adjourned.

On Saturday two plants in Providence, R. I., were visited. One of these was the United States Finishing Co. where the processes of bleaching, dyeing and printing were inspected. The other plant visited was the Gorham Mfg. Co. where the manufacture of solid silver and silver plated table and ornamental ware was inspected. The standard thickness of silver plate in this plant is 1/1000 in.

About 1 ton of silver is used daily. The casting and finishing of large bronze statues was inspected with great interest.

* * *

Patent System Reform

As mentioned above, a resolution to President Wilson was adopted asking him to appoint a committee of experts on the patent system. This resolution is as follows:

"To the Honorable Woodrow Wilson, President of the United States:

"Whereas, the patent system of the United States has had an immense influence on the industrial development of this country, by stimulating competition by improvement, beneficial to the consumer as well as to the producer.

"Whereas, our patent system has become inadequate to cope with the rapid growth and changing conditions of the country so that now it seems imperative that the different departments of our patent system should be modified to meet the new conditions,

"Whereas, inadequate and harmful legislation is apt to result unless the subject is given mature and competent consideration.

"Resolved, the American Institute of Chemical Engineers does hereby respectfully call the attention of the President of the United States to the urgent need of appointing a competent commission to investigate thoroughly the United States patent system, and to prepare a report and submit recommendations before the passage of any legislation bearing on this delicate and complicated subject; the American Institute of Chemical Engineers, furthermore, suggests, that such a commission should be composed of men who are best acquainted with this subject, as well as representatives of all the important interests affected by the patent system, such as the courts, inventors, technologists, manufacturers and patent attorneys.

"American Institute of Chemical Engineers.

"T. B. Wagner, President: J. C. Olsen, Secretary."

* * *

Bill on Occupational Diseases

As part of Dr. McKenna's paper mentioned above, a legislative bill, which has been passed by the legislatures of fifteen states, was given. This reads as follows:

"An act to require the reporting of certain occupational diseases and to provide for its enforcement.

"Be it enacted, etc., as follows:

"Section 1—Report of Occupational Diseases:

"Every physician in this state attending on or called in to visit a patient whom he believes to be suffering from poisoning from lead, phosphorus, arsenic, brass, wood alcohol, mercury or their compounds, or from anthrax, or from compressed air

illness, or any other ailment or disease contracted as the result of the nature of the patient's employment, shall within forty-eight hours send to the state board of health a report stating:

"(a) Name, address and occupation of patient.

"(b) Name, address and business of employer.

"(c) Nature of disease.

"(d) Such other information as may be reasonably required by the state board of health.

"The reports herein required shall be on or in conformity with the standard schedule blanks hereinafter provided for. The posting of the report, within the time required, in a stamped envelope addressed to the office of the state board of health, shall be a compliance with this section.

"Section 2—Blanks for Reports:

"The state board of health shall prepare and furnish, free of cost, to the physicians included in Section 1, standard schedule blanks for the reports required under this act. The form and contents of such blanks shall be determined by the state board of health.

"Section 3—Reports Not Evidence:

"Reports made under this act shall not be evidence of the facts therein stated in any action arising out of the disease therein reported.



T. B. WAGNER

"Section 4—Penalty:

"Any physician who neglects or refuses to send the report or reports as herein required shall be liable to the state for a penalty of ——— dollars for each offense, recoverable by civil action by the state board of health.

"Section 5—Transmission of Reports:

"It shall furthermore be the duty of the state board of health to transmit a copy of all such reports of occupational disease to the (proper official having charge of factory inspection).

"Section 6—Time of Taking Effect:

"This act shall take effect on ———."

Dr. T. B. Wagner is the president of the American Institute of Chemical Engineers. Dr. John C. Olsen, Polytechnic Institute, Brooklyn, N. Y., is the secretary.

The Western Metallurgical Field

Hydrometallurgy in Montana

Leaching processes for copper and zinc ores are attracting attention again and Montana is the scene of the greatest activity along this line in the United States. At Butte the Butte & Duluth company is leaching copper ores with sulphuric acid and precipitating the resulting solution electrolytically. In Anaconda an 80-ton experimental plant is being built at the Washoe smelter of the Anaconda Copper Mining Co., for the purpose of treating tailings from concentration of second-class ores. A third process is under way at Helena, where the Northwestern Metals Co. is developing a process for treating zinc ores by dry chlorination.

A peculiar fascination and interest attaches to hydrometallurgical processes, probably due to the fact that they are so easily demonstrated in a laboratory way. In this respect they resemble the schemes for electrolytic amalgamation, which seem to work so beautifully in a simple beaker test, but develop unconsidered difficulties on a commercial scale. Mechanical problems in the way of handling solutions and noxious gases are usually regarded as the greatest obstacles to hydrometallurgical success. This is particularly true in the treatment of base metal ores, where pounds of metal must be dissolved and precipitated, as distinguished from the cyanidation of precious metal ores which contain only ounces or fractions of an ounce of metal. Another difficulty lies in the necessary use of a solvent which affects several metals similarly, and thus extracts the undesirable with the desirable.

Experience with hydrometallurgy leads to the conclusion that it is applicable with success in special cases, and that a process profitably applied at one place cannot be adopted without alteration in another locality. Thus each proposition must be considered individually, and a process devised that will meet the peculiar needs of each, if such a thing is possible. Natural conditions sometimes favor hydrometallurgy, presenting an ore free from deleterious elements that will foul the solutions, or offering special advantages in the way of cheap power, or acid that can be made as a by-product from other operations.

But if these processes are not of universal application, they are none the less admirable in the special cases in which they are successful, because they frequently provide a means of treating ores and products that would not be handled by standard methods and thus conserve a valuable mineral resource.

The Anaconda experiment is based on a process devised by Mr. Frederick Laist, superintendent for the Anaconda Copper Mining Co. It will be applied to the recovery of copper from concentration tailings. The steps in the process include a roast with salt to render the copper soluble and extraction with dilute sulphuric acid. Precipitation will be by hydrogen sulphide. The reactions involved are simple and the chemicals required can be obtained without great cost. Salt is inexpensive, sulphuric acid can be made by the chamber process as a by-product of roasting concentrates, and hydrogen sulphide can be generated by the action of sulphuric acid on iron matte, the latter being cheaply produced from iron ores in the district. A question has been raised as to the use of hydrogen sulphide on account of its poisonous nature and the possibility

of resulting accidents; but in view of its former extensive use in the chlorination process, the objection seems not important.

The dry chlorination of zinc-lead ores at Helena is one of several experiments of a similar kind that have been considered for several years. Chlorine gas is the active agent employed in attacking the ore and rendering the metals soluble, after which they are recovered by steps in precipitation which are susceptible of considerable variation. It is expected that experiments on a commercial scale will be commenced this summer, and pending the actual operation of the process, no definite information is available for publication. The results will be awaited with interest, for the process has a distinct bearing on the low-grade, complex ore problem. Success at Helena will be encouraging, but failure will not necessarily mean that dry chlorination in the hands of other investigators may not prove successful.

Recovering Sulphur from Smelter Fume

The past few years have been rich in effort and experiment to eliminate sulphur and sulphurous products from smelter fume, and thereby remove the great bone of contention between smelting and agricultural interests. Methods of neutralization followed by bag-house recovery have been proposed and adopted, as also has been the Cottrell process of electric precipitation. In METALLURGICAL AND CHEMICAL ENGINEERING for November, 1912, the Thiogen system of fume control was described, which has been experimented with at the Penn Chemical smelter, Campo Seco, Cal. This process aims to produce elemental sulphur from roaster gases by reducing the sulphur dioxide with carbon contained in an oil spray and passing the mixture over calcium sulphide. Although the usual mechanical difficulties have been experienced in applying this process, it is understood that the prospect is so favorable for its success as to warrant the company erecting a new roasting plant of Wedge furnaces and the necessary equipment for the treatment of the roaster gases.

More recently another process, devised by Mr. William A. Hall, has been announced and is about to be tried in California, where the smelting industry has been seriously handicapped by conflict with the farmers. **The Hall process** takes a step in advance of all others by preventing the formation of oxidized sulphur gases which must be subsequently reduced or neutralized. Mr. Hall's scheme involves the direct distillation of sulphur from the ore and its recovery in some suitable form of gas washer. Thus the problem apparently is attacked in the most direct manner, avoiding intermediate steps of oxidation and reduction.

Distillation in the Hall process depends on roasting ore at a temperature between 700 deg. and 900 deg. C., by the direct application of a reducing, or at least non-oxidizing flame, together with the introduction of a certain amount of water in the form of steam. Under these conditions the base metals are oxidized and the sulphur distilled without combining with oxygen and can be easily collected.

The technical success of the process still leaves unsolved the interesting problem of the disposal of the product. The net consumption of sulphur in the United States in 1911 is given as approximately 200,000 tons, or, in round numbers, 500 tons per day. Considering the magnitude of modern smelting operations, it is conceivable that the application of the Hall process at one or two plants might easily produce more sulphur than is now consumed. And if the cost of production were low, which it may be, the output would seriously affect the market for present producers. Of course, there is always the prospect that cheaper production would stimulate consumption and thus take care of an increased output; but with the wide adoption of the process in this country and abroad it is conceivable that the sulphur market would be promptly glutted. The prospect is similar to that which would result from converting all our smelter fume into sulphuric acid; it could not be utilized and its disposition would become a serious matter. Nevertheless the Hall process is a welcome addition to metallurgy, and its results will be awaited with great interest.

Dry Ore Dressing

A stimulus has been given to dry processes of ore dressing by the invention and promotion of several machines which apparently have a legitimate field of usefulness. Mention has been made in this journal of the complete experimental plant which the **Sutton, Steele & Steele** company has erected in Denver for the demonstration of dry concentrating tables and electrostatic separators. Ample financial support has been given to the processes of this company and has resulted in arousing an interest in dry methods of ore treatment such as has not been manifest for some time. The machines and processes are not entirely new, but have not been widely advertised, and the construction of the present plant is expected to call attention more strongly to the possibilities of dry ore dressing and the advantages of that system in some localities.

An entirely new departure in this field is the dry jig developed in Denver by Mr. **A. M. Plumb**. A pulsating current of air is applied to the jig bed similar to the pulsations of water in wet jigging, and a remarkably clean separation of mineral and gangue, or mixed minerals, results. Used in connection with electrostatic separation or wet dressing, the machine is an accessory which seems to occupy a field of usefulness that has been neglected. The jig has its limitations, chief among which is its inability to separate products ground finer than 150 mesh; but it has the desirable qualities of being practically foolproof and positive in its action. A machine of commercial size has a jigging box about 24 in. long by 6 in. wide and occupies only a small space. A great variety of ores and mill products have been tested and the jig will be placed on the market this summer. One of the first installations will be made at the Sunnyside mill in Silverton, Col.

Rochester Meeting of the American Chemical Society

The 48th annual meeting of the American Chemical Society will be held in Rochester, N. Y., from September 9th to 14th.

The program will open with a general meeting on Tuesday at 10 a. m. in the assembly hall at Kodak Park; in the afternoon the works of the Eastman Kodak Co. will be visited; in the evening there will be a smoker.

Divisional meetings will be held on Wednesday all day, and on Thursday and Friday in the morning at the University of Rochester. The President's address will be given at the East High School on Wednesday night, and a subscription dinner will be held on Thursday night.

Dr. Charles L. Parsons, Box 305, Washington, D. C., is the secretary of the American Chemical Society.

Butte Meeting of the American Institute of Mining Engineers

The 105th meeting of the American Institute of Mining Engineers will be held at Butte, Montana, beginning Monday, August 18th, 1913, with a stop-over at Great Falls, on August 16th.

On Monday, August 18th, technical sessions will be held in the morning and evening, while visits to the mines are arranged for the afternoon.

Tuesday, August 19th, will be devoted to a trip to Anaconda and the Southern Cross mine, while an entertainment is scheduled for the evening.

On Wednesday, August 20th, technical sessions will again be held in the morning and evening, while visits to mines and smelters will be made in the afternoon.

On Thursday, August 21st, the concluding technical session will be held in the morning, while for the afternoon visits to mines and for the evening a banquet are being arranged.

Charles W. Goodale is chairman of the general committee in charge of the meeting.

Bradley Stoughton, 20 West 30th Street, New York City, is the secretary of the American Institute of Mining Engineers.

Colorado Meeting of the American Electrochemical Society

The twenty-fourth general meeting of the American Electrochemical Society will be held in Denver on Sept. 9, 10 and 11, 1913.

The special train with the Eastern members is expected to arrive at Denver on Monday afternoon, Sept. 8. Automobiles will take the party for a ride about the city's comprehensive park and boulevard system, finishing at the Hotel Shirley, the Society's headquarters. In the evening of Monday there will be registration, a meeting of the Board of Directors and a general good time of getting together.

Tuesday, Sept. 9—A morning session will be held in Denver for the reading and discussion of papers. For the afternoon various visits are being arranged. In the evening an informal smoker will be held.

Wednesday, Sept. 10—An early car will be taken for Boulder, where a session for the reading and discussion of papers will be held at the University of Colorado. After lunch automobiles will be taken up Boulder canyon to the power plant of the Central Colorado Power Company, at the entrance to the Boulder tungsten district.

Thursday, Sept. 11—In the morning a session will be held for the reading and discussion of papers at the School of Mines in Golden. For the afternoon various visits are being arranged. In the evening an informal picnic dinner will be held on Look-out Mountain, just west of Denver, behind Golden.

The program of papers to be presented and discussed in the three sessions at Denver, Boulder and Golden will be announced in the next *Bulletin* of the Society. The potentialities of the applications of electrochemistry in Western metallurgy will naturally be prominent in the program.

Various visits are being arranged to places of great natural beauty and picturesqueness as well as to chemical and metallurgical works; there will be much of interest and also some matters of decided novelty.

For the ladies' entertainment during the three days in Denver a trolley ride to the Foot Hills of the Rocky Mountains, a luncheon and a theater party have been outlined with other rides and visits to places of interest.

Friday, Sept. 12—In the morning the return trip will be begun which will take the party first to Colorado Springs, where the forenoon will be spent in a visit to the Garden of the Gods or places of metallurgical interest. In the afternoon a trip will be made up Pike's Peak, where it is intended to hold the concluding session of the meeting, the program of which will be announced in the next *Bulletin* of the Society.

Saturday, Sept. 13—A further side trip will be made from Colorado Springs over the magnificent "Short Line" into the gold district of Cripple Creek.

On Saturday night the special train will start from Colorado Springs on the return trip to the East.

According to present indications the attendance will be large and it is hoped that it may be possible to start a special train right from New York City. The trip from New York to Denver will be made over the New York Central and Burlington lines, the return trip on the Santa Fe. The train will be composed of the highest grade Pullman equipment, drawing-room, compartment, observation and dining-cars; also a combination buffet smoking-library car with barber shop and bath. The entire train will be electrically lighted and all steel equipment will be used.

The complete schedule of the special train (which will be the first electrochemical train ever arranged) will be published in detail with connections from other cities, details of fares, in a pamphlet to be issued shortly.

The special train will leave New York City from Grand Central Terminal on Saturday, Sept. 6, at 10.30 a. m., arrive Chicago on Sunday, 7.50 a. m., leave Chicago at 9.45 a. m. and arrive in Denver on Monday, Sept. 8, at 1.15 p. m.

The return train will leave Colorado Springs on Saturday, Sept. 13, at 10.30 p. m. (Western time), arrive Kansas City on

Sunday at 6.55 p. m., leave at 7.30 p. m., arrive Chicago on Monday at 9.15 a. m., (Central time), leave Chicago at 10.15 a. m., and arrive in New York City on Tuesday, Sept. 16, at 9.17 a. m.

All requests for Pullman car reservations and railroad transportation should be made to the chairman of the Transportation Committee, Mr. J. M. Muir, 239 W. 39 Street, New York City, from whom any additional information concerning transportation may be obtained.

Dr. J. W. Richards, Lehigh University, South Bethlehem, Pa., is the secretary of the American Electrochemical Society.

International Electrical Congress, San Francisco, 1915

Preliminary plans for the International Electrical Congress which is to be held in San Francisco during the week beginning September 13th, 1915, in conjunction with the Panama-Pacific International Exposition, and under the auspices of the American Institute of Electrical Engineers, are being effected by the Committee on Organization.

In the week preceding the Congress there will be a meeting of the International Electrotechnical Commission.

The Congress is being divided into twelve sections, as follows:

1. *Generation, Transmission and Distribution.*—Central station and substation design, control and operation. Long distance transmission of electric power.
2. *Apparatus Design.*—Generators, motors and transformers. Prime movers and their relations. The rating of machinery.
3. *Electric, Traction and Transportation.*—City, surface and rapid transit railways; interurban and trunk lines; electric vehicles, ship propulsion, mining, railways, elevators and hoists.
4. *Electric Power for Industrial and Domestic Use.*—Factories, mills, refrigeration, heating devices, etc.
5. *Lighting and Illumination.*—Arc and incandescent lighting; the science and art of illumination.
6. *Protective Devices; Transients.*—Switches, circuit breakers; condensers; electrostatics; disruptive phenomena; high frequency phenomena.
7. *Electrochemistry and Electrometallurgy.*—Electrolytic and metallurgical apparatus and processes.
8. *Telegraphy and Telephony.*
 - (a) All communication of intelligence by the use of wires.
 - (b) Electromagnetic waves and radio telegraphy and telephony.
9. *Electrical Instruments and Electrical Measurements.*—Switchboard, portable, standard and absolute instruments. Testing and standardization methods; absolute measurements.
10. *Economics of Central Stations and Systems.*—Load factors, power factors and all problems affecting the economy of central stations; also rates and regulation by public service commissions.
11. *Electro-Physics.*—Radioactivity; Röntgen rays; gas and vapor conduction; electron theory; constitution of matter.
12. *Miscellaneous.*—Such as history of literature of electrical engineering; symbols and nomenclature; engineering education and ethics.

In each section it is desired to include as many as practicable notable papers dealing with the status or the progress of the art. Offers of papers and suggestions in this connection should be directed to the Secretary of the Committee on Organization, Dr. E. B. Rose, Bureau of Standards, Washington, D. C.

The Non-Ferrous Metal Market

During July the market for non-ferrous metals has been dull and decidedly weak in some cases. The usual midsummer dullness has dominated the market generally and in the case of tin the conditions abroad have thoroughly unsettled the domestic market. Spelter has rallied somewhat, and the present prices probably will forestall the further closing of mines and concentrating plants. Copper and lead have not shown much tendency to change.

Copper.—Buyers have not appeared anxious to enter the market, but it is believed that they cannot hold off much longer. Prices have been reduced in an effort to stimulate business but sellers of Lake brands have shown a disposition to stand by the fixed prices and await improvement in the market. Electrolytic is quoted at 14.20@14.25 and Lake at 14.45@14.50 cents.

Tin.—Continued liquidation in the London market for several weeks has resulted in greatly reducing the quotations for this metal and the continued decline has stopped buying in this market. The last quotations are unsettled and may go lower. July tin was quoted in New York at 39 cents.

Lead.—This market has remained unchanged since our last report. The St. Louis quotation is 4.17½@4.20 cents; New York, 4.20@4.35 cents.

Spelter.—There has been an improvement in the price for spelter since our last report. A curtailment in production has followed the closing of several mines and concentrators, but at last reports all inquiries for metal were being freely met. St. Louis prices are 5.05@5.15 cents and New York, 5.20@5.30 cents.

Other Metals.—Business in aluminium is light and prices are unsteady at 23@24 cents, New York. Antimony is quiet and prices are unchanged at 7.50 to 8.75 cents for various brands. The quicksilver market is quiet with prices at \$40 per flask of 75 lb., New York, and \$39.50, San Francisco.

Carrying the Injured in Accidents

It has recently been said, and is undoubtedly true, that there is at present such a strong tendency among the large American manufacturers to protect their employees against accidents that in this respect the manufacturers are willing to go to extremes. Nevertheless accidents will happen, and it is an important problem how to render first help in an accident.

With respect to the question of carrying an injured man safely away, "the splint stretcher" which has been devised after years of experimentation by Dr. C. F. Stokes, surgeon-general of the United States Navy, marks an important advance. The Stokes Splint Stretcher is a basket of heavy iron wire, properly braced, long and wide enough to accommodate a man, with nickel plated handles on the sides and at the corners. The braces also serve as runners when dragging the injured is the only way removal can be accomplished. Within the basket a partition extends from the foot and well up toward the center. This keeps the legs well separated and affords each an extra support. Since the majority of accidents needing a stretcher require bandaging of one or both legs, the advantage of this arrangement is obvious.

The construction and the method of handling are described and clearly shown in illustrations in a little bulletin of the Kny-Scheerer Company, 304 West 27th Street, New York City.

It is perhaps interesting to note in this connection that this "splint stretcher" has first found extended use among the copper mining companies, being employed in not less than six of the large copper companies of Arizona, like the Inspiration, Copper Queen, etc. More recently it is being introduced also by various iron mining companies.

Flotation in Arizona

Interesting progress of the flotation process is reported from Arizona. The Inspiration Consolidated Copper Company has had for some time a comparatively large experimental plant in operation, using the Minerals Separation process, with a capacity of 50 tons daily.

The results have been so encouraging that the Inspiration Company is now going to build a larger plant of 600 tons daily capacity. This is expected to be in operation in about six months.

A detailed and fully illustrated description of the Minerals Separation Flotation process, as worked at the Kyloe Copper Mines, N. L., was given in a paper by H. Hardy Smith in the March issue of this journal, page 131.

Cyanide Practice in the Black Hills, South Dakota.—II.

By H. C. Parmelee

Bismarck Consolidated Mill

Practice similar to that at the Wasp No. 2 mill, described in our last issue, is to be found at the mill of the Bismarck Consolidated Mines Co. Conditions at these two adjoining properties are similar in that the ore is low-grade, \$2 to \$2.50 per ton, and is readily amenable to cyanidation by leaching when crushed to $\frac{1}{4}$ in. The Wasp and Bismarck mills are the only two now operating in the Black Hills following the practice of dry crushing and sizing. The Bismarck ore is oxidized and offers few obstacles to successful cyanidation. In crushing it probably produces more fine material than does the coarse quartzite of the Wasp No. 2, but not sufficient to cause serious difficulty in leaching.

The mill site is a rather steep hillside, and the mill is built in terraces, with separate bins for the storage of ore at the head of the coarse-crushing, fine-crushing and leaching departments. The building is of frame construction and well lighted, and the interior arrangement is such that all machines are readily accessible for inspection and repair. Electric power is used throughout, separate motors being conveniently arranged for the operation of the different departments. The capacity of the mill is 300 tons per 24 hours.

Coarse Crushing in Gyratories

Ore from the different mine workings is delivered to a 500-ton bin at the head of the mill. The initial breaking is done in a No. 5 Gates Gyratory which reduces the ore to 3-in. size, this product being delivered onto a 22-in. belt conveyor, 28 ft. centers, running at an inclination of 21 degrees. The belt discharge is split and flows over two grizzlies arranged in the form of an inverted V. These are each 2 ft. wide by 6 ft. long, and have $\frac{1}{2}$ -in. openings. The oversize passes directly to two No. 3 Gates gyratories, where it is crushed to $1\frac{1}{2}$ in. and combined with the grizzly undersize. An 18-in. belt and bucket elevator, 35 ft. centers, running at 250 ft. per min., raises the crushed ore to the roll-bin of 400 tons capacity. The three gyratories, conveyor and elevator are all operated from one line shaft driven by a 75-hp. motor.

Rolls are used as secondary crushers. From the roll-bin the ore is delivered by a shaking feeder to a set of 15 in. by 36 in. rolls which crush to about $\frac{1}{2}$ or $\frac{3}{8}$ in. This product is elevated in a 16-in. belt and bucket elevator, 51 ft. centers, running at 207 ft. per min., and is delivered to a shaking screen vibrating 240 times per min. The screen is of woven wire, 2 ft. by 7 ft. in area, with apertures $\frac{1}{4}$ in. by 1 in. The undersize is a finished product and passes to a bin of 500 tons capacity, while the oversize is returned to a second set of rolls similar to the first, where it is crushed to $\frac{1}{4}$ in. and again elevated to the screen. The rolls and elevator are driven by a 65-hp. motor.

Leaching Vats Filled by Belt Conveyors

The leaching department contains six vats arranged in two rows of three each, across the mill. Five of the vats are 28 ft. diameter by 9 ft. deep, and have 216 tons capacity; the sixth is 30 ft. diameter by 12 ft. deep, and has a capacity of 334 tons. They are filled by a system of conveyor belts. No. 1 belt runs parallel with the crushed-ore bin, from which it receives ore through four rack and pinion gates. The ore is discharged onto No. 2 belt, running at right angles with No. 1, and extending over two vats, one at the end of each row. Belts Nos. 3 and 4 run at right angles with No. 2, extending over the other two vats in each row. By this system it is possible to divert ore from one belt to another, and thence to any of the six vats. Each conveyor is driven at a speed of 320 ft. per min. by a separate 3-hp. back-gear motor. The capacity is from 75 to 100 tons per hour. The dimensions of the belts are as follows:

Belt No.	Width, Inches	Centers, Feet
1.....	22	32
2.....	20	47
3.....	25	60
4.....	20	60

Leaching Practice

The method of applying solution to the ore differs from that used in other mills in the district. While the leaching vats are being filled with ore, 5-lb. cyanide solution is forced through the false bottom of the vat, at a rate which causes it to rise with the ore, but never to come above its surface. When the tank is filled and leveled, the moisture in the ore is estimated, and enough cyanide is dissolved on top of the charge to bring the strength of the solution up to five pounds per ton. The ore is covered to a depth of two or three inches with solution, and allowed to stand six hours before the leaching cock is opened. When the 5-lb. solution has nearly all percolated through the charge, the cock is closed and the ore is flooded with 4-lb. solution. As soon as air bubbles cease to appear at the surface, the leaching cock is opened and percolation proceeds. The ore is kept covered with solution until 24 hours have elapsed following the first covering with 5-lb. solution, and then allowed to drain. After this the vats are flooded at intervals with 2.5-lb. solution for a period of 72 hours, and allowed to drain before the application of wash water. From 35 to 45 tons of wash water is applied to each vat, and from 16 to 24 hours is required to thoroughly wash and drain a vat.

On account of the situation of the mill with reference to the railroad, and the consequent lack of room for tailing dump, it is necessary to unload the vats by hand and stack the tailing dry. Each vat has four discharge gates in the bottom, through which tailing is shoveled into cars beneath, and trammed to the dump. From 14 to 16 hours time is required to dump and fill a vat.

All water used in the mill flows by gravity from the mine to storage tanks placed on a level with the crushing floor of the mill. Storage tanks for strong and weak stock solutions, and gold solution, and sump tanks for barren and weak solutions, are suitably placed. Two 5 by 6 Deming triplex pumps are used to transfer barren and weak solutions from sump to stock tanks. Solutions are standardized to proper strength in the stock tanks, and drawn to the leaching vats as required. Lime is added dry at the large gyratory, being shoveled in with ore at the rate of about six pounds per ton.

Precipitation on Zinc Shavings

The zinc boxes have a combined capacity of 68 cu. ft. of shavings, and under present operating conditions the quantity of solution precipitated averages 160 to 170 tons per 24 hours, or about 1 ton per ton of ore treated. Only strong solution is precipitated; the gold-bearing weak solution is pumped back to the strong solution stock tank as required to maintain the necessary volume for leaching.

The strength of solutions has already been indicated: 4 to 5 lb. KCN per ton for strong, and 2.5 lb. for weak. Protective alkalinity is maintained at 0.8 to 1.2 lb. CaO per ton. During the month of April, 1913, the chemical consumption per ton of ore treated was: CaO, 6 lb.; Zn, 0.405 lb.; KCN, 0.402 lb. The mill has not been in operation long enough to determine what the average extraction should be, but thus far it has ranged from 70 per cent to 80 per cent on individual vats.

The value of ore and tailing is determined by sampling each vat before and after leaching. The tonnage of gold solution precipitated is estimated by a meter of the oscillating box type, placed below the zinc boxes. The tailing solutions from the several boxes are combined and flow through a launder in which is a smaller launder that cuts out 1 to 2 of the stream and delivers it to the meter.

There are no unusual features in the clean-up and acid-treatment of the gold precipitate.

The New Reliance Mill

The cyanide practice at the mill of the New Reliance Gold Mining Co., near Trojan, S. D., presents some features which are unusual in the Black Hills. Some months ago the present management made important changes in the mill system, chief among which was the adoption of the continuous decantation system of slime treatment in Dorr thickeners, followed by

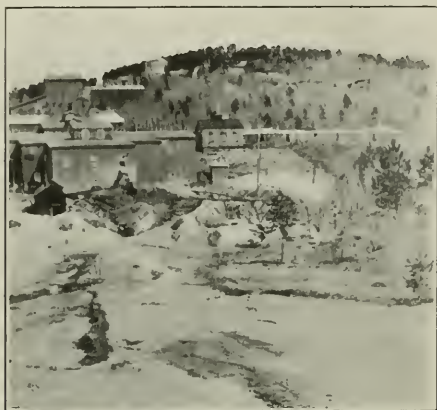


FIG. 1—NEW RELIANCE MILL, TROJAN, S. D.

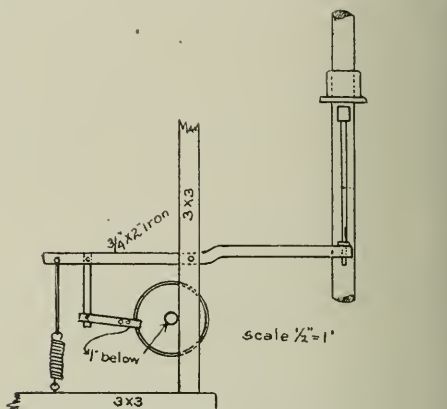
vacuum filtration in a Portland rotary filter. In this respect the New Reliance mill is radically different from the other mills in the district. Another point of difference is the use of stamps instead of rolls and Chilean mills. Considering the simplicity of all operations in the slime department, and the general improvement in results obtained, the changes in the system appear to have been fully justified. A view of the mill and part of the slime pond appears in Fig. 1.

Crushing with Stamps in Cyanide Solution

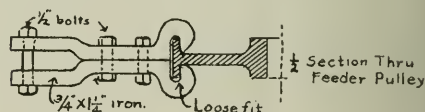
Excluding the Homestake, the New Reliance is the only mill in the Black Hills using stamps. Rolls and Chilean mills are generally adopted, and while it is true that these machines are proving successful as grinders for very hard ores, nevertheless the higher extraction obtained on hard refractory ores at the New Reliance suggests one point of superiority of the stamp as a machine for crushing in solution. The agitation of the ore and solution in the stamp mortar is much more effective than in rolls or Chilean mills, and undoubtedly aids the process of solution, since it leaves less of the precious met-

The gyratory product is elevated to the stamp bins by the ordinary belt and bucket elevator. Some trouble has been experienced on account of fine, moist ore packing in the elevator buckets, but lately that has been overcome by a simple device consisting of a plate of iron fitting in the bottom of each bucket, and loosely bolted in that position. The bolts are long enough to allow the plate a vertical movement of about an inch. The result is that when the bucket passes over the head pulley of the elevator to discharge its load, the plate falls and clears the bucket of ore. The device is shown in detail in Fig. 2.

Thirty 1000-lb. stamps dropping 96 times per minute through a height of from 6 to 9 in. show an average daily duty of $3\frac{1}{3}$ tons per stamp. Solution is used at the rate of 4 to 4.5 tons per ton of ore. A modified form of Challenge feeder is used, in which a more positive motion is imparted to the disc than



Arrangement of Stamp-Feeder Mechanism



Details of Grip
Scale $1\frac{1}{2}''=1'$

FIG. 3—RELIANCE STAMP-FEEDER MECHANISM

can be obtained by the usual friction band. The arrangement of the mechanism and details of the grip are shown in Fig. 3. The idea was suggested by the ease with which a pulley may be turned by gripping its edge with a monkey wrench.

Screen Analyses

The stamp mortars are fitted with No. 423 Ton-Cap screen, 0.023 in. slot. An average of 410 tons is passed through each screen before it is removed on account of wear. This appears to be a low average, but much of the ore is exceedingly hard and causes excessive wear on screens. The stamps yield a product having the following screen analysis:

Mesh	Per Cent
— 20 + 40	10.00
— 40 + 60	14.17
— 60 + 100	20.00
— 100 + 150	18.33
— 150 + 200	4.17
— 200	33.33
	100.00

The pulp is separated into sand and slime products in a Dorr classifier, a light spray of mill solution being applied to the sand just before it is discharged. From 60 to 65 per cent of the crushed ore is classified as slime, and 40 to 35 per cent

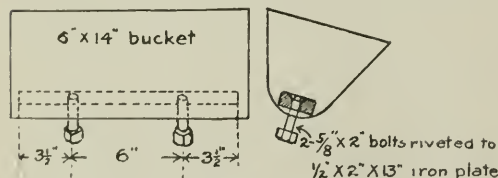


FIG. 2—BUCKET CLEANER

als to be dissolved in the subsequent sand and slime treatment.

The mill feed is a mixture of oxidized, semi-oxidized and hard blue ores, the latter constituting from 20 per cent to 30 per cent of the whole. The successful treatment of so large a percentage of blue ore is important, in view of the fact that formerly it was regarded as too refractory, and consequently was not mined to any great extent.

The ore is first crushed in a gyratory, and automatically sampled by a bucket sampler which cuts out about 1/100 of the feed. This sample is further crushed, and passed through Jones riffles to obtain a suitable sample for the assay office.

as sand. The following screen analyses show the efficiency of the separation:

Sand Product		Slime Product	
Mesh	Per Cent	Mesh	Per Cent
+ 40	27.50		
+ 60	25.00		
+ 100	28.34	+ 100	3.12
+ 150	14.17	+ 150	10.94
+ 200	1.66	+ 200	6.25
— 200	3.33	— 200	70.69
	100.00		100.00

Sand Treatment

The sand product from the classifier is leached in five 100-ton vats, which are filled by the ordinary revolving distributor. The first solution applied is the overflow from No. 1 thickener of the continuous decantation system. Percolation proceeds while the vat is filling, and commences as soon as clear solution can be obtained. This method of treatment yields a good grade of clear solution for precipitation, and eliminates the necessity of a clarifying tank ahead of the zinc boxes. A further advantage gained by leaching with the overflow from No. 1 thickener, instead of with mill solution, is that the former is clearer than the latter, and does not tend to clog the sand with slime.

When a vat is full, the charge is allowed to drain, after

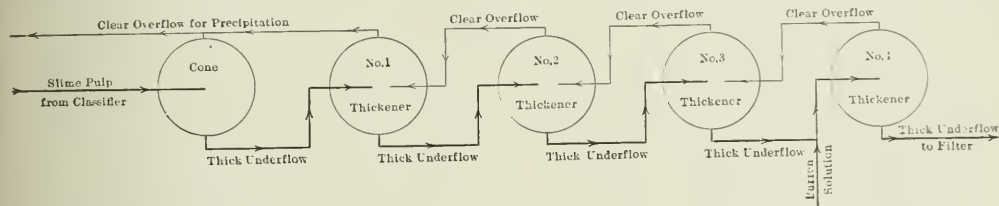


FIG. 4.—DIAGRAMMATIC REPRESENTATION OF CONTINUOUS DECANTATION TREATMENT FOR SLIME AT NEW RELIANCE MILL

which it receives successive washes of barren solution and water. The percolating solutions from both of these washes flow to the mill solution sump.

Continuous Decantation for Slime

The slime product from the Dorr classifier is subjected to continuous decantation and thickening, with counter-current washing, as outlined diagrammatically in Fig. 4. The thin pulp flows first to a 20-ft. cone which was part of the old mill equipment, and then successively through four Dorr thickeners in which the precious metal is dissolved, leaving a nearly barren pulp for final dewatering in a Portland filter. In order thoroughly to mix the pulp and solution flowing into any thickener, the launder in which these products are combined is provided with tapered or wedge-shaped baffles extending obliquely across the launder alternately to the right and left with a clearance of one inch between the lower end of the baffle and the side of the launder. Thus the pulp passing the lower end of any baffle is obstructed by the higher end of the next, which also changes the direction of the flow. The effect is to give every opportunity for the solution and solids to become thoroughly mixed before entering the thickener.

The feed to No. 1 thickener contains 80 per cent moisture, a ratio of solids to solution of 1:4. The underflow from No. 4 thickener carries about 55 per cent moisture, a ratio of 1:1.2. Washed slime pulp entering No. 1 thickener assays from \$1 to \$1.50 per ton, and the washed pulp discharged from No. 4 thickener runs from 40 to 80 cents per ton. These figures show wide variation, as also do the assays of the solutions overflowing the different thickeners, but they probably will be held within narrower limits when the new pumping system is installed, giving more perfect control of the flow of pulp.

The thickened underflow from No. 4 thickener, containing about 55 per cent moisture, passes to a Portland filter. In order to prevent settlement of solids in the filter tank, and to maintain the pulp at uniform consistency, the slime is circulated from the bottom of the tank through a diaphragm pump which discharges back to the tank. This action is supplemented once a day by inserting a compressed-air pipe into the filter tank, and scouring the bottom with a jet of air.

Steam Used to Clean Filter Cloth

An innovation in cleaning the filter cloth has been devised by Mr. F. C. Bowman, manager for the New Reliance company, consisting in the occasional use of steam to disintegrate the limey incrustation which clogs the filter pores. The steam, under about 10 lb. pressure, is introduced through the pipe ordinarily used for compressed air to discharge the cake, and that section of the filter through which the steam passes is lightly scrubbed with stiff brushes. By this method the lime is readily disintegrated, and thoroughly removed by a subsequent wash with hot water.

Before steam cleaning was adopted at the New Reliance, the customary acid treatment was used every two weeks; but at the present time, with even higher protective alkalinity in the solution than formerly, and with steam cleaning at intervals of two weeks, it has been necessary to resort to acid treatment only once in sixty days.

In the acid treatment a hot solution of 1 part commercial

hydrochloric acid and 5 parts water is poured over the filter from the top, while the machine is in motion and the vacuum on. The hot solution acts very quickly, being carried into the fabric and ultimately discharged by the vacuum pump. Only very slight scrubbing is required in places. By making suitable arrangement for applying the acid, the filter can be cleaned in twenty minutes.

A good quality of cotton cloth is used for the filter fabric, and the material is specially selected for freedom from flaws and irregularities in weave. The slight additional cost for specially selected material is warranted by the increased life of the cloth and better filtration. The cloth is placed on the drum with the finished side down, leaving the "wrong" or unfinished side of the cloth exposed. By this arrangement the slight nap or pile of the cloth produces a valve action in the meshes, being drawn inward when the vacuum is on, and forced out again when pressure is applied, leaving the pores clean and free.

As the dissolution of the precious metals is practically effected in the continuous decantation system, the chief function of the filter is to dewater the slime and discharge it to the dump. The average value of the whole pulp sent to the filter is from \$1.40 to \$1.60 per ton. The loss in dissolved gold is from \$0.05 to \$0.12, and the mechanical loss of cyanide is $\frac{1}{2}$ lb. per ton of dry slime. Vacuum is maintained at from 14 to 10 in., and the air for discharge of the cake is under a pressure of 10 lb. The slime cake has an average thickness 5 to 6 in., with extreme limits of $\frac{1}{4}$ to $\frac{1}{2}$ in. just before and after cleaning the cloth. The only wash applied to the cake is hot water coming from the jackets of the air compressor and flowing onto the descending side of the filter. The moisture in the filter cake will average 32 per cent, no attempt being made to discharge a dry cake, as about 40 per

cent moisture is required to carry the slime away from the mill. The filter is 12 ft. diameter with a 7.5 ft. face, and handles 65 tons of slime per day without reaching its full capacity.

Solution and Pulp Assays

The data given herewith refer to the month of May, 1913. The mill solution averaged 1.1 lb. KCN, with protective alkalinity at 1.5 lb. CaO. Higher protective alkalinity has occasionally been found desirable to reduce consumption of cyanide, and has been carried as high as 2.2 lb. CaO. Consumption of chemicals in pounds per ton of ore treated: KCN, 0.385; CaO, 6.25; Zn, 0.305. The value of the gold solution varies from \$1.20 to \$1.80 per ton; the zinc box tailing contains between 1 and 6 cents per ton. About 134 tons of solution is precipitated per ton of ore. The total cost of milling was \$1.234 per ton. This includes all labor, power, repairs, renewals, insurance, taxes, and general expense at the mill office.

The extraction as shown by bullion recovery is between 85 and 87 per cent. The sand tailing has an assay value of \$0.75 per ton, and the slime from \$0.55 to \$0.60. The difference between the two values is smaller than that observed at some of the other mills, and both are lower than some quoted elsewhere in these notes.

There are no unusual features in the methods of precipitation or clean-up. The gold solution is measured ahead of precipitation by means of an oscillating-box meter with counting attachment.

Power Data

The following tabulation gives the power consumption on the different groups of machines:

1 5K Gates gyratory crusher.....	} No load, 8 kw. Av. load, 11 kw. Extreme, 37 kw.
1 50-ft. elevator, 14 in. by 6 in. buckets	
1 16-in. belt conveyor, 22 ft. centers...	
Each 10-stamp battery.....	20 kw.
Dorr Classifier	0.36 kw.
1 3½ x 8 Deming triplex pump.....	} 10.25 kw. Each thickener requires about 0.075 hp.
1 5 x 6 Gould triplex pump.....	
1 4 x 6 Gould triplex pump.....	
1 7 x 6 Gould vacuum pump.....	
1 Portland filter.....	
4 Dorr thickeners, 20 ft. diameter.....	
3 line shafts.....	

For the data given in these notes, and for courtesies extended while visiting the mills, I am indebted to Messrs. F. B. Hitchings and D. A. Elliott, of the Bismarck Consolidated Company, and Mr. F. C. Bowman, of the New Reliance.

A new alloy, consisting of manganese, titanium and silicon, has been patented by F. M. Becket, of Niagara Falls, N. Y. It may also contain carbon in proportions varying from a fraction of one per cent to sufficient quantity to furnish all or part of the carbon requisite in the recarburization of steel. The manganese and titanium together are preferably in excess of fifty per cent of the alloy. When added to steel the component metals of the alloy are said to diffuse through the molten mass more uniformly and rapidly than do the corresponding ferro-alloys.

Utah Metal Production.—The total value of gold, silver, copper, lead and zinc produced in Utah in 1912 was \$4,922,302 according to advance figures by the U. S. Geological Survey. There was a decrease of 10 per cent in the value of gold produced, compared with 1911; an increase of 10 per cent in quantity of silver; a decrease in copper; an increase in lead, and about the same production of zinc. The total production showed an increase in value of about \$6,000,000.

A new soft-solder recently patented by John T. Dwyer, of St. Louis, Mo., is said to possess superior properties, and to be made at less cost than ordinary lead-tin solder. It contains approximately 41½ per cent tin, 0.02 per cent phosphorus, 2 per cent antimony, and the balance lead.

The Ostwald Process for Making Nitric Acid from Ammonia

Its Proposed Combination with the Manufacture of Calcium Cyanamide

The Ostwald process for making nitric acid from ammonia, which attracted considerable attention some ten years ago, has recently entered into a new stage of its commercial development in connection with the formation of the Nitrogen Products and Carbide Company, Ltd. This new British company issued in May a prospectus calling for a capital of \$10,000,000. Among its directors are Mr. Albert Vickers, Chairman of Vickers, Ltd., and Sir Richard Awdry of the Nobel Dynamite Trust Company, Ltd.

Among the products of this new company will be calcium carbide and calcium cyanamide, and the latter is to be employed to yield the ammonia needed as raw material for the manufacture of nitric acid by the Ostwald process. Use is here made of a well-known reaction by which ammonia is produced by heating cyanamide in presence of water $\text{CaCN}_2 + 3\text{H}_2\text{O} = \text{CaCO}_3 + 2\text{NH}_3$. (See, for instance, Erlwein, this journal, March, 1907, vol. V, p. 79). This reaction has been employed in some European works for the production of ammonium sulphate from cyanamide. The present proposal to work up the ammonia into nitric acid by the Ostwald process is novel.

The Ostwald process for making nitric acid from crude ammonia liquor is the subject of a long article in the *Iron and Coal Trades Review* (London) of May 23, 1913. This is reproduced in the following almost in full.

The fact that ammonia could be oxidized to nitric acid in the presence of atmospheric oxygen by the catalytic action of platinum was discovered by Kuhlmann about 1830. He was engaged in industrial work and, therefore, tried to apply this reaction in the commercial manufacture of nitric acid, but the time was an unfortunate one.

The great beds of sodium nitrate, on which we still depend for our nitric acid, were just being developed, and the price of nitric acid was falling rapidly in consequence.

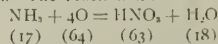
Besides this, there was no great production of ammonia or nitrogenous compounds, such as exists at the present time; by-product coke was not yet known, and there was no source of cheap ammonia.

The facts discovered by Kuhlmann, therefore, remained without commercial significance until within the last ten years, though the fundamental reaction, viz., the oxidation of ammonia to nitric acid in the presence of platinum, has been shown to every student of chemistry.

In 1900 Professor Ostwald and his assistant, Dr. Brauer, began to work on this same old reaction, having in view the application to a commercial process. At the end of about three years of investigation in the laboratory on a comparatively small scale, the process was considered ready for a test on a small commercial scale, and a year or so later the first commercial plant—a small one—was erected. This was studied with a view to the elimination of all possible sources of trouble and much time and labor have been spent in finding proper materials for the various parts of the apparatus, and in working out a completely automatic process.

About six years ago a full sized plant was begun, the results on the commercial experiment plant being by this time so satisfactory as to warrant the step. This large plant converted about 25 tons of ammonia gas per month into about 150 tons of rather dilute (36 deg. Baume) commercial nitric acid.

The chemical reaction is a very simple one. Ammonia gas mixed with air is passed over a plug of platinum, so arranged as to expose a large surface of contact to the gases which are forced through it. The reaction is:



For every seventeen parts by weight of ammonia used, 63 parts by weight of absolute nitric acid is formed.

Not long after the beginning of the experimental investiga-

tion of the reaction, an important fact was discovered. If the mixture of ammonia and air was allowed to pass slowly through the platinum contact, the yield of nitric acid was very small: only a few per cent of the theoretical yield was obtained.

If the stream of gas was forced quickly through the contact the yield was very nearly the theoretical value. This is to be explained by the fact that nitric acid is not the final product of the oxidation of ammonia, but only one of the intermediate ones. The final product is nitrogen gas and water.

So in the commercial plant the stream of ammonia mixed

"The form of catalyser to comply with these conditions may vary. If platinum is employed generally a length of one to two centimeters of platinum, over which the gas mixture streams with a velocity of one to five meters per second, is found to be sufficient; when the flow velocity is less, a shorter layer of platinum may be used."

In Patent No. 8,300 of 1902, Dr. Ostwald says:

"The mixture of ammonia and oxygen (air) should contain a large amount of oxygen with reference to the quantity of ammonia, it being necessary that a quantity of oxygen should be present corresponding at least to that represented by the following equation:



"It is, however, advisable to use a larger quantity of oxygen, preferably in the form of air. A further condition for obtaining a practical result is that the temperature employed in the process should exceed 300 deg. C. It is preferable to maintain the temperature between dark and bright red heat. A further condition is that the products of the reaction should be exposed for as short a time as possible to the reaction, as otherwise rapid decomposition of the products would result. For this reason I employ catalysers as short as possible and cause the gases to pass over them at a high velocity. I have found it advisable to so arrange the length of the catalysers and the velocity of the gases in such a manner that the contact of the latter with the former should not exceed one-hundredth of a second.

"As the velocity of the gases should be as high as possible, and also the temperature suitable for the oxidation should be maintained, it is necessary to maintain the velocity of the gases constant. This shows, however, practical difficulties. By a reduction of the velocity of the gases the catalysers would have a low temperature, as the heat of reaction developed in a unity of time would be smaller. By raising the velocity of the gases excessive heating of the catalysers would be caused. For the purpose of avoiding the aforesaid difficulties, I heat the gases to be treated by means of the hot products of the reaction before they come into contact with the catalysers."

The plant required to carry out this process on a commercial scale in connection with a source of ammonia consists of three parts: (1) Apparatus for producing ammonia gas. (2) Apparatus for the actual reaction, or catalysers. (3) Condensing plant.

Fig. 1 shows a general plan of such a plant and indicates the relative space occupied by the various parts. It is adapted to

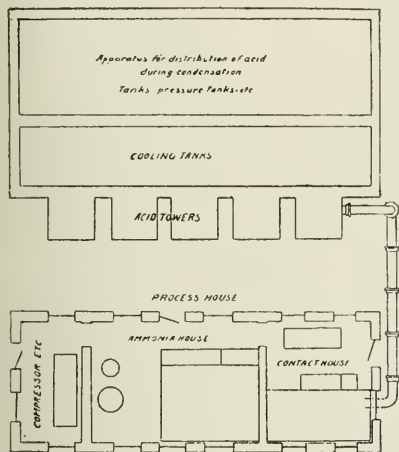


FIG. 1—GENERAL PLAN OF PLANT

with air is sent through the platinum contact plugs at a high rate of speed, and the result is a very high percentage of the theoretical yield, and a high efficiency of the apparatus used.

Dr. Ostwald in his patent No. 698, of 1902, describes his invention as follows:

"It is well known that nitric acid can be obtained from ammonia through the oxygen of the air by means of the catalytic action of spongy platinum or platinum black, although up to the present no process is known for thoroughly effecting this change in a useful technical matter. I have ascertained that under the influence of compact platinum alone, or platinum that is partly covered with spongy platinum or platinum black, a mixture of ammonia with an excess of air can be oxidized to nitric acid or to a higher oxide of nitrogen.

"Besides the oxidation of the ammonia to form nitric acid or higher oxides of nitrogen, another reaction usually takes place which leads to the formation of free nitrogen. In order, therefore, to attain a technically useful process the operation must be so conducted that the first reaction is thorough and practically complete, whilst the second should be as small as possible.

"This result is attained by using, for example, smooth or solid platinum which is either employed direct in this state or is first either partly or entirely coated with a layer of spongy or of black platinum. When now a mixture of ammonia with ten or more times the volume of atmospheric air is directed over it, a fairly high velocity being maintained, and simultaneously the temperature brought to red-glow heat and kept at the same, the smooth platinum causes the ammonia to be burnt to nitric acid, the second reaction which produces free nitrogen being practically unnoticeable.

"The finely divided platinum on the other hand accelerates both reactions, the second one more than the first. By moderate use of the finely divided platinum (platinum black or platinum sponge) with the smooth platinum the operation can be so performed that the reaction takes place rapidly but without any great formation of free nitrogen.

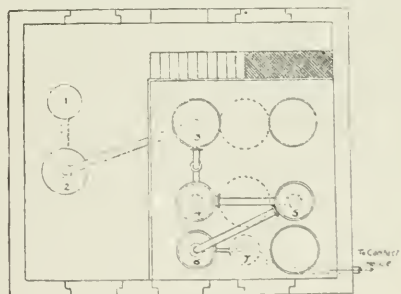


FIG. 2—PLAN OF AMMONIA HOUSE

the use of ammonia liquor. In the case of using cyanamide small modifications in the ammonia plant would be necessary. The ammonia plant takes in crude gas liquor and sends out nearly pure, dry ammonia gas, freed from carbon dioxide and hydrogen sulphide. The gas is not freed from traces of organic materials which may come over with it, for these impurities have been shown to exert no harmful influence on the process.

The ammonia plant for using gas liquor is a standard one, similar to those in use generally. This apparatus is indicated in

Fig. 2. The crude gas liquor is received in the heating apparatus numbered 1 and 2, and to assist in driving out the ammonia gas, heat and a current of air are both used. The gas so drawn off is purified in 3, 4, 5 and 6, by washing and treating with milk of lime. The current of air is supplied by a small compressor.

The air so supplied furnishes a portion of the oxygen necessary for the reaction, and its amount is regulated by automatic differential manometer and regulating valve. The rate

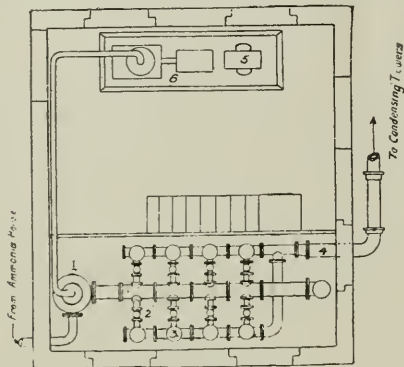


FIG. 3—PLAN OF CONTACT HOUSE

at which the crude gas liquor and the steam are supplied is regulated by hand. The production of ammonia gas by the above apparatus must be perfectly continuous to be efficient.

Mixed with a small amount of air the ammonia gas then passes into the contact house or catalyser house. Here it is mixed with more air, supplied with a fan, and the mixture is regulated by the automatic differential manometer and regulator 1 of Fig. 3. The gas mixture then passes through the platinum contacts 3, by way of the cocks 2, so arranged that any contact can be cut out of service without disturbing the rest. Here the real reaction takes place. Up to this point only ammonia gas and air have been present, and either cast or wrought-iron piping can be used with perfect safety. In the reaction apparatus itself nitric acid vapors, at a high temperature are to be handled. As soon as liquid nitric acid begins to form, stoneware is the only available material.

The hot gases leave the reaction plant through the pipe 4 and pass to the condensing plant, which is shown in Fig. 4. The gases pass into the condensing towers at 1 and meet with a large cooling surface, which is wet by nitric acid trickling continually through the stone filling of the tower. Water is not used because as strong an acid as possible is desired and enough water passes along as the result of the reaction itself.

The condensation serves a double purpose. The nitric acid formed during the reaction, and the water formed, are condensed together to liquid nitric acid. Beside this, any other of the higher oxides of nitrogen which may have been produced are given time to oxidize further, to meet with water and to change to nitric acid. The nitric acid which has been allowed to flow down through the towers is, therefore, used over and over again, becoming stronger in the process. This acid is cooled in the cooling apparatus 3, before sending it back to the tower, and stored in the storage tanks 4 during the interim.

From 4 it flows into the pressure apparatus 5, and from here it is forced by air pressure back to the top of the tower. Three coolers and one storage tank are used in supplying cooled acid to each tower.

The plant shown in Fig. 1 has five towers, each about 60 ft. high, and all acid is transported through stoneware. Whenever acid is to be lifted this is accomplished by air pressure.

From the opening of the last tower there escape nitrogen, oxygen, and a small amount of uncondensed acid, which can be further treated if desired.

It will be noticed that the condensing plant is an important part of the process. The gases from the reaction chamber pass through the five towers, one after the other, and this large condensing surface is necessary to prevent loss. The acid is tapped off at the foot of the second tower, and here it has a density of about 36° Be. It is a pure, not very strong, commercial grade of nitric acid.

Ammonia is the only raw material used in large quantity. A small amount of lime is also needed in the driving off and purifying of the ammonia gas. The product, nitric acid (36° Be.), is a current market chemical. The demand for this strength is, however, not very great, and a more concentrated acid, such as is used for making explosives, is more in request. The average price of ammonia in crude gas liquor, taken all over England, is not far from 10 cents per lb.

Estimate of Profits

The following is an estimate of the profits of a plant for the production of nitric acid from ammonia from the Ostwald process. As a basis it is assumed that each catalyser produces 200 kilograms of nitric acid (36° Be.) in 24 hours, and a plant will be taken employing monthly 25 tons of nitrogen (N) equivalent to 830 kilograms of ammonia (NH_3) per day—to produce nitric acid.

These 25 tons of nitrogen, or about 115 tons of sulphate of ammonia, can easily be obtained from 64 by-product coke ovens. In a gas works we must consider for this purpose a production of about 120,000 cubic meters (4,237,990 cubic feet) in 24 hours.

Out of these 25 tons of nitrogen per month—which corresponds with 83 cubic meters of 1 per cent ammoniacal liquor in 24 hours—we should be able to get, by means of the Ostwald process (taking as a basis a yield of only 85 per cent), the following quantities of different concentrations of acid:—Absolute nitric acid 78.7 tons per month; 93 per cent acid, or 48 deg. Be., 85.6 tons per month; 53 per cent acid, or 36° Be., 148.4

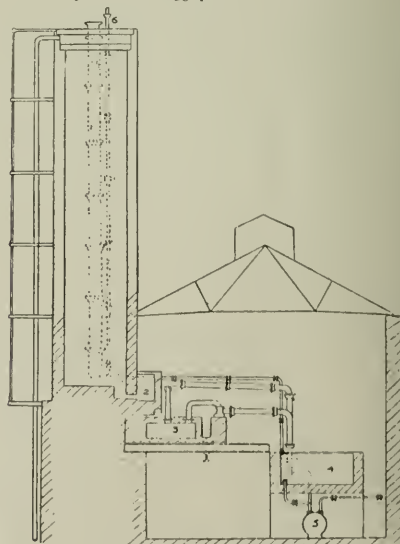


FIG. 4—CONDENSING PLANT

tons per month. The last named product is produced directly by the process; the 93 per cent acid is obtained from the 53 per cent acid by concentration; the "absolute" acid is not steady at all.

For the manufacture of nitrate of ammonia this quantity of nitric acid—say 148.4 tons of 53 per cent—can be used by adding, say, 21 tons of ammonia per month; therefore, from a total quantity of 26 tons of ammonia per month we should

get 99 tons of nitrate of ammonia (allowing 2% as waste) as monthly yield.

The cost of a plant producing 148.4 tons of 53 per cent acid per month by means of the conversion of 25 tons of ammoniacal nitrogen into 53 per cent nitric acid is estimated as follows:

Plant for generating ammonia from gas liquor.....	\$7,500
Plant for conversion (catalysers)	5,000
Condensing plant	17,500
Foundations for towers and iron work.....	5,000
Acid storage tanks	750
Air compressor and motor.....	3,000
Platinum (initial expense).....	2,500
Buildings and foundations.....	10,000
Unforeseen outlays.....	3,750
Total.....	\$55,000

Running Expenses

(1) *Steam*.—The same quantity will be required as that to volatilize ammonia out of diluted liquor for other purposes—for instance, for the manufacture of sulphate of ammonia—that is, 20 to 25 kilograms of steam per 100 kilograms of 1 per cent ammonia liquor. As 83 cubic meters of ammonia liquor are used in 24 hours, 20,000 kilograms of steam may be allowed. Taking 100 kilograms of coal at the rate of 46 cents there would be a daily cost of \$12.50, or monthly, say, \$337.50.

(2) *Cooling water*.—A consumption of 200 cu. meters per day may be allowed; this, estimated at something more than a farthing per cubic meter, amounts to \$1.50 per day, or \$45 per month.

(3) *Power*.—To lift the liquid in the condensing house, and for direct air pressure, 12 to 15 h. p. are required and for the running of a fan a further 5 h. p.—that is, 20 h. p. altogether. This means per month 14,400 h. p. hours, and at the rate of 3 cents this amounts to about \$437.50.

(4) *Lime*.—In employing a carbonic acid absorber, the consumption is something more than 1 kilogram of lime (CaO) per kilogram of ammonia (NH₃). In consequence, something like 30 tons of lime are used per month. Figuring the ton at \$3.75 this represents \$112.50.

(5) *Platinum*.—For the whole plant 30 contact elements are required, having a weight of 50 grams, which for a daily production of somewhat more than 100 kilograms of nitric acid corresponds with 200 kilograms of 53 per cent acid per contact element. The wear and tear is not very considerable; it may be estimated at 1.5 grams per day or 45 grams in the month. The contact elements may be used for a month or six weeks, after which time they ought to be scrapped as old metal and replaced. For this exchange it is only necessary to estimate the difference in price between the old and the new metal, which represents about 6 cents per gram. Assuming a monthly exchange, the outlay for all the contact elements requires about \$95. The total expense for platinum amounts to \$140 per month.

(6) *Wages*.—For superintending and reversing of valves and cocks two men are sufficient per shift; for cleaning, a third man is required—altogether six men per 24 hours. Superintendence should not absorb more than \$100 per month.

The cost of manufacture per month, including repairs and superintendence, would comprise accordingly:—

Steam	\$337.50
Cooling water	45.00
Power	437.50
Lime	112.50
Platinum	140.00
Wages	202.50
Repairs	125.00
Superintendence	100.00
Interest and depreciation (15% on \$55,000) per month	687.50
Total.....	\$2,187.50

The cost of ammonia in the form of dilute gas liquor is

calculated at the rate of \$17.50 per 100 kilograms. The total quantity required amounts to \$4,375 for the 25 tons needed for this size plant. The yield represents 148.4 tons of 53 per cent nitric acid. Reckoning the 100 kilos at \$9, the receipts amount to \$8,902.50.

From these figures it would seem that the operation of a plant for converting monthly 25 tons of ammoniacal nitrogen from the ordinary liquor of gas works, etc., into 53 per cent nitric acid gives the following results:

	Total per month	Per 100 kilos of 53% acid
25 tons of ammonia.....	\$4,375.00	\$2.95
Cost of conversion.....	2,187.50	1.46
Cost of production.....	\$6,562.50	\$4.41
Sales of 14,840 kilograms of nitric acid at \$9.	8,902.50	6.00
Net profit per month.....	\$2,340.00	\$1.59

The above figures of estimated cost, reproduced from the (London) Iron and Coal Trades Review, are of course, based on British conditions. For American conditions various of the figures would have to be changed. The figures are given here as they may serve as a basis of a modified calculation.

The Manufacture of Ammonium Nitrate

Ammonium nitrate is a valuable chemical and there is a large demand for it for safety explosives of all kinds. The market price is fixed by the value of ammonia and that of Chile saltpetre.

The Ostwald process is remarkably well adapted to the manufacture of ammonium nitrate. The nitrate acid is neutralized with a further supply of ammonia, and the solution of ammonium nitrate so obtained is evaporated until crystallization occurs. The neutralization is carried out in wooden tanks and the liquid is then evaporated in vacuum pans almost to dryness. Pure ammonium nitrate separates, and this is dried in centrifugals and packed for shipment. The impure salt left in the mother liquid is dried out and sold as second grade.

There must also be erected a plant to obtain pure ammonia out of the diluted liquor, and an installation for the production of the solid salt. For this purpose different combinations are possible, according to the mode of working, viz., to get the nitrate of ammonia direct from the gas or from diluted liquors and again either refining the salt produced before or after its formation.

In consequence the cost of an installation will vary between \$25,000 and \$35,000—buildings included. This for a production of 99 tons of nitrate of ammonia per month, and in which case we have to use altogether 46 tons of ammonia; out of which—as stated above—25 tons are for nitric acid, and the remaining 21 tons for pure ammonia.

Taking the price of nitrate for explosives at \$17 per 100 kilograms, and reckoning for the purification of 21,000 kilograms of ammonia \$1.050 and further for the preparing of 99 tons of salt another \$900 as wages, etc., the profits of this plant are established as follows, (the figures again referring to conditions in Great Britain):

	Total per month	Per 100 kilo- grams of salt
Value of ammonia, 46,000 kilograms at 17.5 cents.....	\$8,050.00	\$8.13
Of which 25,000 kilograms converted into acid—wages and deprecia- tion	2,187.50	2.21
Purification of the remaining 21,000 kilograms of ammonia.....	1,050.00	1.07
Wages for manufacture of salt.....	900.00	1.00
Cost of production for 99 tons of nitrate of ammonia.....	\$12,277.50	\$12.41
Receipts from 99,000 kilos at \$15....	16,830.00	17.00
Net profit per month.....	\$4,552.50	\$4.59

It may be estimated that the normal production of nitric acid in the United Kingdom is approximately 25,000 tons per annum, reckoning the acid as being 80 Tw. In ordinary years the amount used up on the spot will be about 20,000 tons, rising to 30,000 tons when the explosives trade is brisk. The miscellaneous trade will account for 5000 to 7500 tons according to requirements.

Action of the Salts in Alkali Water and Sea Water on Cements

An extended investigation on the subject is described in Technological Paper No. 12 of the Bureau of Standards, Washington, D. C., the authors being P. H. Bates, Chemist; A. J. Phillips, Assistant Chemist; and Rudolph J. Wig, Associate Engineer Physicist of the Bureau of Standards. The bulletin comprises 158 pages with many tables and diagrams and only a summary of the object of the investigation and the results can be given here.

The disintegration of cement structures, when placed in contact with sea water, is a phenomenon, which has attracted the attention of cement manufacturers and cement users almost from the first time that such material was used for marine construction. There are cement structures which have withstood the action of sea water for years and probably will continue to do so, yet there are structures which have failed; and it is also possible in the laboratory by artificial solutions to destroy almost completely a briquet, or cube, or cylinder made of cement mortars or concrete.

The cause of this disintegration is not certain, though it is almost universally believed that it is the reaction of sulphate of magnesia of the sea water with the lime and the cement (formed during the setting) and the alumina of the aluminates of the cement, resulting in the formation of hydrated magnesia and calcium sulpho-aluminates, which crystallizes with a large number of molecules of water.

The other constituents both of the sea water and the cement are usually considered of little effect, though lately attention is being drawn to the fact that both sodium chloride and magnesium chloride rapidly attack the silicates.

Concrete structures have been made (especially in this country) which are resisting the attack of sea water to a marked degree. It is, therefore, not surprising that many engineers attribute the disintegration when it does occur to poor workmanship or materials, or to the abrasion of the sand or floating bodies in the water, and to the mechanical action of waves and frost action (as the disintegration usually takes place at or near the water line) rather than to any chemical reaction or crystallization due to such reaction.

The investigation of Messrs. Bates, Phillips and Wig were planned for the purpose of determining the suitability and permanency of various cements in structures exposed to the chemical and mechanical action of sea water and alkali salts and, if possible, the cause of failure or disintegration of cements and concretes.

The study of the subject was begun in such a manner as to determine, if possible, just what reaction would take place when the salts, commonly present in sea water and alkaline soils, were allowed to act on cement and cement mortars. In order that this study should be complete, information should be obtained not only as to what salts present in any solution might cause destruction, but also in what manner this destruction is being accomplished.

Both chemical and physical investigations were made in the laboratory, and field tests in sea water were made at Atlantic City, N. J. In both series of laboratory tests there have been used at various times, in addition to sea water from Atlantic City, solutions of sodium chloride, sodium sulphate, sodium carbonate, magnesium chloride, magnesium sulphate, ferrous sulphate, and also solutions in which there were present in equal parts by weight two salts as sodium chloride-sodium sulphate, sodium chloride-magnesium chloride, sodium chloride-magnesium sulphate, sodium sulphate-magnesium sulphate, so-

dium chloride-sodium carbonate, sodium chloride-calcium chloride, sodium sulphate-sodium carbonate, magnesium chloride-magnesium sulphate.

It will be noted that a solution of calcium sulphate was not used. This salt is comparatively insoluble, and a series of tests using it, even in a saturated solution, would hardly be comparable with the series in which the above were used and in which the solutions contained but 2 per cent by weight of the anhydride salts. Moreover, when this solution contained the sulphuric anhydride radical, calcium sulphate would be formed in the cement.

The conclusions of the authors are limited by the scope of the investigation and since the physical tests reported cover a period of exposure not exceeding 3½ years the conclusions should be considered, as the authors point out, as somewhat tentative. They are as follows:

1. Portland cement mortar or concrete, if porous, can be disintegrated by the mechanical forces exerted by the crystallization of almost any salt in its pores, if a sufficient amount of it is permitted to accumulate and a rapid formation of crystals is brought about by drying; and as larger crystals are formed by slow crystallization, there would be obtained the same results on a larger scale, but in greater time if slow drying were had. Porous stone, brick, and other structural materials are disintegrated in the same manner. Therefore in alkali regions where a concentration of salts is possible, a dense non-porous surface is essential.

2. While in the laboratory a hydraulic cement is readily decomposed if intimately exposed to the chemical action of various sulphate and chloride solutions, field inspection indicates that in service these reactions are much retarded if not entirely suspended in most cases, due probably to the carbonization of the lime of the cement near the surface or the formation of an impervious skin or protective coating by saline deposits.

3. Properly made Portland cement concrete, when totally immersed, is apparently not subject to decomposition by the chemical action of sea water.

4. While these tests indicated that Portland-cement concrete exposed between tides resisted chemical decomposition as satisfactorily as the totally immersed concrete, it is felt that actual service conditions were not reproduced, and therefore further investigation is desirable.¹

5. It is not yet possible to state whether the resistance of cements to chemical disintegration by sea water is due to the superficial formation of an impervious skin or coating, which is subsequently assisted by the deposition of shells and moss forming a protective coating, or by the chemical reaction of the sea salts with the cement forming a more stable compound without disintegration of the concrete, or by a combination of both of these phenomena.

6. Marine construction, in so far as the concrete placed below the surface of the water is concerned, would appear to be a problem of method rather than materials, as the concrete sets and permanently hardens as satisfactorily in sea water as in fresh water or in the atmosphere, if it can be placed in the forms without undue exposure to the sea water while being deposited.

7. Natural, slag, and other special cements tested in concrete mixtures showed normal increase in strength with age both in sea water and in fresh water.

8. In the form of neat briquettes most of the Portland cements of high iron cement, several of the cements of high or normal alumina content and one special slag cement did not show any marked difference in tensile strength whether ex-

¹In service the concrete extends from the sea bottom to a point above high tide, where the wall or pile would always be exposed to the atmosphere. With this condition the sea water could be drawn up the wall by capillarity, the moisture evaporating and leaving the salts, which would become concentrated, and thus possibly cause disintegration especially if mixture is porous. An additional series of tests is now being made in which short piles, 7 feet in length are being placed in sea water so that 2 feet of the center portion will be exposed to the atmosphere. After various periods of exposure the piles will be sawed and the various sections tested for elastic properties and compressive strength.

posed to fresh or sea water for all periods up to two years. Other cements of various compositions showed signs of disintegration after a few weeks.

9. All cements resisted disintegration in sea water better in mortar mixtures than in the form of neat briquettes. In most cases the mortar briquettes had normal strength up to 2 years' exposure.

10. The physical qualities of the cement, which depend essentially upon the method of manufacture, would seem to determine its resistance to decomposition when brought into intimate contact with the sulphate and chloride solutions.

11. Contrary to the opinion of many, there is no apparent relation between the chemical composition of a cement and the rapidity with which it reacts with sea water when brought into intimate contact.

12. Tri-calcium-sulpho-aluminate could not be formed, and therefore disintegration could not result from this cause.

13. In the presence of sea water or similar sulphate-chloride solutions:

a. The most soluble element of the cement is the lime. If the lime of the cement is carbonated it is perfectly insoluble.

b. The quantity of alumina, iron, or silica present in the cement does not affect its solubility.

c. The magnesia present in the cement is practically inert.

d. The quantity of SO_2 present in the cement up to 1.75 per cent does not affect its solubility, but a variation in the quantity present may affect its stability by affecting its rate of hardening.

14. The change which takes place in sea water when brought into intimate contact with the cement is as follows:

a. The magnesia is precipitated from the sea water in direct proportion to the solubility of the lime of the cement.

b. The sulphates are the most active constituents of the sea water and are taken up by the cement. Their action is accelerated in the presence of chlorides. No definite sulphate compound was established.

c. The quantity of chlorine and sodium taken up by the cement is so small that no statement can be made as to the existence of any definite chloride or sodium compound formed with the cement.

15. The SO_2 added to a cement in the plaster to regulate the time of set is chemically fixed so that it will not go into solution when the cement is brought into intimate contact with distilled water.

16. Metal reinforcement is not subject to corrosion if embedded to a depth of 2 in. or more from the surface of well-made concrete.

The production of secondary metals in the United States is shown in the following table, compiled by the Geological Survey:

Metal	1912	
	Short Tons	Value
Secondary copper, including that in alloys other than brass	66,441	\$21,593,325
Remelted brass	101,523	27,279,516
Secondary lead	30,266	6,045,120
Recovered lead in alloys	3,902	
Secondary spelter	52,251	7,750,494
Recovered zinc in alloys other than brass	3,912	
Secondary tin	8,333	14,301,368
Recovered tin in alloys	7,068	
Secondary antimony	13	426,020
Recovered antimony in alloys	2,493	
Total value		\$77,395,843

A crisis in the German potash industry is reported, due to over-production. It appears that the funded securities of the leading companies of the Potash Syndicate, which have been selling at high premiums, have fallen below par, and a number of newly established companies have become insolvent. The law passed two years ago for putting an end to the concessions granted by certain companies to American importers accomplished its object, but has failed to limit the production of potash. A consular report states that it will take several years to equalize the supply and demand of potash, and that legislation is now in progress to regulate in future the production of potash.

Efficiency in Chemical Industries

Presidential Address Delivered Before the American Institute of Chemical Engineers

By T. B. Wagner

During the eighth International Congress of Applied Chemistry, held in New York in 1912, a number of general lectures were delivered, among them one by Dr. Carl Duisberg of Germany, entitled "The Latest Achievements and Problems of the Chemical Industry." With others, I was profoundly impressed with this recital of achievements—a splendid record, indeed, of the distinguished services rendered by the chemist and the chemical engineer. It is but fair that we give the latter full credit, for while the "pure" chemist takes the initiative by developing ingenious theories and syntheses, it is the chemical engineer who follows up the former's work and transform it into commercial success, by which the value to mankind of scientific effort and accomplishments is judged. Dr. Duisberg's essay surely brought to many of us the thrill of romance, but below it all lay a deeper meaning, for it spelled the "triumph of efficiency."

Would not one have to be void of all sentiment not to be stirred when learning of such work, and not to take pride in being a member of so useful and illustrious a profession as that of chemists and chemical engineers? We need not go abroad, however, to learn of the magnificent results accomplished by them, for we have abundant evidence in this country of the marvelous changes which their efficiency has brought about, with the result that this country occupies today a commanding position among nations in the field of applied chemistry.

"Efficiency in Chemical Industries," the subject which I have chosen for this occasion, leads one, as you have seen, into such vast fields of accomplishment and possibilities that I fear my fate tonight might be that of the parson who arose before his congregation and said: "My dear brethren, I shall divide my sermon this morning into three parts—firstly, I shall announce my text; secondly, I shall depart from it, and thirdly, I shall never get back to it."

To avoid such a calamity, the calamity of going astray, I shall speak today only on what efficiency has done for the industry with which I am identified—that of corn products. I trust I have walked in the highways and by ways of that particular domain long enough not to get lost.

A further reason why I have elected to speak on this subject is that experience tells me that we rather welcome an account of accomplishments in industries other than our own, for the precedent of another's difficulties encountered and overcome acts like a tonic to cheer us over the stumbling blocks in our own paths, but they are only stumbling blocks after we have passed them and left them well behind; when they loom before us, they seem to have the proportions of mountains.

The manufacture of corn products in this country is not an infant industry, and much has been written about it, its history and evolution. The first product manufactured from corn was starch, and the first manufacturer was Thomas Kingsford, his plant being located at Oswego, N. Y.

He commenced operations about 70 years ago, at a time when the raw material—Indian corn—was a low-priced commercial commodity, for in those years the population of this country was much smaller than today, home consumption was limited correspondingly, and a demand had not, as yet, been developed abroad. The raw material was cheap and the price of the finished product was high.

The margin of profit, therefore, was considerable, and an attempt at efficiency did not appear imperative, particularly not since competitors were "few and far between," and thus the principal stimulus for efficiency was lacking. The almost exclusive aim—a commendable one—was to produce starch of the highest quality. The matter of yield and of cost was of secondary consideration.

With good quality and rather handsome profits, why talk about economy or efficiency in the boiler house, in the engine room, in the transmission of power, or with respect to labor?—for labor too was cheap in those days and labor unions were unknown.

If the efficiency were to have been gauged, as it should, among other things, by the recovery of products from a given unit, that is to say, by the yield, it cannot be denied that a manufacturer could have taken but little pride in revealing his degree of efficiency.

This should not be taken as a reflection upon the industry at that time, for, to be fair, we must not judge conditions retrospectively. It was the status of the industry in those days, but that the future had many and far-reaching changes in store soon became evident.

The price of corn gradually rose, and, due to growing competition, the price of the finished product declined, narrowing the margin of profit. The manufacturer, therefore, had to begin to pay attention to the subject of yield and to an increase of revenue possibly by recovering some of those products which were allowed to run to waste.

First among these were the nitrogenous ingredients of corn, the so-called "gluten," but the method of recovery was very crude. It was collected while in a wet condition and its sale was in the nature of a local business, that is to say, it was sold to the farmers located in the neighborhood of the factory, and used by them in feeding cattle.

It was not known as a commercial commodity, yet the proceeds from its sale added materially to the reduction in the cost of starch. With an increased output, the local markets eventually proved too small, and it became necessary to recover the gluten in a condition which would permit of shipping it to distant points.

This led to the drying of the gluten, and while the recovery of this product in itself was a step in the direction of efficiency, the methods and appliances resorted to in drying were primitive. It did not take long, however, for efficiency to make its entrance; the apparatus first in use was capable of evaporating about 1000 pounds of water per hour; each unit of the apparatus employed today has an efficiency, expressed in terms of evaporation, equivalent to 9000 pounds of water per hour. The amount of dry gluten recovered in one of our large factories is 250,000 pounds per day.

The hull of the corn, the so-called "bran," underwent the same transformation, and was changed from a waste product, or offal, to a valuable staple product, the amount recovered per bushel being approximately the same as that of the gluten.

In the course of time, the manufacture of glucose was started, and the enterprise flourished. The introduction of this new product from corn furnished a splendid opportunity for the display of efficiency.

In a measure, the industry was an imported one, for glucose had been made from potatoes in Germany many years prior to the establishment of the first factory in this country.

In those early days it was necessary not only to import the machinery, but skilled labor as well. Among the most important were the men in charge of the vacuum pans. They were brought here at the expense of the manufacturer and engaged at extravagant salaries. They were quick to realize their advantage and soon became the "bosses" of the plant. They ruled absolutely. Their work was surrounded with great mystery, but it had to give way to efficiency, and today the position of a pan man is no more important than that of any other workman, in fact, unskilled men are selected for this work and soon become experts at their posts.

The problems which arose in the manufacture of glucose taxed the ingenuity and efficiency of the chemical engineer. To illustrate, I will cite the history of the Chicago Sugar Refining Company, which was organized by Chicago business men with a paid-in capital of \$1,250,000, a large amount of money even in present-day enterprises, but particularly so in 1880, when this factory was constructed.

Its purpose was the manufacture of anhydrous grape sugar

(pure dextrose), the production of which, on a commercial scale, had been made possible by Dr. Arno Behr. His method differed from all previous ones in that alcohol was not required in the process of crystallization.

The price of sugar was high in those days; I believe it sold at 8 cents a pound. Anhydrous dextrose, or grape sugar as it was called then, could be manufactured quite cheaply, as corn was still sold at a relatively low price.

The scheme was to blend the soft cane sugar from Louisiana with this anhydrous dextrose, and sell the product to the confectioners, for whose purposes it was suited most admirably. The factory started off a great success. The product turned out was of the highest purity and called forth the commendation of every one who tried it.

Thousands of barrels were sent out in the course of a few months, but in the course of another few months the shipments commenced to be returned to the factory, because it was found that the anhydrous sugar had absorbed the moisture contained in the soft sugars, with the result that the contents of the barrels had solidified into a hard mass.

It was impossible for the confectioner to use this product, and the company stood face to face with the failure of an enterprise into which, by this time, approximately \$1,500,000 had been invested. There was no alternative but to shut down the factory. The question now arose what to do next.

Many suggestions were submitted, among them very worthy ones, but they were discarded until the chemical engineer stepped into the breach and suggested that this anhydrous sugar plant be converted into a glucose factory. I shall not go into details, but simply recite the fact that after untold efforts and untold moments of anxiety, the new venture succeeded, and in the course of time it prospered. It may be properly said that it was principally responsible for the development of the American industry of corn products, as we know it to-day.

Even though it prospered, there were many problems. Every step in the manufacture of glucose requires the most careful attention, and chemical control is needed incessantly. The aim is first to obtain the raw material—starch—in as nearly chemically pure a state as possible; this means the elimination of all nitrogenous and fatty matter, a task which offers the greatest difficulties, because of the enormous scale on which the business is conducted, because of the condition of the raw material—whether fully matured or not, whether entirely sound or slightly deficient, even because of atmospheric conditions, which exercise a certain influence.

The ambition of the manufacturer is to produce water-white, crystal-clear glucose, which will retain its color in all seasons, in all climates and even if kept in transit or storage for a long period of time. Such quality depends upon a thousand and one factors, some of which it is very difficult to control. The refining of the glucose and the treatment of the refining agents are important problems, which, however, have been worked out most successfully as the result of applied efficiency on the part of the chemical engineer.

The recovery of the gluten and the bran was an important step forward, but one of the most valuable parts of the corn was still allowed to go to waste, viz., the germ, which yields the oil. There was no incentive to recover it, since no one seemed to want the oil. Had it been produced on a large scale, there would not have been a market for it, although it was realized, even in those early days, that a vegetable oil of the character of that obtained from corn must eventually become a valuable article of commerce; so it did, but it took many years of hard work to bring about this condition.

Shortly before I entered the business, less than 15 years ago, they had just started producing corn oil on a large scale, but had no market for it—the oil went begging; that was a serious condition. The Chicago factory produced at that time about 15,000 pounds of oil per day, and selling, as it did, for less than 15 cents a gallon, the revenue from this production was but \$250. Today the value of a gallon of corn oil is nearly 50 cents, and the amount of oil recovered from the

same number of bushels as in the Chicago factory is worth \$2,700.

This is not due altogether to the increased selling price, but also to the greater efficiency prevailing in the separation of the germ, and the extraction of the oil, resulting in a yield three times larger than in the days of the Chicago factory. Today the gross value of the oil recovered from one bushel of corn, together with that of the oil cake obtained as a by-product, is equivalent to 25% of the purchase price of the corn.

The ingenuity of the chemical engineer did not stop there. The price of glycerine was rising steadily, hence the proposition suggested itself of separating the glycerine from the fatty acids. What this meant is evidenced when I recite that glycerine today sells at 18 cents per pound and corn oil at about 6½ cents, and that 100 pounds of corn oil yield 12 pounds of glycerine; the price of crude oil and fatty acid is substantially the same.

To the list of by-products recovered, gluten, bran, oil cake, oil and derivatives such as glycerine and fatty acids, another was soon to be added. We still had a waste product, which, if recovered, seemed likely to be converted into a revenue producer of the first magnitude.

I am referring to the solids contained in the so-called "steep water," the water in which the corn is immersed and softened prior to grinding. It contains the most valuable ingredients of the grain, viz., the organic phosphorus compounds, magnesium and potash salts, nitrogenous bodies and sugars.

Many efforts have been made to recover these solids in dry form, but, owing to the hygroscopic condition of the residue, such methods were found to be impracticable.

In some localities serious consequences were the result. I have in mind the experience we had at one of our factories located in the State of Iowa, and grinding about 10,000 bushels of corn per day. The steep water was not recovered, but, together with the wash waters from starch and gluten, was run into a creek, which in turn discharged into a river of fair size.

This method of disposing of the steep water soon became a menace to the plant, for proceedings were instituted in the courts with a view to having the factory shut down and declared a public nuisance because the steep water contaminated the river, killing the fish, and contributing in many ways to the discomfort of the residents.

The owner of the factory being unable to satisfactorily dispose of this waste as a last resort constructed a pipe line to a farm located about three miles from the factory and entered into an agreement with the owner of the farm whereby he was permitted to conduct the steep water through the pipe line from the factory to the farm and discharge it there. For that privilege, besides furnishing a most excellent fertilizing material, he had to pay the farmer \$3,000 annually.

Today we recover this former waste by collecting it, concentrating it in vacuo and adding it to the gluten feed in the form of a syrup, with which it is subsequently dried—the feed acting as an absorbent. Instead of investing in pipe lines and paying for dumping rights, the waste of a 10,000 bushel plant was thereby converted into a revenue of almost \$100,000 per year. Applied to the industry as a whole, this former waste furnishes today an annual gross income of approximately \$1,500,000.

Very little is now running to waste. Whatever small loss there is, is caused by the soluble solids carried off in the water used in the washing of starch and gluten. However, these solutions of solubles being very dilute, it is doubtful whether an efficient method will ever be devised for their recovery at a profit. It will be a question of economical evaporation, therefore, largely an engineering problem. Nor will the recovery of these solids add materially to the efficiency record in point of yield.

Efficiency has manifested itself in our industry in many ways. As production increased and finally exceeded the de-

mand ways and means had to be found for diverting the surplus into new channels. This meant partly new products and partly new markets. The latter were secured by entering our starches into competition with products mostly foreign, such as rice, wheat, potato and cassava starch; and by studying closely the needs of the respective industries requiring starchy materials we have succeeded, in a large measure, in replacing rice, etc., with products of corn.

The development of new products occurred simultaneously, and to the series of bulk products were added, in the course of time, a large number of special products, such as thin boiling and modified starches, suitable for every conceivable technical purpose, dextrines and gums, special varieties of glucose, mixed table syrups, so-called "70" and "80" sugars, and refined, hydrogenated and vulcanized corn oils.

There is every prospect of further additions to the present number. Products of corn, in one form or another, are consumed today in almost every industry, and they reach every corner of the civilized world.

Figuratively speaking, this branch of efficiency was raised from 1 to 100—that is to say, where originally the industry was based upon the manufacture of only one product, to wit, corn starch, today the number of separate articles obtained from corn presents the formidable array of 100 products and more.

Hardly anything, however, contributes so much to the general efficiency of an industry as the individual plant capacity. This is the part of our business which requires the special attention and skill of the engineer.

It has been only ten years since we stopped operating plants the unit of which was 500 bushels of corn per day. These plants could not compete with modern conditions for the simple reason that they did not keep pace with the demands of efficiency. How can a plant grinding 500 bushels of corn a day enter into competition with a plant converting into commercial commodities 30,000 bushels or even 50,000 bushels of corn, which is the daily capacity of the latest addition to our chain of factories, viz., the mammoth plant at Argo, Illinois?

It is the concentration of operations that makes it possible for this plant to carry off highest honors in point of efficiency, and this concentration is carried through every department of the plant, each department being organized as a manufacturing unit of its own.

The boiler house is our steam factory. It is the business of the man in charge to manufacture and deliver steam at the lowest possible cost and if you could go through the boiler house you would be surprised, perhaps, to find to what extent a chemical control is observed, and to what extent it contributes towards the low cost of steam.

The efficiency of the boiler house depends, of course, not only upon its operation, but equally as much upon the equipment, and so the old tubular boilers had to give way to modern high-efficiency water tube boilers. Where formerly the standard unit was 100 h.p. it is now 600 h.p. per boiler. We now have only mechanical stokers in place of hand-fired furnaces. Great savings have been brought about in coal and ash-handling machinery, so that the operation of a boiler plant today requires but a fraction of the labor formerly employed.

Coincident with the increased efficiency of the boiler house was the rational, the scientific, use of steam. The evaporation in dryers and vacuum pans is done through exhaust rather than live steam, and one may well point with pride to the fact that in spite of the large amount of exhaust steam produced, our plants are operated without the discharge into the air of a single pound of unused exhaust steam.

The engine room is another such manufacturing unit, and a high degree of efficiency is required to produce and transmit motive power at a low cost. Here the slide valve engines had to yield to the Corliss type, and more latterly to the steam turbine. The costly transmission of power by belts, rope drives and transmission shafts had to give way to elec-

trical power. Efficiency is carried into the minutest detail, even apparently so small an item as lubricating oil did not escape the general policy of efficiency, and while such items do not represent an imposing saving per se, the aggregate foots up a substantial amount.

The milling department is another unit of its own, and so is the refinery, the feed house, the oil plant, the dry starch house, the dextrine department, etc. Each one of these departments is in charge of a chief, responsible for the output, its quality and cost. This general efficiency is carried further in that we operate our own box and carton factories, our printing shops, barrel factories and tin can plants, which latter have a record of efficiency second to none in this country.

You will have noted that we distinguish between two domains of efficiency: first, that of the manufacturing department, and secondly, that of the engineering department. In the latter may be included plant construction, for efficiency does not reign only on the inside, but makes itself felt on the outside as well. Improved plant construction leads to reduction of maintenance charges, to a lowering if not elimination of fire risks, accompanied by a corresponding lowering of insurance rates, to a better safeguarding of the employees, to a reduction of accidents and to an improved sanitary condition. The last named features in turn lead to increased efficiency of the workmen, without which any attempt at a high degree of general efficiency must come to naught.

Another link in the chain of general efficiency is furnished by labor-saving devices, the result of which is that less men are employed today in a factory grinding 30,000 bushels than were formerly required to operate a plant with a capacity of only 10,000 bushels.

Gentlemen, I have singled out the industry of corn products because it furnishes one of the most striking examples of what efficiency has done for the development of a chemical industry within a comparatively brief space of time. Unlike other industries, such as steel, or sugar, our industry is a conglomerate of a number of others. Per se, the manufacture of starch has nothing to do with that of oil; the manufacture of corn syrup nothing with the production of concentrated feeding stuffs; the manufacture of dextrines nothing with that of dextrose, etc., yet these various departments are so related to the whole as to form inseparable parts of it, and the efficiency of the whole depends, therefore, upon the efficiency of each branch.

The net result of our efforts with respect to efficiency is perhaps best illustrated by the fact that we pay today for our raw material—corn—three times as much as our predecessors, and yet we sell our products in the markets of the world at one-third of their price.

To continue and increase this efficiency, it is necessary that we should have young men with the proper training—not only with a scientific schooling, but such as have received a thorough training in engineering, who know the parts and functions of machinery and apparatus, who have a fair knowledge of factory operations, and who have a general understanding of the principles of manufacturing.

It is gratifying to note that the number of colleges and universities is increasing from year to year, where a course of chemical engineering is added to the curriculum of the chemical student.

The chemical industries of this country have grown from a small beginning into a position of great importance. A large share of the credit is due to the schools which furnished the men. They must give us men of ideas, men with initiative, men with mental and physical equipment, to carry through their creations of mind into creations of fact.

No matter how small a cog the individual may be in the big wheel of industry, he may well be proud of his share in the building up and development of the chemical industries which contribute materially toward the prosperity of this country. If his work should go unnoticed by the public at large, he will find comfort in the fact that "the

greatest joy of those who are steeped in work, and who have succeeded in finding new virtues and understanding the relations of things to each other, lies in work itself."

Such work cannot result other than in efficiency; efficiency stands for all that is great and potential. Efficiency is the making of a man, efficiency is the making of an industry, efficiency is the making of a nation.

The Use of Waste Heat Gases in the Paper and Pulp Field

One of the ideas which Philip B. Sadtler, of the Swenson Evaporator Company, Chicago, has included in the design of the apparatus of the Chesapeake Pulp & Paper Company, West Point, Va., is the use of waste heat gases from the rotary black ash furnaces to produce the generation of steam in the boiler, and then with this steam to carry out all of the evaporation of the black liquor in the Swenson multiple-effect evaporator. This idea of using waste heat from special furnaces to generate steam in boilers is not new, although it is a decided novelty in the pulp field, the engineers having been slow in the past to adopt this particular means of insuring economy. It has, however, been tried out and found successful.

It is interesting to note that the original source of the heat accomplishing the evaporation in the multiple-effect evaporators is woody organic matter in the black liquor itself. The strong liquor from the evaporator, which has had most of the water driven from it, is burned in the rotary incinerating furnace to an ash. This ash, containing considerable carbon, passes into a so-called melting furnace, operated under an air blast, and from which originate the hot gases to generate steam in the boiler.

By using a quadruple effect evaporator in this way, there is sufficient heat provided by this ash to accomplish the entire evaporation required. This question of multiple-effect evaporation is what makes a success; where Swedish engineers have failed because they attempted the same thing in a single-effect open-pan evaporation, that is, used this direct heat in connection with an open evaporator. In this case there was far from enough heat to produce the required results. It is evident that the type of evaporation is the key to the situation, so far as economy is concerned.

Ton of Coal Makes More Coke Than Formerly

The quantity of coal required to produce a ton of coke is much less than formerly. The average gain in 1912 compared with 10 years ago is probably at least 160 pounds. It is doubtful if in the earlier years the actual yield of coal in coke exceeded 60 per cent, whereas in 1912 it was 67 per cent, according to the United States Geological Survey. This gain is largely due to the increase in the production of by-product coke, in which the yield of coke from a ton of coal is very much higher than in making beehive coke.

In Illinois, Indiana, Massachusetts, Michigan, New Jersey, New York and Wisconsin, where coke is made exclusively in by-product plants, the yield varies from 60.6 per cent (in Wisconsin) to 81.8 per cent (in Indiana), whereas in the States where beehive practice prevails the yield in 1912 varied from 50 per cent (in Georgia) to 66.5 per cent (in Pennsylvania).

Potash Importations.—The importation of "potash salts" for consumption into the United States during 1912 amounted to 622,179,164 pounds, valued at \$10,602,285, according to W. C. Phalen, of the United States Geological Survey. This importation is only a part, however, of the potash salts entering the United States. To it should be added the importation of kainite and "manure salts," including "double manure salts." The imports of potash salts of these classes amounted to nearly 700,000 long tons, valued at more than \$4,000,000. The imports for consumption of materials entering largely into the fertilizing industry, plus the domestic phosphate rock, reached the total valuation of over \$46,000,000.

	Furnace Number		
	No. 1.	No. 2.	No. 3.
Charging hole from front 1.	1200° C.	1308° C.	1523° C.
" " " 2.	1305°	1353°	1504°
" " " 3.	1346°	1440°	1723°
" " " 4.	1330°	1540°	1723°
Temperature of slag.....	1202°	1200°	1310°
Melting point of slag.....	1100°	1100°	1285°

Reverberatory Flue.—The temperature was determined at ten points in a flue 6 ft. square and 110 ft. long, which conducted the gases from a reverberatory furnace to a stack, 60 ft. in height.

Temperature of gas at furnace.....	1300° C.
" " " 14 ft. from furnace.....	1217°
" " " 27 " " "	1112°
" " " 41 " " "	1097°
" " " 54 " " "	1045°
" " " 67 " " "	911°
" " " 80 " " "	807°
" " " 94 " " "	767°
" " " 107 " " "	727°
" " " 120 " " " (foot of stack)	642°

The accompanying curve shows the tendency of the temperature of gas to rise when it is forced to descend, as well as the tendency of the temperature to fall when it ascends.

Copper Refining Furnaces.—At the different plants, where temperatures were taken of the refining furnaces, the results agreed closely. The following is average practice:

Charge melted and ready to rabble.....	1141° C.
After 25 minutes rabbling.....	1103°
" 75 " "	1103°
At end of rabbling.....	1103°
After 20 minutes poling.....	1110°
At end of poling.....	1117°
Heated to	1125°
After ladling 20 minutes.....	1121°

Copper Converters.—

Matte introduced.....	1170° C.
Turned down to skim.....	1297°
Turned back to blow.....	1284°
Cooling during skimming.....	13°
Temperature of escaping gas at end of 10 minutes....	1260°
" " " " " " 20 "	1270°
" " " " " " 30 "	1275°
" " " " " " finish	1195°

Lead Blast Furnaces.—The temperature of the slag, produced by a lead blast furnace when smelting a given charge, is remarkably constant. The skilled smelter, universally, judges the condition of his furnace by the character of the slag. One of the most important factors is the temperature of the slag. Irregular operation of the furnace is at once shown by a change in the mean temperature of the slag.

The slag from a furnace, running upon a 2/5 slag of the following composition,

Fe	= 30.0 per cent
CaO	= 12.0 per cent
Al ₂ O ₃	= 8.0 per cent
SiO ₂	= 31.0 per cent
Zn	= 10.0 per cent

showed a mean temperature of 1126 deg. C.

The temperature of the slag when using a half slag was 1134 deg. C.

Four furnaces, running nominally upon a 3/5 slag, actually gave the following slags and corresponding temperatures upon two consecutive days. The higher silica content of the slag forced the furnaces off their reduction; but the restoration of the regular slag resulted in a normal furnace. (The table for the second day is given on top of the next column.)

First Day	
Average Slag Analysis	Temperature of Slag
SiO ₂ = 34.0 per cent	Furnace No. 1.....1171° C.
FeO = 30.5 per cent	" " 2.....1159°
MnO = 3.7 per cent	" " 3.....1167°
CaO = 15.0 per cent	" " 4.....1170°
MgO = 1.3 per cent	—
Al ₂ O ₃ = 6.4 per cent	Average 1170°
Zn = 5.8 per cent	

The temperature change for small variations in the silica content of lead blast furnace slags appears to be not far from

Second Day	
Average Slag Analysis	Temperature of Slag
SiO ₂ = 32.0 per cent	Furnace No. 1.....1145° C.
FeO = 31.0 per cent	" " 2.....1145°
MnO = 3.8 per cent	" " 3.....1156°
CaO = 14.5 per cent	" " 4.....1148°
MgO = 1.3 per cent	—
Al ₂ O ₃ = 6.6 per cent	Average 1149°
Zn = 5.8 per cent	

9 deg. C. for each per cent of silica contained in the slag. Upon the above basis a 34 slag (which contains 36 per cent of SiO₂) should give a temperature of 1185 deg. C. The mean of ten determinations upon slags of approximately this composition gave a temperature of 1080 deg. C.

Roasting Furnaces

Long Hand Reverberatory.—Temperatures in the hand reverberatory when roasting leady mattes show considerable variation, depending upon the stage of the roast. The following

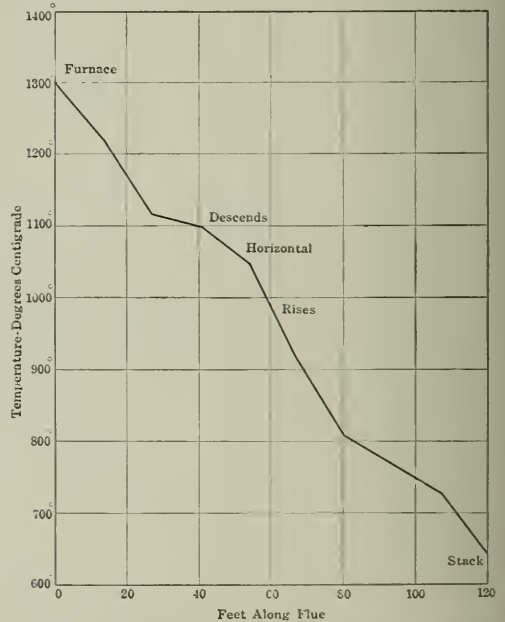


FIG. 2.—FLUE CONVEYING GAS FROM A REVERBERATORY SMELTING FURNACE

series of temperatures, illustrated by Fig. 2, were taken in a furnace 58 ft. long, having ten doors on a side. This furnace did not have a fuse box.

The temperatures on side "A" were taken after moving the charge forward and during the highest heat, while those on sides "B" were taken with a low fire as the charge was about to be drawn.

Door No.	Side "A"	Side "B"
1.....	523° C.	487° C.
" 2.....	575°	612°
" 3.....	668°	*537°
" 4.....	702°	737°
" 5.....	742°	757°
" 6.....	802°	757°
" 7.....	876°	857°
" 8.....	1010°	952°
" 9.....	1215°	1084°
" 10.....	1235°	1197°

*Door No. 3 was opened by a workman for the inspection of the charge

Pearce Turret.—The temperatures at six equidistant points in a single-hearth furnace, roasting heavy sulphide ore, are shown in Fig. 3. In every case the end of the couple was 1 in. above the surface of the ore. The first and last of the three fireplaces were in use.

Herreshoff Furnace.—The temperature of a furnace of this

is 590 deg. C. for the third floor and 680 deg. C. for the fourth floor.

I am indebted to Dr. M. N. Bolles for furnishing the data from which this article has been prepared.
Palo Alto, Cal.

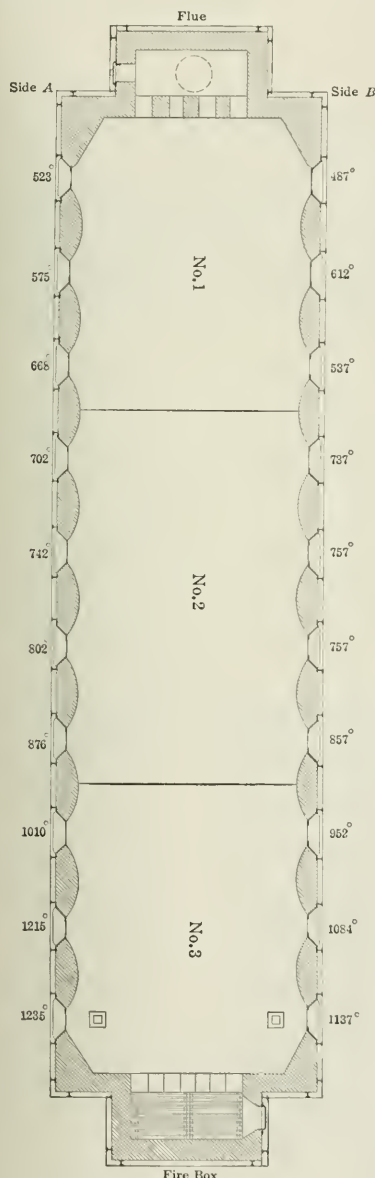


FIG. 3.—LONG HAND REVERBERATORY ROASTING FURNACE

type apparently need not be confined within narrow limits. The upper limit is fixed by the avoidance of sintering and the formation of refractory ferric oxide, while the lower limit is fixed by the necessity of a sufficiently complete expulsion of the sulphur. Temperature readings show that a satisfactory temperature for efficient work and avoidance of ferric oxide

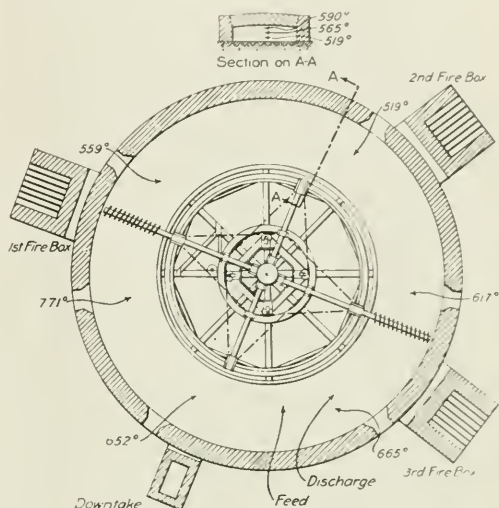


FIG. 4.—PEARCE TURRET ROASTING FURNACE

The metal production of New Mexico in 1912 was valued at \$8,528,000, an increase of \$5,823,000 as compared with the previous year. The important feature of the year's production was the great increase in copper. The principal increase in copper was due to the operations of the Chino Copper Co. The zinc production also showed an increase due to operations around Kelly, where the Tri-Bullion mill produced concentrates by wet dressing and magnetic separation. The Ozark company has recently opened a new mill, employing a flotation process and it is probable that the 1913 production of zinc will show a further increase.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in April, 1913, was sixty-two. The total quantity of ore milled during that period was 2,356,204 tons. There were 10,046 stamps in operation with an average duty of 8.85 tons per twenty-four hours. Tube mills in commission numbered 285. The yield for the month was 784,974 fine oz. gold.

By-product coke ovens have been used in the United States for twenty years, the first having been built at Syracuse, N. Y., in 1893. The original plant consisted of twelve Semet-Solvay ovens, but has since been enlarged to forty ovens. The first ovens of this type had a charging capacity of 4.4 tons of coal and the time required for coking was twenty-four hours. Ovens of the same make to-day have a capacity of 10 tons of coke. Other types of by-product ovens in use in this country are the Otto-Hoffman, the Koppers, the Didier and the Klönne.

The Institution of Mining and Metallurgy, London, has received from Lady Wernher and the co-executors of the will of the late Sir Julius Wernher, the sum of £10,000 as a special endowment fund in memory of Sir Julius. Of this sum, one-half was given by Lady Wernher personally, the remaining half being a bequest. The only condition attaching to the gift is that Lady Wernher desires it to be held intact as an aid in permanently strengthening the institution in which Sir Julius Wernher took a great interest.

The Arizona Copper Company produced during the month of June, 1913, 1500 tons of copper.

Iron and Steel Market

July has seen a curious reversal of sentiment among steel producers, whose views had been chiefly pessimistic, curious in that it is subjective, not due to any discernible change in the mental position or the actions of buyers whose commitments have continued through the month of July at the low rate reached late in June. Normally such changes in sentiment are due either to a change in the flux of business in the industry or to a change in general industrial and money conditions, presaging, by reasoning from the general to the particular, a change in the branch of human activity known as the steel industry. Perhaps this latter suggestion tends to discredit the much quoted dictum that "iron is the barometer of trade," for a barometer is expected to presage changes and if a change in the rate of flow of steel orders presages a change in general industrial conditions, then these general conditions cannot be expected to act as prophet for the steel industry. The facts seem to be that if iron ever was really the barometer of general trade, it was so only to those who could not discern other signs just as trustworthy, though less spectacular, and that steel makers now undertake to see farther ahead than formerly, in which effect they sometimes meet eventual disappointment.

It is the duty of this report to depict, rather than to defend, the state of mind existing in the industry. The facts upon which the improved sentiment among sellers seems to be based are simply that the money strain appears to show signs of relaxing, the crops will soon be out of danger, uncertainties arising from Congress will soon be resolved, and the consumptive powers of the country, even in a time of great reserve among buyers and consumers, have been shown to be large. These facts, however, are also the property of buyers, who have not changed their attitude.

It does not, however, follow that if there is an improvement in the outlook buyers should at once change their policy. They are taking full tonnages of material; they would buy for forward delivery only to protect themselves against loss by prices advancing or by deliveries upon belated purchases proving inadequate. No such dangers confront them. As to prices, when there has been a prolonged advance such as that of 1912, followed by a period of stationary prices, the normal development is a decline, not a fresh advance superimposed upon the previous one. The dip is large or small, depending upon the recuperative power of the mill position through the order books tending to fill instead of to empty further. In the past few months the producers have had to face an eventual readjustment in prices, and merely to conclude that the readjustment will prove small in extent and brief in duration would be sufficient for them to take the more hopeful view of the future which they now seem to hold.

The comparison between productive capacity and the demands of the country, plus the export trade, is unusually favorable. In the past few years capacity has increased at much less rate than demand has normally increased, and in no quarter has any valid argument been made that the rate of increase in demand is destined to undergo a material decrease. It should, therefore, be unusually easy to fill the steel mills to capacity. This, indeed, has been the experience of recent years, for all those who have observed steel market conditions over a period of years have expressed surprise at the prosperity of this industry, compared with business prosperity in general.

The July output of iron and steel has been very large for a July, showing a smaller decrease from May and June than is usually forced by weather conditions alone. Shipments have been readily taken. Nevertheless some mills are rapidly reaching the end of full engagement and must soon receive fresh business at a greater rate or materially curtail production. A peculiar feature of the situation is that there is much diversity in the position of different mills, for some are very well booked for several months of substantially full operation.

Prices of steel products have undergone no material change. The majority of products are firmly held, though the grip can hardly be maintained long without an improvement in actual

bookings. In wire products there is a continuation of the weakness, which may be excused as seasonable. In sheets the market is somewhat softer, and the leading interest has now been forced to take cognizance of the fact by adjusting prices to the market upon current shipments made under contracts at its full prices, below which the open market began to sag months ago, but which did not come into actual operation until about July 1. It does not revise the contracts as a whole, nor is it soliciting fresh contracts at the cut prices.

Pig Iron

The pig iron market as a whole has not developed a definite turn. Southern iron has grown distinctly stronger, but northern iron thus far has not responded fully to the leadership which in all previous movements, whether upwards or downwards, it has recognized. Most northern quotations dropped during July, but only slightly, and in that very fact suggesting that the end of the long decline is in sight. Buying has been considerably heavier, though not altogether as heavy as is usual at a time which history finally shows to have been distinctly a bottom point. The purchases have been fairly numerous and have covered fairly extended deliveries, frequently to the end of the year, but the individual purchases have not averaged as great, in point of tonnage, as is expected from the buyers. Early in the month the central western market was expected to respond to the coke market, the operators insisting upon \$2.50 for renewal of contracts for the second half, a high price relative to pig iron. Furnaces have contested this price, some even banking, but the operators have maintained their position, curtailing production to the extent that contracts were not renewed. The furnaces which accepted the price bought for July only, and another contest is marked with respect to August coke. The market is quotable as follows: Southern No. 2 foundry at Birmingham, \$10.50; No. 2X foundry, Philadelphia, \$15.50; No. 2X foundry, Buffalo, \$13.75; No. 2 foundry, Cleveland, \$14.75; No. 2 foundry, Chicago furnaces, \$14.75; at valley furnaces (90 cents higher, delivered Pittsburgh) Bessemer, \$15.75; basic, \$14.35; No. 2 foundry, \$13.75; forge, \$13.50.

Steel

The market for billets and sheet bars has increased slightly in strength, to the extent that buyers are making less effort to break prices. At no time has steel been otherwise than scarce. In sympathy with cut prices on wire products rods have declined slightly. The market is quotable at \$26.50 for billets, \$27.50 for sheet bars and \$28 for rods, at maker's mill, Pittsburgh or Youngstown.

Finished Steel

Regular prices are as follows, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Common iron bars, 1.65 cents, Pittsburgh; 1.57½ cents, Philadelphia; 1.50 cents, Chicago.

Wire nails, \$1.70, base; plain wire, 1.50 cents.

Sheets, blue annealed, 10 gage, 1.70 to 1.75 cents; black, 28 gage, 2.25 cents; galvanized, 28 gage, 3.30 cents; painted corrugated, 28 gage, 2.45 cents; galvanized corrugated, 28 gage, 3.35 cents.

Merchant steel pipe, 79 per cent off list for ¾ to 3-in.

Steel boiler tubes, 3½ to 4½-in., 69 per cent off list.

Standard railroad spikes, 1.70 cents, Pittsburgh; 1.75 cents, Chicago.

Button head structural rivets, 2 cents; cone head boiler rivets, 2.10 cents.

Case-Hardening and Heat-Treating of Steel, "a book of useful information and practical rules" (38 pages), has just been issued in its fifth revised and enlarged edition by the Ideal Case-Hardening Compound Co., U. S. Rubber Co. Bldg., New York.

Evolution of Methods of Handling Slime. V.

The Rand, South Africa.

By H. N. Spicer.

In the article published last month, slime treatment on the Rand was revised in a general and historical way. It is proposed now to describe in greater detail some of the special methods that have been developed or introduced there from

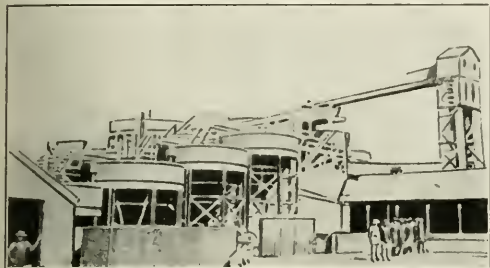


FIG. 1.—SLIME TREATMENT CONES AT MEYER AND CHARLTON

time to time, and to consider the practice at some of the individual mills. As stated in the last article, intermittent decantation has been the accepted method of recovering dissolved gold from slime, from the time of the first commercially successful attempt at the Primrose mine in 1804 until several years ago; and despite the introduction of other methods is still the leading process for slime treatment.

The first complete slime plant to operate continuously at a profit on slime coming directly from the stamps was erected at the Crown Reef mine about the middle of 1806. Settling pits were used for collection and decantation, and milk of lime was employed to assist settlement. The collected slime was mixed with cyanide solution and pumped to treatment vats where intermittent settling and decantation took place. This plant, though crude, was the forerunner of the present design of intermittent decantation as used today on the Rand.

Efforts to Supplant Intermittent Decantation

On account of the very evident disadvantages and drawbacks of the process, efforts have been made from time to time to abandon it in favor of some less cumbersome and more efficient method. One of the first attempts in this direction was made at the Robinson mine in 1807, where an adaptation of the Deeble slime vat was used. The Deeble machine, which had been introduced from Australia, was nothing more than a paddle agitator with a provision for raising the arms out of the pulp while the slime was settling. As used at the Robinson, the agitator tank was fitted with a filter bottom. After agitation, and during decantation and settlement, a vacuum was created beneath the filter for the removal of more solution than was possible by decantation alone. By this combination small charges of slime could be dewatered to about 28 per cent. moisture, and in successive periods of thickening, dilution and agitation with cyanide and wash solutions, the slime was impoverished and finally discharged to the dump.

This process gave promise of great results: it not only reduced the quantity of solutions to be handled, but yielded a gold solution of much better grade than was obtained by direct decantation. It developed weak points, however, in the excessive wear and tear on the filter bottom, and in the cost of operating the vacuum pumps, and was finally abandoned. A point of interest that should not be overlooked is that this process may be considered as the first attempt to use vacuum filtration on the Rand.

First Attempt at Continuous Slime Treatment.

While the Deeble slime vat was being tested at the Robinson, a small experimental plant for continuous decantation in cones was in operation at the City and Suburban mine. The preliminary results obtained were so encouraging that a large plant was designed for the Van Ryn, which was to have been erected in 1899, but owing to the Boer war was not built until 1903.

The underlying principle of the process was to pass the slime pulp through a series of conical tanks, so arranged that the thickened underflow from the first would gravitate to the second, and so on. The thickened slime entering the successive cones was diluted and washed with an inflow of barren solution, and at the same time an enriched solution overflowed. By this means it was expected that the soluble gold would gradually be washed out of the slime. But a serious obstacle was encountered in getting a regular cone discharge that contained even a reasonably low percentage of moisture; in other words the cones did not prove efficient as uniform thickeners, and it was impossible to thoroughly wash the dissolved gold from the slime.

This attempt at continuous decantation was not without value, however, even though it failed of its original intention. It showed clearly the advantage to be derived by impoverishing slime as much as possible prior to filtration, and this knowledge was used by the same engineers who later built the Meyer & Charlton and New Goch Mills. In designing these plants, the idea of continuously decanting gold-bearing solution from the slime was retained, but plate-and-frame presses were added as a means of washing out the last traces of dissolved gold.



FIG. 2.—NEW GOCH MILL, SHOWING CONES FOR SLIME TREATMENT

Thus a combination of continuous decantation and filter pressing was developed on the Rand, but was never widely adopted. Filter presses had been used successfully in the mills of Western Australia prior to their adoption on the Rand. The expense of their maintenance and operation, however, was too great to be borne by low-grade ore, with the consequence that the combination did not prove economically successful, and was

never a serious competitor of intermittent decantation. In Fig. 1 is shown the cone decantation plant at the Meyer & Charlton. This was built in 1905 and is still in operation. Fig. 2 shows the plant of the New Goch Company with the decantation cones in the center of the photograph.

Nichols' Method of Slime Settlement.

During recent years there have been but few attempts to supplant the recognized system of intermittent decantation. Notice should be taken of the effort to displace slime collecting tanks by a device embodying the principle of accelerated settlement, *i. e.*, continuously removing the settled solids which acted as an obstacle to the free settlement of the solids above. Considerable work was done along this line with the Nichols slime dewaterer which removed the slime as fast as it settled on to a belt traveling slowly up the inclined bottom of the slime receptacle.

This process was effective in producing thickened slime, which was discharged at the head pulley with about 30 per cent. moisture. This was a great improvement over the results ob-

place. Barren solution was then introduced through the central tube, the pressure being so regulated as to keep the slime in suspension but leave from three to six inches clear solution on top. Decantation then commenced and the clear solution was drawn off continuously for precipitation at the same rate at which the washing solution was admitted. This operation was continued until the gold-bearing solution was displaced, when the inflow was stopped. The effect of the operation was to keep the slime in suspension, at the same time maintaining several inches clear solution on top, and gradually displacing the valuable solution from the bottom upward. In like manner the slime could be washed with water, and, after settlement and decantation, sluiced to the dump.

This process was operated on a large scale, and is reported to have given good metallurgical results. It required delicate adjustment, and difficulty was experienced in preventing the clogging of the pipes. Its advocates claimed better extraction, lower working cost, and a saving of one-third in the plant required for ordinary decantation. At the Robinson mine the process was finally replaced by a large, modern Butters filter.

The Crosse Method of Slime Treatment.

Another attempt was made to supplant intermittent decantation by the use of the Crosse method of slime treatment at the Crown mines. As a preliminary step to the process, the slime was collected and thickened by settlement in the ordinary manner. The subsequent treatment was given in a conical vat of special construction. It was equipped with a central air-lift for maintaining the slime in suspension while the vat was being filled, and had also a loading well or inner concentric baffle extending about half way to the bottom. This provided an annular area free from disturbance, from which fairly clear solution could be decanted for precipitation. In conjunction with the treatment vat, another conical vat with concentric baffle but no air-lift, was used for the final settlement and decantation of the impoverished slime.

The operation was as follows: Dewatered slime from the collectors was pumped with cyanide solution to the treatment cone, and while the latter was filling, its air-lift was operated to keep the slime in suspension and aid solution of the gold. When the cone was full, agitation was discontinued, and the charge was circulated from the bottom of the cone through an outside air-lift, being delivered again inside the circular baffle of the cone. During this operation the nearly clear solution in the quiet annular zone was being decanted for precipitation, and barren solution in equal quantity was being added to the circulating slime. When circulation had continued long enough to dissolve and remove the gold, the pulp was transferred to the conical settler for the recovery of solution by settlement and decantation.

The Crosse method continued in use at the Crown mines for about two years, but was eventually discarded, although it was reported to give an extraction of 95 per cent.

Arbuckle Cones.

The last attempt to supersede standard decantation, other than adopting vacuum filtration, was made at the Benoni mine in 1910. At that time a large plant was erected to operate the Arbuckle process of continuous decantation in large cones. A view of these devices under construction at the Benoni is given in Fig. 3.

The cones were fitted with circular baffles to assist settlement. The inventor proposed to use inclined screw conveyors to elevate the thick slime from the bottom of one cone to the top of the next, but the scheme was a failure. The greatest difficulty was encountered in maintaining uniform conditions which would yield a clear solution overflow and a thick underflow that could be elevated by screws. After making several alterations the entire slime plant was abandoned in favor of the regular decantation process.

Crushing in Cyanide Solution.

To the same engineers who designed and built the Meyer & Charlton and New Goch mills must be given the credit of first

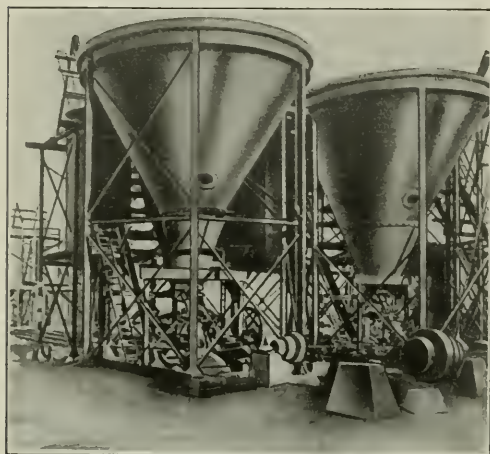


FIG. 3.—ARBUCKLE CONES AT BENONI MILL

tained by settlement in slime collectors, but the small capacity of the machines was a disadvantage and they were found impractical.

The Adair-Usher Process.

In 1909 a company was formed to operate the Adair-Usher process of washing and decanting slime, and a plant was built at the Robinson mine. In this process the slime was collected in large settling tanks as for ordinary decantation, and after the water was decanted the thick pulp was transferred to the Usher treatment vat where it was mixed with cyanide solution.

The Usher treatment vat had either flat or concave bottom, and was fitted with a central vertical tube connected at the bottom with perforated pipes radiating from the center, parallel with and near the bottom of the vat. The central tube was 10 or 12 inches in diameter and projected several feet above the top of the vat. At the bottom it was enlarged into a chamber 4 to 6 inches deep and about 18 inches diameter, closed at the bottom by a flange to which the radiating pipes were attached by elbows.

In operation the slime was transferred from the collector to the treatment vat, and at the same time cyanide solution was admitted through the central tube. The solution passed out through the perforations in the radiating pipes, rising and mixing with the slime. The solution and slime formed a thin pulp which gradually rose until it filled the vat to the desired height. At this point the charge could be circulated in the vat by means of pumps until dissolution of the gold had taken

adopting the practice of crushing in cyanide solution. As mentioned in the last article this practice had been attempted on a small scale as early as 1892 but was abandoned. With this minor exception, and until the Meyer & Charlton and New Goch mills were built, it had been customary on the Rand to crush in water. The disadvantages of the system, particularly in the resulting slime treatment, have already been referred to: the necessity and difficulty of dewatering, and the large and cumbersome plant required for decantation.

Notwithstanding the fact that apparently excellent work was done at the Meyer & Charlton and New Goch mills by crushing in solution, a number of difficulties arose. Chief among the arguments against the process were the impossibility of accurately sampling the ore, and the difficulty of overcoming the bad effects of cyanide on the amalgamation plates. In the light of our present metallurgical and mechanical knowledge it would appear that these difficulties should not be insurmountable; but the fact remains that the practice has been generally abandoned on the Rand. The Meyer & Charlton is today the only mill crushing in cyanide solution, and it is probable that the practice would be abandoned there but for the fact that the mine probably would be exhausted in a few years, and the prospect does not warrant the capital expenditure for the necessary changes.

Within the last two years the area of amalgamated surface used per ton of ore treated has been greatly reduced, so that the task of keeping plates properly dressed is not as great as formerly. In view of this great change the writer ventures the opinion that it will not be long before the other obstacle to crushing in solution, viz., sampling, also will be overcome, and that some engineer will again attack and solve the problem of crushing in solution.

Agitation of Slime.

In the beginning of slime treatment on the Rand, paddle agitators or mechanical stirrers were commonly used to hasten dissolution of gold by giving a thorough mixture of solids and solution. But with the wider adoption of intermittent decantation, more dependence was placed on centrifugal pumps to give the agitation and aeration required. In the usual decantation plant, as described in the last article, the charge of slime is transferred from one vat to another, or circulated within the same tank, by means of centrifugal pumps, and the agitation thus secured has been considered sufficient. Indeed with one or two recent exceptions no plants have been built with other provision for agitation.

In 1903 a plant was built at the Bonanza mine, in which the slime collectors were provided with paddle agitators. After settlement and decantation of the water, cyanide solution was added in the ratio of four parts solution to one of slime, and the pulp was agitated for eight hours. The charge was then pumped to a settler from which solution was decanted for precipitation. The thick slime was again returned to the agitator, mixed with barren solution in the same ratio as before, and given a short agitation. After settlement of the slime, the solution was decanted, being used for the treatment of the next charge of slime. The impoverished slime was finally pumped to a settler filled with water, and after final settlement was sluiced to the dump.

The extraction by this system at the Bonanza mill was given officially as 83 per cent. Although the process was fairly efficient from a metallurgical point of view, the cost of operating the agitators was too high and the method was never widely adopted.

Modern Agitators Introduced.

The introduction of vacuum filtration on the Rand has necessitated the use of modern forms of agitators which have been so widely used in other countries, and today the Brown or Pachuca, the Hendryx and the Dorr agitators may be found in operation. Continuous agitation in a series of Pa-

chuca tanks has been tried at the East Rand Proprietary mines.

Owing to the sandy nature of the slime from the deep level ores, the continuous system has met with difficulty in preventing the classification of coarse particles and the consequent accumulation of this material in the tanks. This has been true particularly of cone-bottom agitators. The Dorr mechanism, however, is operated in flat-bottom tanks, and absolutely prevents the building up of solids.

Speaking broadly, Rand slime requires only from six to eight hours' agitation to get the gold in solution, and it is probable that if the ore were crushed in cyanide solution the time required for agitation would be materially lessened.

Vacuum Filtration.

The most striking change which has taken place in Rand slime treatment was inaugurated in 1910 when the first Butters vacuum filter was built at the Crown mines. In a comparatively short time following the operation of this plant, other vacuum filters were built at the New Modderfontein.



FIG. 4—SAND AND SLIME PLANT, MODDERFONTEIN B MILL

Brakpan, Robinson and other mines. Today there are eight such installations with an aggregate capacity of 6500 tons of slime per day.

The mill at the Crown mines commenced operations in October, 1910. Its equipment consists of 160 heavy gravity stamps crushing in water, ten 22 ft. by 5 ft. 6 in. tube mills, amalgamation plates, cone classifiers, five 50 ft. by 10 ft. 6 in. sand collectors, twelve 50 ft. by 12 ft. sand treatment vats, five 56 ft. by 12 ft. slime collectors, Pachuca agitators and Butters vacuum filter.

The slime is first settled to about 50 per cent. moisture, mixed with cyanide solution in the ratio of two parts solution to one of solids, and agitated for about eight hours prior to filtration. Thus the agitation and filtration plants displace the first and second treatments in the ordinary decantation system. A further advantage is found in having to precipitate only two tons of solution per ton of slime instead of three tons as in decantation. The extraction probably is 5 per cent. better than in good decantation practice.

With the exception of the fact that the ore is crushed in water, this mill may be considered the first up-to-date plant on the Rand.

In Fig. 4 is shown the sand and slime treatment plant at the Modderfontein B mine. The ore is crushed in water, and classified into sand and slime by means of cones. Three 50-ft. slime collectors, shown in the right center of the view, are used to dewater slime by decantation, after which it is pumped with cyanide solution to the four Pachuca agitators. A gravity vacuum filter plant is used, and the discharged slime is sluiced to settling ponds for the recovery of the water.

Recovery of By-Products in the Modern By-Product Coke Oven Plant*

By C. A. Meissner

Chairman, Coke Committee, U. S. Steel Corporation

The recovery of the by-products from the gases evolved in the coking process in a modern by-product coke oven plant—namely, gas, ammonia, sulphate of ammonia, and tar as well as benzol—is a very broad subject and I will try to confine myself as closely as possible to the points of main interest.

Briefly, the gas as it leaves the ovens through the foul gas

I understand that in Germany they recover up to 95 per cent of the total ammonia available in the coal. A factor that assists the Germans is that their coal is almost invariably washed and, therefore, contains a certain amount of moisture, which is effective in obtaining a larger percentage of the available ammonia.

So far as our experience has gone, we seem to have no material difference in our recovery percentage when running between 16½ and 20 hours coking time. We have practically no experience over any period of time on 24-hour coking time, excepting at Farrell, but there the conditions are to my mind

not of such a nature as to be quite comparable to our conditions at the other plants. This subject is undergoing careful study at all our plants.

The same applies more or less to the question of whether a hot top of the oven is more likely to increase the loss of recoverable ammonia than a cooler top. The general impression so far has been that the

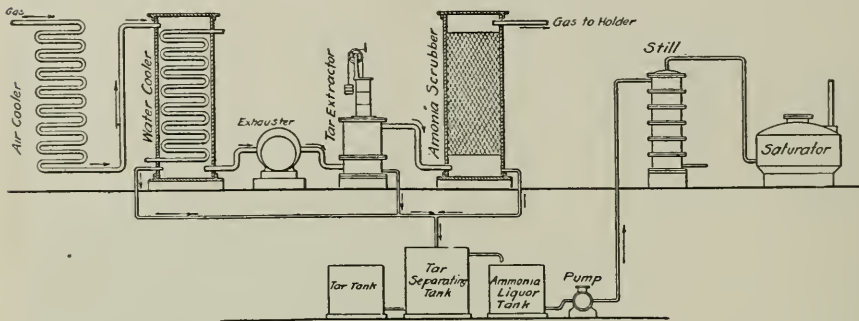


FIG. 1.—THE INDIRECT AMMONIUM SULPHATE PROCESS

main, located above the ovens, is taken over to the by-product plant in large mains and ordinarily is there cooled, the tar extracted, naphthalene thrown down as far as possible, the ammonia is scrubbed out of the gas by passing it through water towers or other apparatus, weak ammonia liquor is distilled, the ammonia passed into lead-lined saturators containing sulphuric acid and sulphate of ammonia is formed, or the distilled ammonia is placed in tanks and shipped as strong ammonia liquor. This is an outline of the *old indirect process*.

In the *direct recovery process* the scrubbing of ammonia from the gases is avoided by passing the gases through the saturator after the tar separation has been accomplished. In some of the so-called direct processes the tar is separated by cooling the gases and in this case a certain proportion of the ammonia is precipitated with the tar and must be distilled. In some of the direct processes the tar is separated without precipitation of the ammonia salts. These latter processes are, therefore, more preferably called direct than the processes where some of the ammonia is precipitated, which should be more properly called semi-direct.

There is also another process in which the sulphur contained in the gases is utilized, generally referred to as the "thio-sulphate process," of which I will give later a brief but more detailed description.

We have, therefore, at present really four processes that are largely used all over the world for recovering by-products:—namely, the *indirect*, the *semi-direct*, the *direct*, and the *thiosulphate*.

The effect of the coking time upon the recovery of by-products, particularly ammonia, is a subject that has not been fully worked out. Apparently the longer coking time increases the percentage of recovery of ammonia and the losses in ammonia become greater as we decrease the coking time up to a certain point. I doubt whether this point can at the present time be definitely settled upon, because there are other reasons for ammonia losses than the coking time itself. We have generally felt that with a short coking time we do not get as much of the total available ammonia as at plants having a longer coking time.

cooler top is the better in this respect. Points like these have to be carefully balanced, because from them arises the question of what is more economical to produce, a good blast furnace coke or a larger by-product recovery. The object of the Steel Corporation has always been, and I think rightly so, to make a good furnace coke and then to recover as much of the by-products as is consistent with this. The money loss through inferior coke and its detrimental effect on blast furnace practice soon offsets any slight gain through increased ammonia yield at the coke oven.

I believe you will be interested at this time in a few illustrations and a short description of the different by-product apparatus for extracting the tar, ammonia and benzol from the gas, which latter is then returned to the ovens for heating or carried off as surplus gas to the steel plant or other places where it is to be used.

The Indirect Process for Recovery of Ammonia

In the indirect process (Fig. 1) the gas coming from the ovens is air- and water-cooled to allow the deposition of tar and combined ammonia liquor, and is then passed through scrubbers where the free ammonia contained in the gas is

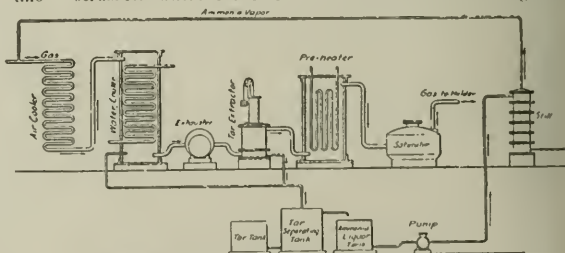


FIG. 2.—KOPPERS' DIRECT AMMONIUM SULPHATE PROCESS

washed out by means of water. This weak ammonia water, together with the ammonia liquor separated from the tar, is then treated with steam and milk of lime in a still, and the ammonia vapors either passed into an acid saturator or else condensed into a strong ammonia liquor, depending on whether it is desired to produce sulphate or liquor.

*A part of a paper on "The Modern By-Product Coke Oven," presented at the recent meeting of the American Iron and Steel Institute.

Koppers Semi-Direct Sulphate Process

In this process (Fig. 2) the gas leaving the ovens is first air- and water-cooled to a temperature of 104 deg. Fahr. This throws down most of the tar, the water vapor, the combined ammonia and some free ammonia.

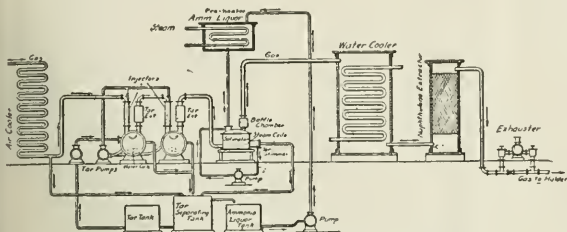


FIG. 3.—OTTO DIRECT AMMONIUM SULPHATE PROCESS

The ammonia liquor is separated from the tar and treated in stills with steam and milk of lime in the usual way, the ammonia vapors being either converted into ammonia liquor or else returned to the system for conversion into sulphate, by entering the gas main at a point prior to the water coolers and traveling with the gas to the saturator.

The gas, after leaving the water-coolers freed from most of the tar, passes through an exhaustor to P. & A. tar extractors, where the remainder of the tar is removed. Thence the gas is returned to a heater where the temperature of the gas is raised to about 176 deg. Fahr., the heating medium being exhaust steam or the uncooled gas from the ovens, usually the former in the later plants.

The heated gas then passes into the saturator, where the ammonia in the gas is neutralized by the sulphuric acid with the formation of ammonium sulphate. Leaving the saturator at a temperature of about 120 deg. Fahr., the gas passes without further cooling to the gas holder for return to the ovens and its other uses.

Otto Direct Sulphate Process

In the Otto direct sulphate process (Fig. 3) the gas from the ovens is air-cooled to a temperature of about 200 deg. Fahr. and is then further cooled to a temperature a little above the dew point of steam, about 160 deg. Fahr., by means of a tar spray.

The object of cooling to about the dew point is in order to prevent the deposition of ammonia liquor, which would occur if cooled to a lower temperature, and thus necessitate the use of stills as in the other processes. By keeping the temperature a little above the dew point this ammonia remains as vapor in the gas. Experience has shown, however, that some ammonia is invariably thrown down with the tar.

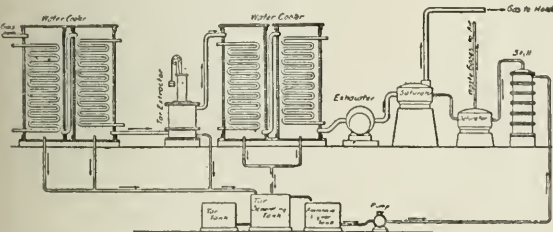


FIG. 4.—COLLIN SEMI-DIRECT AMMONIUM SULPHATE PROCESS

The spraying with tar is performed by Koerting injectors in a two-compartment tar tank in series, so the gas is twice sprayed. The temperature of the spraying tar is kept at the right degrees by means of water-cooling pipes in the tank.

The surplus tar washed out of the gas overflows into a regular separating tank, where the ammonia liquor which has come

down with the tar is removed. The amount of this liquor is small.

The tar-washed gas then passes through a stationary bell similar in function to a P. & A. tar extractor, where the last of the tar is supposed to be removed, but on account of the high temperature of the gas some tar continues to remain therein.

The gas then passes through a covered gas main to the saturators, where neutralization of the ammonia occurs with the formation of ammonium sulphate, which is ejected and dried in the usual way.

On account of imperfect separation of the tar from the gases, some tar is always deposited in the saturators; therefore, it is equipped with a lead dam and siphon to allow the top portion of the mother liquor to be withdrawn into an open top tank in which the tar is skimmed off and the mother liquor returned to the saturator.

On account of the gas entering the saturator being fairly well saturated with water vapor, which, if deposited here, would weaken the acid bath and prevent proper absorption of the ammonia, the saturators are equipped with steam coils made of lead, and the bath is kept heated by means of steam to a temperature high enough to insure the retention of the water vapor in the gas.

The ammonia liquor separated from the tar is heated by steam coils in a small tank, and is then introduced direct into

COLLIN DIRECT SULPHATE PROCESS.

AT
KÖHLER BERGSCHAFT THAL, COLLIERY, GERMANY

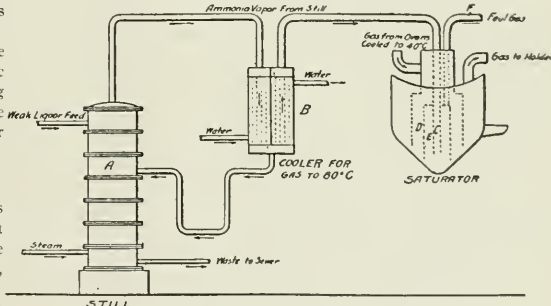


FIG. 5.—COLLIN DIRECT SULPHATE PROCESS

the saturator. After leaving the saturators, the gas enters a tank containing brick baffles to allow separation of the acid and thence to horizontal tubular water-coolers, where the gas is cooled to separate the water and as much naphthalene as possible. At some places a further naphthalene extractor is being installed.

After leaving here, the gas is drawn through an exhaustor and pumped to the gas holder, ovens, etc.

Temperatures taken at a representative plant at the various stages of the operation were as follows:

Entering Koerting injectors.....	200° F.
After spraying with tar.....	162° F.
Entering saturator.....	160° F.
Leaving saturator.....	185° F.
Entering water coolers.....	171° F.
Leaving water coolers.....	102° F.

Collin Semi-Direct Sulphate Process

In this process (Fig. 4) the gas from the ovens passes through the usual horizontal tubular water-coolers where most of the tar and combined ammonia are separated, then through a P. & A. tar extractor where the remainder of the tar is separated. The cleaned gas then passes through another set of water-coolers exactly similar to the first set, and then through the exhaustors to the saturator.

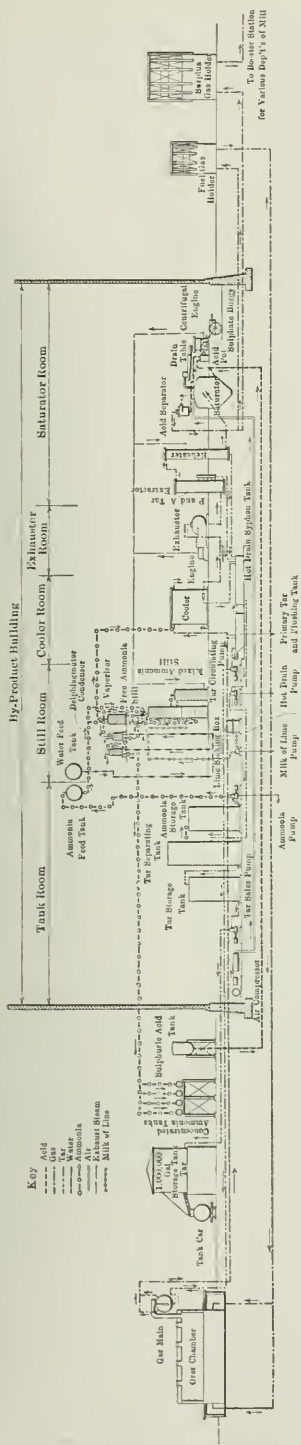
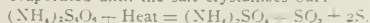


FIG. 8.—DIAGRAMMATIC SKETCH OF FLOW OF GASES, EXTRACTION OF VARIOUS BY-PRODUCTS, ETC., IN A BY-PRODUCT COKE OVEN PLANT

the sulphur is filtered off, and the solution of ammonium sulphate is evaporated until the salt crystallizes out:



Recovery of Benzol

Benzol is defined as a colorless, inflammable, liquid compound having the chemical formula C_6H_6 .

Benzol is recovered from coke oven gases by scrubbing them with light oil, in washers of various types, after the ammonia has been separated from the gases; that is, after the gases leave the ammonia scrubbers in the indirect process, or after the gases issue from the saturator in the direct process. Some benzol is also precipitated with the tar.

The oil used for scrubbing is usually in this country a light oil of specific gravity about 0.8; it should be neither too thin, in which case long enough contact with the gas is not obtained, nor too thick, in which case there is a tendency to solidify.

After scrubbing, the mixture of washing-oil and benzol is conveyed to a still, where the benzol is driven off, with steam as the heating medium. The benzol so obtained is designated as "crude" benzol and contains about 50 per cent benzol. This is either shipped as such, or is purified by further distillation to various grades of benzol or fractionally distilled with separation of toluol and xylol when required. The oil used for scrubbing is used over and over again until it gradually becomes too thick by accumulation of naphthalene and tar.

The yield of benzol depends largely on the volatile contents of the coal; usually the higher the volatile matter the higher is the yield of benzol.

Uses of By-Products

Tar.—This is used largely for two purposes, namely, distillation for production of creosote and light oils and pitch, and for use in road making. Both of these uses of tar have increased very greatly in the last few years and the tendency for still further increase indicates a very large proportion of the future production will be similarly used. Tar is further being used successfully in open-hearth furnaces, replacing producer gas; also in heavy oil engines, providing it does not contain too much free carbon. It has also been burnt with success under boilers through specially devised burners.

Ammonia and Ammonium Sulphate.—The ammonia in the form of concentrated ammonia liquor is used in making anhydrous and aqua ammonia for refrigeration, and other purposes, in the production of soda ash, explosives, and in other lines of chemical manufacture. Sulphate of ammonia is very largely used as a fertilizer, although in some cases it is converted to ammonium hydrate for use in chemical manufacture. When used as a fertilizer it is usually mixed in definite proportions with other fertilized ingredients, such as acid phosphate, potash, etc., according to the conditions.

Coke Dust and Coke Breeze.—The coke dust is used to advantage in the steel mills in soaking pits, etc. Coke dust up to $\frac{1}{8}$ in. in size can also very profitably be burnt in specially designed grates under boilers. The coke breeze is sold for domestic purposes, having been screened and sometimes sized. In locations where coal is cheap the disposal of the coke breeze may at times be somewhat troublesome.

Surplus Gas.—The surplus gas obtained from the coke ovens, which has a value of 400 to 650 B.t.u. per cubic foot, depending on the coal used, is one of the most valuable by-products. It can be used for all sorts of heating purposes at the steel plant and its value in replacing coal for all purposes at one of our steel plants is equivalent to approximately one-half million net tons of coal per year, based on 40,000 cu. ft. of surplus gas being equivalent to one net ton of coal at that point.

This surplus gas is also used for illuminating purposes and can be piped long distances for this purpose. In Europe it is used quite extensively in gas engines. Experiments have also been made using it in the open hearth furnaces alone or mixed with blast-furnace gas, the latter method having been largely adopted in certain parts of Europe and I believe ultimately in this country will receive due consideration.

Benzol.—The principal uses of benzol and its products are:

- (1) As a motor fuel.
- (2) To enrich the illuminating power of gas.
- (3) In manufacture of aniline dyes.
- (4) As a solvent for rubber and similar substances.
- (5) In manufacture of gunpowder.

Production of benzol in Europe has increased enormously in recent years due to its increasing consumption as a motor fuel in place of gasoline. At the same time the price has steadily increased, more than keeping pace with the increased production.

Notes

The metal production of Idaho in 1912 is estimated by the Geological Survey to be valued at \$21,466,500, an increase of \$2,365,600 over 1911. The output of gold remained about the same, being valued at \$1,381,200. The silver output of 8,294,745 oz. was less than 1 per cent greater than in 1911. The production of copper increased about 2,340,000 lb., and the production of lead 11,620,132 lb. Record shipments of zinc concentrates were made, the production being 13,905,502 lb., an increase of 5,565,253 lb.

Recent publications of the Bureau of Mines include *Technical Paper No. 37*, "Heavy Oil as Fuel for Internal-Combustion Engines," *Technical Paper No. 44*, "Safety Electric Switches for Mines," and *Bulletin No. 51*, "The Analysis of Black Powder and Dynamite."

Synthetic tannin is reported to have been made by Dr. Edmund Stiasny, of the University of Leeds, England. The product, which is called neradol, is made from tar distillation products, the synthesis being carried out by sulphonation cresylic acid and combining it then with formaldehyde. The product yields a white color to tanned materials.

Cleansing of Sulphate Pulp.—It has been pointed out by Mr. Philip B. Sadtler, of the Swenson Evaporator Company, Chicago, that by using a battery of blow tanks to systematically wash the liquor out of the pulp, this can be done with the least expenditure of water and the highest cleansing effect. Incidentally, this is one of the best methods for doing away with the very objectionable odor of the sulphate pulp mill. Pulp is blown under pressure from the digesters directly into the blow tanks, the battery being used in countercurrent system for washing. The small expenditure of water leaves a minimum of water to be evaporated and hence produces economies in this end. This is one of the several innovations which Mr. Sadtler is introducing into some of the newer mills with which he is connected.

The **Hoskins Manufacturing Company**, of Detroit, Mich., has issued its illustrated bulletin No. 3, entitled *Hoskins Thermo-Electric Pyrometers*, which in addition to being a catalog of the well-known pyrometers of their manufacture contains a considerable amount of information of interest to pyrometer users.

The **Bristol Company**, of Waterbury, Conn., has issued bulletins 175 to 178. Bulletin 175 gives a brief illustrated description of a long-distance indicating electric resistance thermometer, which has recently been developed by this company and is especially adapted for accurate measurement of low temperatures over long distances. Bulletin 178 describes the Bristol wet and dry bulb recording thermometers, which have been designed for recording temperatures of wet bulb and dry bulb on the same chart, on the basis of which the humidity of the atmosphere can be computed.

The **United States Metal & Manufacturing Company**, 105 Broadway, New York, has recently added to its line of railway specialties the sale of the Lincoln arc welding and cutting machines, products of the Welding Materials Company. These machines will weld cracks and seams in locomotive fire boxes, flues in the sheet, engine frames, and other work of similar character, incidental to steam railroad material, also broken motor cases, truck frames, broken and worn armature shafts, etc., steel, grey iron and malleable castings, and will cut off

heads, risers, mis-runs, rails, etc., and wreck steel structures, cut up scrap boilers, etc.

The **Chicago Pneumatic Tool Company** have just issued illustrated Bulletin No. 127 on pneumatic drills, reamers, wood borers, flue rolling and tapping machines and grinders. Their Bulletin No. 34L gives "general pneumatic engineering information"; it shows not only the large number of types of compressors made by this company, but should prove particularly useful on account of the various convenient tables and data which can be used in connection with compressor or compressed-air calculations.

The Rational Absorption of Hydrochloric Acid.—The article by Dr. Theodor Meyer on this subject, published in our May, 1913, issue (page 267), has been reprinted by the German-American Stoneware Works, 50 Church Street, New York City, and copies of this reprint may be had on application from this company. The German-American Stoneware Works are the American associates of the Deutsche Ton-und Steingewerke of Charlottenburg, in conjunction with whom Dr. Meyer developed the designs of absorption apparatus described in that article.

The first **International Exposition of Safety and Sanitation** ever held in America will take place in New York City from December 11th to 23th, 1913, under the auspices of the American Museum of Safety, 29 West Thirty-ninth Street, New York City. This is the only museum of this kind in this country, while there are 21 museums of safety in Europe. By a special act of Congress, exhibits from Europe and other foreign countries are to be admitted free of duty.

For the **Anglo-American Exposition**, which will be held in London from May to October, 1914, the Philadelphia Commercial Museum will undertake the organization of the American Industrial Section.

The **Edison Storage Battery Company**, West Orange, New Jersey, for its new building, now being erected for the manufacture of storage batteries, have placed through the New York office of the Quigley Furnace & Foundry Company, Springfield, Mass., one of the largest contracts for accurate-temperature furnaces ever placed. This contract is for fifty-nine of the Quigley overfired accurate-temperature furnaces burning oil fuel. The Edison companies have used this same type of furnace in all of their plants in the past, for a series of years.

The **Tomboy Gold Mines Co.**, after operating two Hardinge conical mills for fourteen months, are installing two additional mills of the same type.

Volute Centrifugal Pumps.—Centrifugal pumps are divided into two general types—the volute pump and the turbine pump. The turbine pump is practically the converse of the water turbine, while the volute pump derives its name from the form of the casing, which is similar in shape to the involute curve. The Worthington catalog W-202, just issued by the International Steam Pump Co., 115 Broadway, New York, gives a profusely illustrated description of Worthington volute pumps. These are built in three standard designs—the single side suction type, the double suction solid case type, and the double suction pump with horizontally slit casing. The catalog also contains directions for installing, operating, and testing centrifugal pumps, and numerical tables useful in pumping practice.

Oxygen and Hydrogen Plants in Operation is the title of a little illustrated booklet showing the successful introduction of electrolytic oxygen and hydrogen plants of the I. O. C. system into quite a number of American works. The process was described in our Vol. IX, p. 471 (September, 1911), with additional notes in our Vol. X, p. 394 (July, 1912). The above pamphlet is published by the International Oxygen Company, 115 Broadway, New York City.

Protection of Iron and Steel is the title of a pamphlet issued by the Metal Treating & Equipment Co., U. S. Rubber Co. Bldg., New York. It deals with the "Standard" galvanizing process. This is an electrolytic galvanizing process, but only few technical details are given.

Recent Chemical and Metallurgical Patents

Gold and Silver

Rotary Vacuum Filter.—A rotary vacuum filter for slime pulp or for the separation of a valuable liquid from a finely divided solid has been patented by MR. THOMAS BREAKELL, of Wirksworth, England. The main object of the invention is to filter slime pulp continuously and at the same time provide means for keeping separate the valuable solution and the wash solutions. The invention is along the line of the well-known Oliver and Portland types of vacuum filters, but differs from them in being a closed drum in which a vacuum is maintained and into which the several solutions are drawn and collected in different compartments.

As shown in Figs. 1 and 2 the filter consists of a closed drum or barrel *a*, having a filtering surface *b* and arranged to rotate in a suitable frame. Within the barrel is a receptacle *d* divided by a vertical partition into two compartments *f* and *g*. A space *h* is left beneath the receptacle. On the inner periphery of the drum are arranged numerous ribs *l* which serve to divert the filtered fluid into the proper receptacles. A slime feed-box is placed at *n*, the lower edge of which, *m*, serves also as a knife to scrape the filtered slime from the face of the filter. Suitable pipes and distributing rollers for applying wash solutions are placed at *s* and *t*, and *q* and *u*. Pipes for withdrawing the filtered liquids under reduced pres-

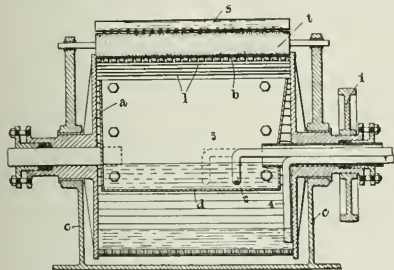


FIG 1—VACUUM FILTER

sure enter the drum through the bearing at one side of the drum, and extend into the several receptacles for solution. In the drawing these pipes are numbered 2, 3 and 4. Their outer ends are connected with a vacuum pump.

In operation the drum is rotated and the vacuum applied. Slime is delivered into the receptacle *n* and thus brought in contact with the filtering surface on the rising side of the drum on which it forms the usual cake. The solution passing through the filter is diverted by the inner ribs into the receptacle *f*. At the top of the drum a wash of barren solution may be applied from the pipe *s*, being distributed over the surface of the cake by means of the roller *t*. This solution, on passing through the filter, is diverted by the inner ribs into the receptacle *g*. Similarly, on the descending side of the drum, a wash of clear water may be applied from the pipe *q* within a gauze distributor *u*. Part of the solution filtered at this point may pass into the receptacle *g* and the balance into the space *h*. The several solutions are removed by the pipes 2, 3 and 4.

It is not apparent that this filter offers any advantage over the better known types of drum filter, either in the method of feeding the slime or in the manner of keeping the solution separate. The use of the rollers in connection with the wash solutions may be advantageous in getting a thorough distribution over the cake and in filling slight cracks that may appear. (1,064,702, June 17, 1913.)

Zinc

Electric Smelting of Zinc Ores.—In Fig. 3 is shown a longitudinal central section of a rotary furnace in which Mr. AUGUSTIN L. J. QUENEAU, of Philadelphia, Pa., proposes to

smelt zinc ores by means of electricity. In contrast to the standard method of smelting zinc ores the inventor provides a large furnace in which a comparatively large charge can be treated at one time. The advantages claimed for the process by the inventor are that partly roasted or even raw sulphide ores can be smelted; that ores containing iron, copper, gold and silver can be treated efficiently; that the furnace lining can be made of material more resistant to the action of slag than

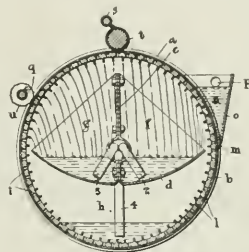


FIG 2—VACUUM FILTER

fire-clay; that ores not suitable to the present methods of smelting can be fluxed and treated by this process; and that preheated ore can be charged to the furnace, such preheating being accomplished, for instance, in the roasting kilns. Referring to Fig. 3, the furnace consists of a steel shell *k*, lined with an outer layer *a* of fire brick, and an inner layer *b* of chrome brick, the latter being inert to slags and zinc vapors. The furnace rotates on tires and rollers, being driven by band gear *f* and gear wheel *g*. Each end of the furnace is closed by a metallic plate *h* which is attached to the shell in such a way that it is held in elastic, adjustable contact with the end of the furnace and may adapt its position with expansion and contraction of the furnace. The end lining is made of a circular course of graphite bricks *n*, fire brick *p* and chrome brick *r*. A condenser *A* and prolong *B* at each end of the furnace are held in position by a casting *s*, and revolve with the furnace, being connected with the latter by means of a refractory connection *q*. The non-condensable gases issuing from the prolong pass up a flue *C*. The plates *h* are recessed at *x* to provide reliable electric contact with the ends of the carbon brick *n*. At the left end of the furnace this recess provides a path for a carbon or metallic brush *y* constituting one of the terminals of an electric circuit so that during revolution of the furnace the current will be transmitted to that end of the furnace and to the circular course of carbon bricks *n*. The other terminal of the circuit *z* makes contact with the metallic shell of the furnace. At the right end of the furnace the ring *j* is in electric contact with the shell and likewise with the plate *h* by means of the spring metal plates *o*.

The metal plates *h* at each end of the furnace are thus out of direct electric connection with each other, and are intended to be electrically connected by a molten liquid resistor

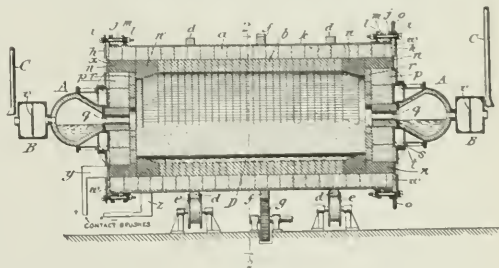


FIG 3—ROTARY ELECTRIC ZINC FURNACE

D which spans the non-conducting area between the plates. Molten cast iron of such composition as to have high specific electric resistance and high fluidity at relatively low temperatures, is used as such molten resistor.

In operation, the furnace having been preliminarily heated, a known amount of molten resistor is introduced after which the furnace is charged with preheated roasted ore and fuel. The furnace is then rotated slowly and the processes of reduc-

tion, distillation and condensation proceed. The passage of the electric current maintains the necessary thermal conditions. When the reduction is complete the furnace may be tapped of its slag and matte, if any be formed. The same aperture may be used for charging and tapping. The distilled zinc is condensed in the vessels *A*. The charge may be fluxed so as to produce a molten slag and copper-bearing material can be added to form a matte to gather the precious metals. (1,064,992, June 17, 1913.)

Iron and Steel

Melting Ferromanganese.—Messrs. Wilhelm Schemmann and Jegor Bronn of the Rombacher Huettenerwerke, Germany, are continuing their experimental electric furnace work with ferromanganese. Electric steel furnaces of various types have recently gained a new field of application, as they are being widely used in Europe to keep a liquid supply of ferromanganese ready at the converter plant, thereby saving in alloy and making the deoxidation more efficient than before. The direct heating of ferromanganese by the electric arc volatilizes too much metal as a brown smoke and can not be depended upon. The arc voltage amounts to 45 to 75 volts. The simple direct resistance heating was perfect as far as the metal

being a plan. Figs. 7 to 10 showing various forms of the anode, the last one to be used for treating sheet metal plates in vertical position. The corrugated form of the anode consisting of a plurality of channelled and perforated members, both upper and lower members, is a special feature of the design. An adjustable guide wheel and a well are provided at the center of the bath, and the electrolyte is pumped from the well to the extensions at opposite ends of the tank. (1,065,090, June 17, 1913.)

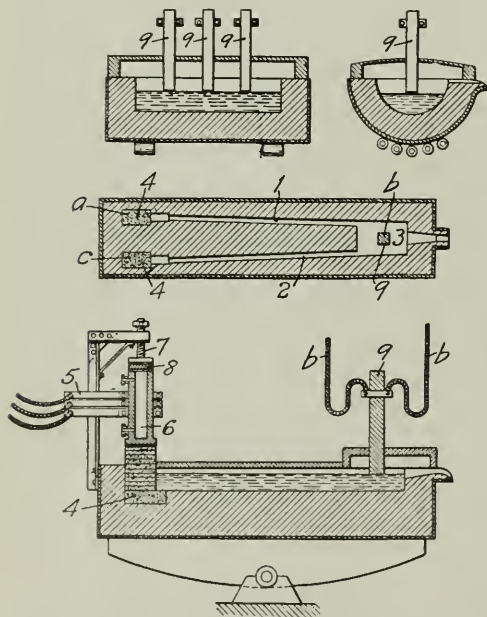
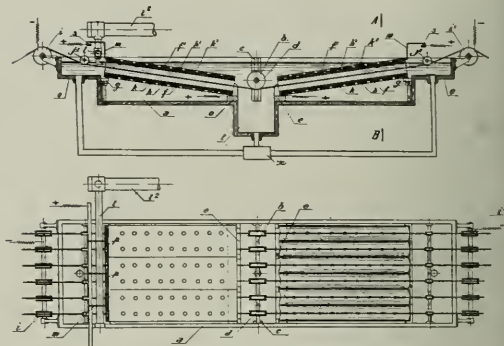


FIG. 4.—MELTING FERRO-MANGANESE BY A COMBINED ARC AND RESISTANCE METHOD (UPPER DIAGRAMS FOR THREE-PHASE CURRENTS, LOWER DIAGRAMS FOR SINGLE-PHASE)

was concerned, the slags however sometimes offered difficulties in removing. The two inventors, therefore, propose a combination process of arc and resistance heating shown in Fig. 4 for 3 or monophasic current. The carbon is lowered to such an extent that it just touches the surface of the bath and that the voltage drop between the carbon and the metal amounts to only from 16-18 volts. This enables one to heat the metal by resistance and the slag by the "semi-arc" method and thus saves considerably in carbons and metal. (1,061,016, May 6, 1913.)

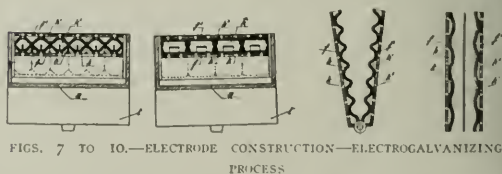
Continuous Galvanizing Process for Strips of Metal, Wire, etc.—Mr. Frederico Werth of Milan, Italy, has designed a new apparatus for a continuous electrogalvanizing process which will be readily understood from the drawings, Fig. 5 representing the electrolytic tank in longitudinal section, Fig. 6



FIGS. 5 AND 6.—CONTINUOUS ELECTROGALVANIZING PROCESS

Cupola Furnace.—Mr. William Newton Best of New York introduces what seems to be a novel feature in cupola melting by using crude oil or any other liquid hydrocarbon as fuel instead of coal or coke. Figs. 11 to 13 are self-explanatory, showing a bath furnace combined with the cupola like a fore-hearth. The flame passes from the combustion chamber through the bath furnace and then upwardly through the cupola. The metal may be tapped directly from the cupola into a ladle if cast iron is desired. If dense iron is to be made, the metal can flow into the hearth and accumulate and be thoroughly mixed by poling. (1,061,158, May 6, 1913.)

Regenerator or Hot Blast Stove.—Messrs. Max Schroeder of Berlin and Hugo Reinhard of Oberhausen, Germany, describe several designs of vertical regenerators filled with refractory material through which air or gas are forced by a compressor. Their invention permits reducing the temperature of the waste gases to below 100 deg. and even as low as 50 deg. Centigrade and allows to attain the highest possible furnace temperatures, whereas the thermal efficiency of the regenerators now in use is only about 50 per cent and the gases leave at a temperature averaging between 300 deg. and 500 deg. Fig. 14 illustrates in vertical section an arrangement for heating the air blast only. This is to be employed for gases of high calorific value. The arrangement as per Fig. 15 is intended for poorer gases, where both air and gas must be preheated. Fig. 16 illustrates the application of the inven-



FIGS. 7 TO 10.—ELECTRODE CONSTRUCTION—ELECTROGALVANIZING PROCESS

tion to the heating of blast for blast furnaces. With these purposes in mind the drawings can be easily understood. The letter *c* always designates a blower or compressor for fuel gas, *re* one for air and *x* an exhaustor for waste gases. (1,062,122, May 20, 1913.)

Making Pig Iron from Steel Scrap.—In view of the fact that on the Pacific Coast the selling price of pig iron during the past 10 years has ranged from \$20 to \$32 per ton, a simple

method of converting steel scrap back into pig iron from which it was made may be of interest to these districts where scrap, fuel oil and carbon are plentiful. Mr. Horace W. Lash of Cleveland, Ohio, melts for this purpose scrap, carbon, and silicon in any suitable type of furnace, preferably in a reverberatory furnace. The melt is drawn off in the usual

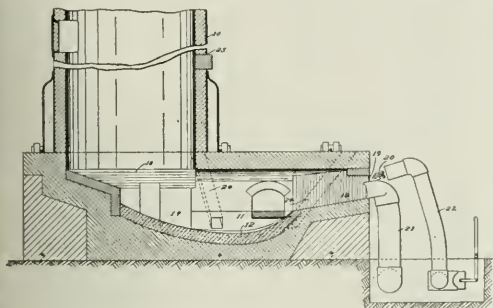


FIG. 11—CUPOLA DESIGN

manner and cast into pigs. It is more economical to obtain the silicon required from that contained in the mixture than to add ferrosilicon or siliceous pig iron. The major portion of the carbon charged should be placed in or near the bottom

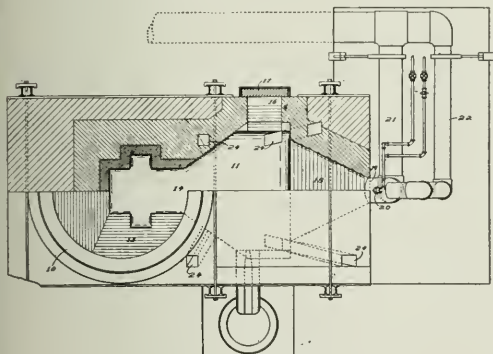


FIG. 12—CUPOLA DESIGN

of the furnace, where it is protected from the oxidizing flame. (1,061,189, May 6, 1913.)

Method of Stripping Sheets.—Mr. Joseph McFetridge of Vandergrift, Pa., describes a method of stripping defective galvanized sheets in preparing them for regalvanizing and

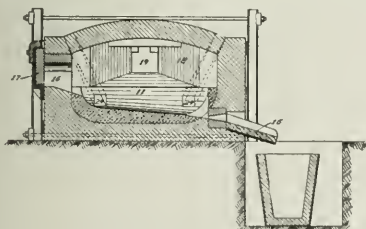
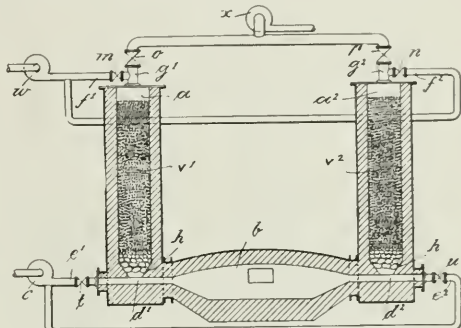


FIG. 13—CUPOLA DESIGN

recovering the removed zinc. Heretofore the practice had been to place the sheets in a solution of ferrous salts and free acid and to keep them in this bath until the zinc and zinc-iron alloy is dissolved and the surface of the base material uncovered. This results in a moderately dilute solution of zinc salts high in ferrous salts and the solution has no commercial value.

The present inventor, instead, provides two baths, in the first of which he removes the greater part of the zinc and zinc-iron alloy. In the second bath the operation is completed. The first bath contains a zinc salt and from 2 to 3 per cent free acid. After it has been neutralized to an acid content of between $\frac{1}{2}$ and 1 per cent portions of the first solution



FIGS. 14 AND 15—REGENERATOR DESIGN

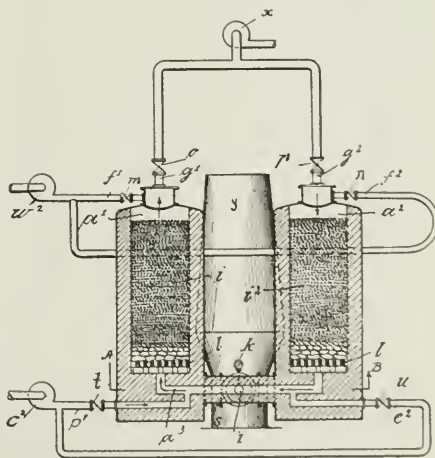


FIG. 16—HOT-BLAST STOVE

are removed to form the second bath and acid is added to the first bath to restore its original strength. The solution of the second bath, which is distributed in two tanks, can be elevated into a container from which it trickles or flows downwardly through a tower packed with quartz. The lower end of the tower is provided with an inlet through which a supply

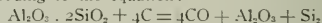
of oxidizing gas, such as chlorine gas, can rise upwardly through the quartz and oxidize the iron into the ferric form. In special precipitating tanks the liquor is treated with zinc carbonate. The pure zinc solution is decanted and filtered and finally delivered into storage tanks. It can thus be utilized commercially and is not allowed to go to waste. (1,063,054, May 27, 1913.)

Various Electrochemical Processes

Production of Ammonium Compounds.—Dr. Charles P. Steinmetz of the General Electric Company has devised a process of producing ammonium compounds by passing a mixture of steam and air or nitrogen through an electric arc. According to the proportion of the gases employed varying

may be built with a melting chamber in the center and refining chambers at both sides. The heating current is supplied at low voltage through electrodes, 4b and 5b, mounted in two opposite sidewalls of the melting chamber. The path of the current being across the chamber is correspondingly short. The furnace is started with a mass of molten glass poured into the melting chamber, or by the aid of an arc. If necessary, the refining chamber can be kept warm by indirect resistance heating by means of register 19. (1,062,362, May 20, 1913.)

Process of Producing Elementary Silicon and By-Products.—Mr. Florentine J. Machalske of Philadelphia, Pa., claims the following peculiar process of producing metallic silicon from silicates. Carbon is mixed with the siliceous compound in molecular proportion to such an extent that there is but sufficient carbon to combine with the oxygen of the silica only. If this mixture is treated in a commercial electric furnace, preferably of the resistance type, the silicon is supposed to be reduced to the elementary state and the other component part of the silicate is not decomposed. A furnace of about 100 kw with a variance of voltage from 20-50 at 2000-5000 amperes is used. To start the reaction about 1 per cent of sodium chloride and sawdust each are mixed with the furnace charge. The process is described in detail for anhydrous kaolin according to the equation:



It is said it can also be used for ores such as tungsten silicate when metallic silicon is obtained with tungstous oxides as by-product. In order to separate the products after fusion in the furnace they are directly subjected to centrifugal action while allowing the current to pass at a voltage somewhat less than that used during the electrothermic fusion. (1,062,682, May 27, 1913.)

Purifying Carbon for Carbon Filaments.—Dr. Jean Billiter of Vienna, Austria, known by his research work on alkali electrolysis, sets forth that the falling to dust of the ordinary carbon filament in lamps is mainly caused by impurities which are contained in all known carbon or graphite filaments. Almost all such impurities will assume a gaseous form at and above 2000 deg. Centigrade and these gases will gradually disintegrate the filament, even if the latter has been

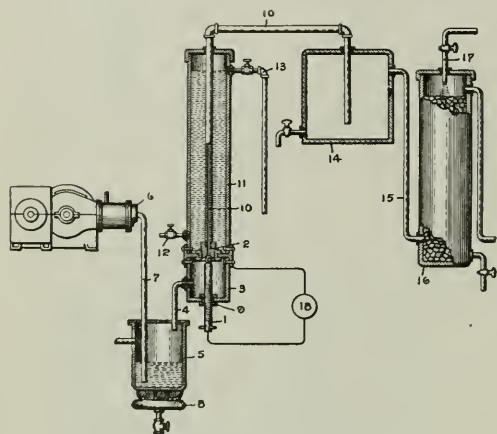
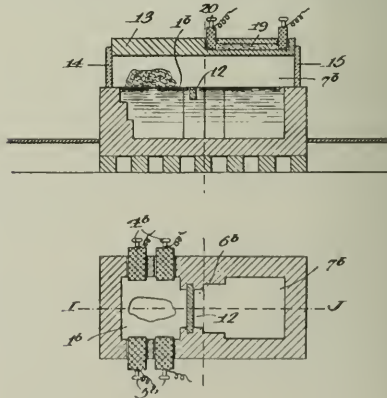


FIG. 17—PRODUCTION OF AMMONIUM COMPOUNDS AND NITRIC AND NITROUS ACIDS

proportions of ammonia and nitrous or nitric acids are formed. The proportion of nitrate to nitrite varies with the relative amount of oxygen and the length of time the oxygen is in contact with the compounds which are formed. The process is best described by following the apparatus shown in Fig. 17: The air, or nitrogen, supplied by a pump 6 carries with it steam from the tank 5, the amount being regulated by gas burner 8. The mixture of air and steam passes into the pressure chamber 3, and from thence through an arc formed between electrodes 1 and 2. The products of the reaction are cooled below the temperature at which the reverse reaction can take place by contact with the cooled walls of the tube 10. In the reaction chamber 14 oxygen combines with the nitric oxide to form nitric dioxide, which with water gives nitric and nitrous acids. These acids in turn combine with ammonia to form ammonium nitrate and nitrite. Some of the ammonium nitrite formed by a combination of the ammonia and the nitrous acid in the tube 10 and in the chamber 14 is oxidized to form ammonium nitrate. The watery vapor unacted upon is condensed in the chamber 14. The uncombined nitric and nitrous acids, or ammonia, pass over into the absorption tank 16, where they are either dissolved in water to form nitric and nitrous acids, or ammonium hydrate, as the case may be. Uncombined acids combine with the lime or other reaction material to form nitrate or nitrite of calcium or of other metal. (1,062,805, May 27, 1913.)

Electric Furnace for the Continuous Manufacture of Glass.—The old problem of melting glass in the electric furnace is taken up again by Mr. Marius Sauvageon of Colombes, France. In pursuit of his older patents of 1910 he describes a furnace made up of a melting and refining division and a working chamber. Both are separated from each other by the floating dam 12 shown in the vertical section, Fig. 18, and in the horizontal section, Fig. 19. Of course, a furnace



FIGS. 18 AND 19—ELECTRIC GLASS FURNACE

preliminarily heated up to 3000 deg. Centigrade. The filaments made from the purified carbon of Dr. Billiter's invention contain less than 0.1 per cent. impurities and at least 94 per cent of the carbon in the form of graphite. The essential feature of his purification process is to transfer the carbon or graphite, which has been purified as highly as possible, into a colloidal form (in which they will remain in suspension for days or weeks without precipitating). It is then separated by suction or "ultra filtration" from a great part of impurities which

do not assume a colloidal form. To produce the colloid the material is, for instance, heated by a current until it disintegrates to dust. This process must generally be repeated and is aided by a treatment with fluorine. Thus the carbon filament may be anodically polarized in hydrofluoric acid. (1,062,431, May 20, 1913.)

Synopsis of Recent Chemical and Metallurgical Literature

Lead and Zinc

Electrolytic Lead Refining.—In a paper read before the Western Branch of the Canadian Mining Institute, Mr. JOHN F. MILLER, superintendent of the electrolytic lead refinery of the Consolidated Mining & Smelting Co., of Trail, B. C., described the process in use at that place.

Base lead bullion produced in blast furnaces contains about 97 per cent lead and 3 per cent of gold, silver, antimony, arsenic, copper, zinc and traces of other metals. It is cast into anode bars weighing about 400 lb. each and loaded into anode cars containing 10 anodes each. Twenty such cars are loaded onto a railway car and transported to the refinery. Here an electric crane unloads the anode cars and carries 20 anodes at a time to the tank room. The tanks are 30 in. wide, 44 in. deep and 7 ft. 4 in. long, holding 20 anodes spaced $\frac{1}{4}$ in. centers. Cathode starting sheets of pure lead are placed between the anodes. The electrolyte contains 12 per cent hydrofluosilicic acid and 6 per cent lead fluosilicate. This is circulated down the cascade of tanks at the rate of 35 cu. ft. per hour. A continuous electric current is passed for five days, at the end of which time the cathodes are removed and replaced with another set of starting sheets. When the second set of cathodes is formed and removed, the anode scrap amounts to about 15 per cent of the original anode weight and is removed to a melting pot to be again cast into anodes. The refined anodes are melted and cast into molds of 5, 10, 100 or 200 lb. weight, according to requirements.

The anode slime containing gold, silver, copper, antimony and arsenic, is washed, dried, melted and cast into doré bars, the impurities being oxidized and slagged. The doré bars are parted by sulphuric acid; the gold sludge is collected and melted, producing gold bars 995 fine; the silver sulphate solution is precipitated by boiling with copper, producing metallic silver and copper sulphate. The resulting silver bars are 999 fine. The copper sulphate solution is crystallized and the resulting blue vitriol sold for use in agricultural and horticultural industries. Formerly the antimony and arsenic contained in the slime were recovered, but owing to the lower percentage of these metals now in the slime and the lower market price, their recovery has been abandoned. The plant is so arranged that the recovery of antimony can be resumed when conditions warrant.

The method adopted for recovery of antimony is to boil the slime with sodium polysulphide which extracts from 80 to 90 per cent of the antimony and 50 per cent of the arsenic. The solution is filtered and electrolyzed in iron tanks containing lead sheet anodes and steel sheet cathodes, using a cathode density of 8 amp per square foot, with a normal voltage of 1.5 per tank. The antimony forms hard, dense deposits on the steel cathodes and is removed from time to time. It contains about 2 per cent arsenic which is removed by alkali fluxes when the metal is melted for casting. The electrolyte is concentrated in steam-coil evaporators and used again, with the addition of a little more sulphur for the treatment of a new batch of slime.

After treatment for removal of antimony the slime is treated with 10 per cent H_2SO_4 to remove copper and silver which are then recovered as blue vitriol and metallic silver. The residue is then melted in an oxidizing atmosphere with a little coal or silica to break up $PbSO_4$, resulting in doré bullion which is then parted and treated as described above.

Returning to the refined lead: it contains no arsenic or

bismuth and the following percentages of other metals, zinc, 0.0005; silver, 0.0013; copper, 0.00075; iron, 0.00075; tin, 0.0001; antimony, 0.0028; lead, 99.9938.

The electrolyte contains an addition agent in the form of glue, which is added in the proportion of $\frac{1}{2}$ to 1 lb. per ton of lead produced. While the action of the glue is not wholly understood, it prevents the formation of soft, spongy and incoherent deposits of cathode lead. In fact, it would be impossible to operate without it, as the tanks would all become short-circuited.

Early Example of Blast-Roasting.—A primitive form of the application of modern blast-roasting of sulphide ores is described by Mr. H. VASSILIADIS in *Bulletin No. 104, Inst. Min. & Met.* In 1895 the author had charge of the metallurgical department of the Balia mines in Asia Minor. The ore was complex, containing galena, blende and pyrite in a calcareous gangue. A notable proportion of the lead saved on Harz jigs and tables was in the form of middlings, assaying as follows: Lead, 19.15 per cent; silica, 7.47; iron, 23.67; lime, 0.80; zinc, 6.50; sulphur, 35.13.

These middlings were roasted and agglomerated in so-called Pelatan furnaces, introduced from Laurium, Greece. Each furnace was 2.9 m long, 0.9 m deep and 1 m high. The sides, back and bottom were cast iron, but the front was water-jacketed sheet iron, removable to facilitate the subsequent manipulation of the roasted cake. The furnaces were built in pairs with a common flue. A hinged lid completed the arrangement. A foot from the bottom was a cast-iron grate of the same dimensions as the horizontal section of the furnace. Three rockers in the form of an arc of a circle were bolted to the bottom of the grate and rested directly on the bottom of the furnace. Air could be blown under the grate through an opening in the floor connecting with an air main.

After the grate and front had been placed in position and all joints luted with clay, a layer of twigs and brush was placed on the grate and over this a 6-in. layer of ore. The wood was lighted and a low-pressure blast, about 5 cm water, was applied. As soon as the ore had ignited more damped middlings were added until the furnace was full. The blast was increased as required until it reached a maximum of 15 to 20 cm water pressure. When fumes ceased issuing from the furnace and the contents had cooled the front was removed and the charge dumped from the grate. It was broken with bars and sledges and the fine material used as the lowest layer of the next charge, while the balance was fed to the blast furnace.

Later Malatra furnaces were added to give the middlings a preliminary roast and thus produce a material containing only about 10 per cent sulphur for final agglomeration in the Pelatan furnaces. This worked very satisfactorily and greatly increased the capacity of the agglomerating plant. The time for agglomeration was reduced from fifteen or eighteen hours to eight, and the troublesome fusion and agglomeration without previous desulphurizing were avoided. The following figures show the improvement in the final product made by preliminary desulphurizing.

	Per cent Pelatan Product	Per cent Malatra-Pelatan Product
Lead	20.25	27.60
Silica	14.50	11.34
Sulphur	17.05	2.20
Iron	27.00	30.00
Lime	1.10	1.20

The plant described has long been discarded but is interesting as an early example of blast-roasting.

Electric Zinc Smelting and Chemical Manufacturing.—From the *Australian Mining Standard*, May 22, 1913, we learn that the Sulphide Corporation has constructed a plant at Cockle Creek, New South Wales, for the electric smelting of zinc ores, combined with the manufacture of sulphuric acid and superphosphate. The plant was supposed to be ready for operation last June but no later information is at hand.

Prior to erecting the 500-hp electric furnace at Cockle Creek the Sulphide Corporation had made extensive experiments on ore shipped to Sweden where the metallurgical results were apparently satisfactory. The power problem was, however, entirely different in Australia and the results obtained in Sweden could not be taken as indicative of what could be done at home. It has been decided to see whether power cannot be cheaply produced by burning gas from Mond producers, recovering ammonium sulphate as a by-product and thus reduce the cost of power. In order to utilize the sulphur now wasted from the roasting furnaces a chamber acid plant is erected to make sulphuric acid, part of which will be used to fix the ammonia from the producer gas and the balance for the manufacture of superphosphate. A large plant for this latter process is in course of erection. By this combination of metallurgical and chemical engineering it is hoped that an economical plant will result.

Principles of Mill Design

"It is vitally important to plan a mill before drawing the plans. This may seem a paradoxical aphorism, but it is not. The most thorough thinking must be done when nothing yet exists. It is necessary to decide the type of mill best suited for the treatment of the ore, where to build it, how to dispose of the general arrangement of the plant and even to investigate carefully whether there is any need for a mill at all."

With the foregoing introductory paragraph, Mr. GELASIO CAETANI opens an interesting discussion of the general principles of mill design, in the June, 1913, issue of the *Mining Magazine*.

The importance of preliminary metallurgical tests is first emphasized. Formerly it was common practice to build a mill of standard design and then adapt it to the special requirements of the ore as determined in operation. This is no longer countenanced and the larger companies even construct experimental mills of considerable capacity in order to test the ore, while smaller operators should always avail themselves of the facilities offered by testing plants which can be found in all large cities.

After settling the metallurgical requirements it becomes necessary to choose the site and determine the general arrangement of the plant. Too much time and effort cannot be spent on the design of the mill for it is so easy to shift the plant around at this stage of the game. "One day spent in thinking may save a fortune in operating expenses." After calling attention to the danger of falling in love with one's own plans the author states that he believes all plans should be drawn three times: first to crystallize the idea, second to determine the structural defects, and third to perfect the solution of the problem. The idea is the main thing—if the idea is wrong the plan is useless and can be destroyed.

In designing an ore-dressing mill the author believes that it is economy, especially in large projects, to separate the crushing and treatment plants. If the crushing department is in the same building as the ore-dressing department, ore bins and some of the heavy machinery must be placed on top of the mill, thus requiring heavy substructures and reducing the capacity of the bins. Further, the handling of dry ore is cheaper than the handling of wet ore, and as much as possible of the preliminary preparation should be dry. This brings up the necessity of keeping dust-forming operations away from the ore-dressing plant and suggests separate buildings for the two processes. Carrying the idea further, stamps might well be in a separate building and amalgamation also could be conducted as a separate department, after the practice in South Africa.

The segregation of a plant in separate buildings is not wholly advantageous; there are great advantages in having as much as possible of the machinery under one roof. The Miami mill is cited as an example of fine construction of this type.

Mr. Caetani also touches on the aesthetic principles in mill construction, calling attention to the desirability of maintaining harmony and proportion in the design. "A thing that is ex-

cellently adapted to the purpose for which it was designed, is beautiful; therefore proportions, general arrangement, facility and economy of operations, good illumination, and so forth, are all features that contribute toward efficiency and so toward beauty."

In considering materials of construction, the author calls attention to the use of concrete in all masonry work, and especially in foundations for buildings and machinery. Floors, jigs, bins, settling tanks, etc., also have been constructed of this material, but its use in this connection is not desirable if many changes are to be made. Concrete construction has long life, but is not plastic, easily moved or modified, and, therefore, should not be used unless the work is of a permanent nature.

Steel is being more widely used for structures, and has advantages in not warping or sagging, and in the matter of fire protection. Nevertheless wood is the more common material of construction, owing to the fact that it can be framed and erected so much easier than steel. It should be preferred in small plants, or in places where changes in construction are likely. A combination of steel and wood construction offers many advantages in designing elegant structures.

The author believes that a hillside is generally the best site that can be chosen for a mill, as the ore can flow by gravity from one department to another. "It is a mistake, however, to believe that no hillside mills require any re-elevation of the ore." Where re-elevation becomes necessary, it is better to make one re-elevation to a considerable height than several at different times and places in the mill. Operations should be concentrated, not only with regard to pumps and elevators, but with other machinery in the mill. Machines that have an organic part in the mill operation should be in duplicate, if possible, and hence it becomes necessary to have as few of these as possible. All machines of one kind should be arranged on one floor, if possible, as this arrangement reduces labor and increases efficiency. Main alley-ways for the handling of large pieces of machinery, and cranes to serve lines of heavy machines are almost indispensable. Classifiers, launders, tanks, and other containers that are likely to overflow at some time or other should not be placed over machinery that will be damaged by such overflow of pulp or water.

One of the most important problems to be determined is the circulating system. The tonnages of the various products and materials must be carefully calculated, and ample provision made for the extreme quantity that may have to be handled. The normal load must be taken as the minimum load, and the system must be made very elastic to take care of unusual conditions. The mill feed may fluctuate considerably, and in closed circuits the volume of material to be handled may suddenly be greatly increased due to the inefficient performance of some machine. Thus trommels and rolls may accumulate an idle circulation of ore which overloads both and causes inefficient results. This is a condition difficult to remedy, and it should not be allowed to occur, or the system should be arranged to relieve temporarily the load on the machines. Elasticity in design thus becomes a very important feature.

The Consolidated Mercur Gold Mines Co., Mercur, Utah, recently ceased operations after a career full of profit and progress. Accompanying checks for a 3-cent dividend, the company sent the following announcement of the closing scenes at the mine. "The last skip of ore was hoisted at 8.40 a. m. Sunday, March 30, 1913, and the Mercur mine was at an end. An hour before, Joe Sullivan, one of our oldest miners, loaded the last car in the mine. The car was decorated with bunting and flags, attached to the last train, hauled out by the motor, and dumped in the Mercur pockets. As the last skip was being hoisted, the flag was raised on the mill, the whistles were blown for one hour, the church bell, school bell and fire bells were rung and the famous old producer passed into history with due ceremony."

The production of crude barytes in the U. S. in 1912, according to J. M. Hill, of the U. S. Geological Survey, was 37,478 short tons, valued \$153,313.

Notes on Chemistry and Metallurgy in Great Britain

(From *Our Special Correspondent*.)

Iron and Steel Institute

In the continuation of the discussion on the paper of Dr. W. Rosenhain and Mr. J. C. Humfrey on the "Tenacity and Fracture of Soft Steel," the first part of which was reported in the last issue, Professor J. O. Arnold went on to say that while he sincerely congratulated the authors upon a most valuable research, it appeared that they found out that there was something unsatisfactory in connection with their electrolytic iron and took for their experiments a tin-plate steel containing about 0.11 per cent carbon and 0.4 manganese. The authors said that the curves of absorption and recalescence were taken with a special apparatus, and that uniform rates of heating and cooling were maintained, while the observations were taken by means of a thermocouple and the delicate and accurate potentiometer of their laboratory. But the curve shown on the screen hardly confirmed the optimistic view of the capabilities of their apparatus expounded by the authors. The adjoining curve depicted an observation typical of many observations by himself and Mr. T. Swinden with the recalescence installation at Sheffield University in 1906, from a steel containing about 0.22 per cent carbon and 0.05 per cent manganese.

Nevertheless the authors' curve did in a way show the fourth recalescence of steel, the discovery of which he had announced to the Chemical Section of the British Association in 1910. He had added the dotted lines to the authors' curve to show this. But Dr. Rosenhain had not managed to split up the Ar₂ into its constituent peaks, and appeared to be laboring under the curious misapprehension that the fourth recalescence of steel was the second peak of Ar₂, which he, nevertheless, scornfully described as a "wobble." This led him (the speaker) to ask the conundrum, "When is a wobble not a wobble?" And the answer was, "When it occurs regularly in scores of observations." The fourth recalescence was shown in the right-hand curve recorded as a prolonged evolution of heat, which kept the observed curve between Ar₃ and Ar₁, to the right of the dotted radiation line. This evolution of heat was due to the falling out, after Ar₃, of the hardenite previously existing in solid solution in the gamma range of temperature.

In 1909 the authors stated as a definite fact that beta iron was sufficiently hard to show no micrographic evidence of plastic strain under a stress sufficient to disrupt the alpha iron, and this in spite of the hardening effect of the lower temperature. In 1913, after making elaborate experiments, they proved that beta iron in spite of that same hardening influence gave the minimum stress recorded in all the three ranges of temperature; and in the middle of the beta range gave the startling elongation of nearly 13 per cent. Then they said in effect that their second research to a considerable extent confirmed the conclusions of the first, but admitted that certain modifications might be necessary.

Modifications, indeed! If in practice they got a steel casting which consisted mostly of gas blow-holes surrounded by steel, they "modified" that casting by throwing it on the scrap-heap. It was his opinion that the authors' unwisely veiled apologia subconsciously trifled with the institution.

There was abundant evidence in the paper that the authors were quite at a loss to account for the unexpected fact that in the beta range the iron seemed to be so soft as to give minimum stress and maximum strain: and, therefore, on the erroneous supposition that they were dealing with results obtained from chemically pure iron and not from their dead mild steel, proceeded to elaborate a theory ascribing the disconcerting fact to the influences of the dimensions of the crystals and of a hypothetical amorphous cement existing between those crystals.

Their valuable observations provisionally showed that the

mechanical hardness of iron in the lower gamma, the beta and the higher alpha ranges of temperature was approximately inversely proportional to the temperature, and was not in any way connected with the Ar₃ and Ar₂ points; but this, the real explanation, had completely escaped their attention. The slide on the screen showed the condition of the micro-chemical constitution of the authors' specimens of mild steel, and this was shown diagrammatically with substantial accuracy in three typical examples. In the gamma range at 924 deg. C. the 12 per cent of slightly manganiferous hardenite (the existence of which had been entirely ignored by the authors) was evenly diffused in solid solution throughout the masses, thus hardening the iron in spite of the softening effect of a high temperature. After the Ar₃ point, coming down the beta range to 836 deg. C., the hardenite had to a great extent fallen out of solution and evolved the fourth recalescence, leaving a ground mass of bright blood-red iron surrounding the masses of ferritic hardenite which constituted about 25 per cent of the area. The tensile strength of the mass now fell and the elongation increased, because stress was now mainly applied to iron without hardenite. Then under the hardening influence of lower temperature in the alpha range the stress increased again and the elongation became lower.

He sincerely thanked the authors for their able experimental proof of the accuracy of the teaching of the physical chemistry of mild steel which for so long had been carried out at Sheffield University. Although the authors maintained that the question of hard beta iron was still open he submitted that it was closed for ever, and that hard beta iron was as dead as the Doges. Looking back for some twenty years and reviewing the evil effect upon the true scientific progress of our knowledge of the physical chemistry of steel brought about by the acres of pseudo-scientific garbage written about the beta iron hypothesis he was inclined to use strong language: but he remembered that this hypothesis was now among the things that had gone, and he would therefore only say of the beta iron theory, "Peace to its ashes."

In the Proceedings of the Royal Society, A. Vol. 83, page 208, in a paper by Messrs. Rosenhain and Humfrey, November 25th, 1909, the statement was made:—"We have the striking fact that, as a result of the α/β transformation, the iron becomes stronger and harder in spite of the tendency of increasing temperatures to reverse this relation. The fact that in the β iron, near the transition temperature, there are very few signs of slip or disturbance of any kind further shows that not only has this material a greater tensile strength, but that it is also harder and has undergone practically no plastic deformation at all under a stress sufficient to rupture an equal section of α iron."

In the Proceedings of the Iron and Steel Institute, in the paper read by Messrs. Rosenhain and Humfrey, on May 2nd, 1913, the following figures were given in Table IV. Rate of extension 0.0005 in. per second per in.

Temp. °.	Range.	Max. stress	
		lb. per sq. in.	Elongation.
924	Gamma	5,350	72
836	Beta	2,780	130
686	Alpha	7,500	72

"In comparing these conclusions with the deductions drawn by the present authors from their earlier experiments described in the paper referred to above, it will be seen that, although in general terms the quantitative measurements confirm the result of the earlier qualitative work, they yet required some modification of the earlier conclusions."

Professor H. C. H. Carpenter submitted that the paper was tantamount to an admission that the authors' interpretation of their paper before the Royal Society was wrong. In criticizing the table of data for stresses, he said that the evidence of change in passing from alpha to beta consisted of results several hundreds per cent outside the range of the experimental error; and, with regard to the bottom curve—when the rate of stressing was very slow—the authors had actually had to invent the discontinuity drawn there. If the curve had

been drawn through the ascertained points it would have been quite smooth throughout the beta range. They seemed to be quite blind to the evidence of this fact, and were so wedded to the beta theory that they must actually invent that kink in the curve. This was entirely unjustifiable; and they should have paused before drawing such a conclusion on such microscopically slender evidence. The authors found it difficult to reconcile this "discontinuity" with Benedicks' theory which regards beta iron as a solution of gamma iron and alpha iron; but they were not dealing with pure iron, and he maintained that their results were exactly in accordance with what might be expected if Benedicks' theory were true.

Sir Robert Hadfield, after explaining why he had abandoned the allotropic theory, remarked that, although the authors said they regarded their earlier experiments as of a preliminary character, it was unfortunate that others had not been allowed to take this view, and the authors' previous results had been quoted repeatedly as proof of the existence of hard beta iron. They had, in fact, lately been the basis of the beta theory, and it seemed to him that the authors could not persist in believing that such a form of iron exists. They had said that the heat relief developed in the beta range is identical with the alpha structure found by etching the cold specimen, and this fact seemed to prove that the authors' alpha and beta iron were the same thing, and he thought the authors should be satisfied that their previous theories were not tenable.

Mr. J. C. Humfrey, in replying, said that Professor Carpenter had quoted figures out of the tables at the end of the paper with reference to the beta change point. As a rule it was very unfair to quote figures of one result in a research like this, which included some 200 experiments. It was more important to see the actual shape of the curve than to select one or two points here and there.

Dr. Walter Rosenhain in his reply said the discussion had been rather extraordinary. First, there had been an impassioned oration from his Sheffield friend, which he might divide into three parts—first, complimentary, which he appreciated highly; secondly, vituperative, which he neglected; and, thirdly, and this only a small proportion, definite argument, which he proposed to deal with. Professor Arnold had sinned in even a worse degree than Professor Carpenter, in picking out isolated sentences in a most misleading way, and endeavoring to hold up the authors to derision; and he would ask his audience to read their paper on its merits, and also the remarks of Professor Arnold on their merits, and he would be content with their judgment. A serious argument Professor Arnold had made was the suggestion that the authors had ignored the influence of 0.1 per cent of carbon, or, at all events, that they underrated it. That might be possible, but they found as a matter of fact in their experiments on electrolytic iron just the same kind of changes of strength and structure as in the particular variety of steel described in the paper, and this seemed to justify the tacit assumption that the influence of 0.1 per cent of carbon at the temperatures in question was very small. With regard to Professor Carpenter's remarks, he considered that his defense of the Benedicks' theory was overdone; and when he suggested that the authors had invented the discontinuity his criticism had gone too far altogether. If they looked at the points plotted on the curve they would see that it was not possible to really get rid of all this discontinuity at A₂ except at the lower curve, and then they would have to invent continuity, not discontinuity. Many other points which had been raised he would reserve and deal with in writing.

The last paper taken was on "The Critical Ranges of Pure Iron" by Professor H. C. H. Carpenter, whose conclusions were that (1) A₂ as an independent critical point or range does not exist (2) A₂ as a critical point or range cannot be detected on a heating curve when the bulk of the dissolved hydrogen has been removed. There are traces of a retardation in the rate of heating spread over a considerable temperature interval, which might be expected to include A₂

within its limits if it existed. (3) A₂ (which is evident on all the cooling curves taken) is the temperature at which gamma iron ceases to be merely metastable and becomes unstable. (4) Of the two values obtained for A₃—viz.: 926 deg. and 916 deg.—the latter is the more probable, because it corresponds within 1 deg. to the temperature at which the iron recrystallizes on heating. (5) The mean temperature of the A₃ peak is 888 deg. C.

The discussion was opened by Dr. Rosenhain, who said that he could not accept Benedicks' theory—much as he would like to—because it was a simplification. His experience of heating and cooling curves of electrolytic iron and nearly all carbon alloys of iron agreed with Professor Carpenter's and Dr. Arnold's, and the whole question was how much had A₂ to do with the gases in the steel?

Sir Robert Hadfield referred to the experiments carried out by O'Shea with electrolytic iron because it was that material which first led him to alter his conviction against the existence of beta iron. Professor Roberts Austen was then claiming that electrolytic iron proved the existence of beta adamantine formation, but when he (the speaker) obtained specimens he could not agree with that conclusion and entirely abandoned any belief in allotropic and adamantine formation of iron. In conclusion he said that the Institute was that morning making history in the Metallurgical World, and was now starting a theory which would do away with the great waste of time they had experienced in the past, and in future they would examine steel on a new basis and work on the behavior and properties of crystalline structures. The old theory had now been proved to be extinct.

Engineering Imports and Exports

The returns issued by the Board of Trade for the five months ended on the 31st may compare very favorably with the figures for the corresponding period of 1912—the only decrease being a small one in electrical goods. Imports of iron and steel, including manufactures, amounted to £6,412,419, an increase of £1,657,547; and exports totaled £23,381,966, an increase of £5,254,172. Imports of other metals, including manufactures, went up to £13,906,499, an increase of £1,855,268; while exports reached £5,806,333, an improvement of £1,161,676. Electrical goods were imported to the value of £624,868, a decrease of £6,091; but exports rose to £2,381,357, showing an increase of £636,603. In machinery the imports were £3,162,493, a rise of £278,970; and exports amounted to £15,190,633, an increase of £2,060,971. Imports of new ships are valued at £13,922, and the exports at £3,370,460, showing increases of £675 and £1,051,601. The figures for the month of May alone also indicate a satisfactory state of trade, and exhibit increases in all classes except imports of machinery.

Market Prices

June, 1913.

Tin opened £212.10, dropped rapidly to £205.15 (10th), recovered to £209 (13th) and fell again to £203.15 (17th), since when it has been about this level or under, showing £193.10 on the 23rd, and closing at £194.

Copper opened £66.15 and showed unsteadiness, dropping £1 in the first ten days, and has not since recovered, declining to £62.17.6 on 23d, but closed better at £64 to £64.5.

Hematite opened 77 9 and fell away to 73/- on the 13th, which price it has kept to the end of the month.

Scotch Pig opened 66/6, declined to 65/- at mid-month and remains at that figure.

Cleveland opened 66/6 and dropped to 54/6, remaining in the neighborhood of that price and closing at 55 1/4d.

Lead opened £20.5, has ruled steady and stronger in the third week, touching £21.50 on the 19th, then fell away to £19.15, top price, on the 30th.

"The Solution of the Worst Boiler Problem" is the title of a little recent pamphlet of the Yarnall-Waring Co., Chestnut Hill, Philadelphia, on their "simplex seatless blow-off valve."

	£	s.	d.
Aluminum, per ton.....	90.	0.	0
Alum, lump, loose, per ton.....	6.	0.	0
Antimony, black sulphide powder, per ton.....	20.	0.	0
Borax (crystal), refined, cwt.....	18.	6	
Copper ore, 10 to 25%, unit.....	11/3		11.9
Copper sulphate, ton.....	22.	5.	0
Carbolic acid, liquid 97-99%, gal.....	1.	4	
Caustic soda, 70%, ton.....	9.12	6	
Ebonite, rod, per lb.....	4.	6	
Hydrochloric acid, cwt.....	5.	0	
India rubber, Para, fine, lb.....	3.	9	
Mica, in orig. cases, medium, lb.....	3/6 to	6.	0
Petroleum, Russian spot, gal.....	9 1/4		
Quicksilver, bottle (Spanish).....	7.10.	0	
Sal ammoniac, lump, firsts del U. K., ton.....	44.	0.	0
Sulphate of ammonia, f. o. b. Liverpool.....	12.17.	6	
Sulphur, recovered, ton.....	5.	5.	0
Shellac, cwt.....	4.	0.	0
Platinum.....	9.	5.	0
Tin ore, ton.....	130 - to	132.	0.
Zinc, Vienne Montagne, f. o. b. Antwerp, ton.....	25.17.	6	

Electric Arc Welding

The immense progress which has been made in metallurgical practice in recent years is well illustrated by the various new methods of welding, which have been developed in the last decade or so. Based on very different principles, they all have their own field of usefulness: the process of thermit welding based on the ingenious aluminothermic reaction of Dr. Hans Goldschmidt, the gas welding processes making use of the high temperatures of the oxyhydrogen or the oxyacetylene flame, and finally of the electric arc welding process. Thus it has come that the day has passed when ordinary defects such as shrinkage cracks, blow holes, etc., in expensive castings necessitated the making of another casting, or when a crack in a street car truck or a break in a locomotive frame meant the keeping of the car or locomotive out of service for long periods while a new part was ordered from the builder.

The following article gives details of the electric arc weld-

operator can then move his electrode about over the work and the arc will follow, thus enabling him to concentrate the heat on one spot or spread it over a wide area.

Electric arc welding may be divided into two classes, namely, that requiring the graphite electrode and, secondly, the metallic electrode.

The graphite electrode method requires a potential of 50 to 60 volts at the arc, and a current of 300 amperes or more, and



FIG. 2.—COMBINATION ELECTRODE HOLDER FOR BOTH METALLIC AND GRAPHITE ELECTRODES. HEAD SHIELD.

can be used for all kinds of welding, filling in, building up, cutting, etc. In this method the filling in or welding metal is supplied from an outside source, that is, the operator either has a rod of soft iron or steel or uses scrap metal which the arc reduces to a state of fusion and forms into a homogeneous mass with the part being repaired.

When the metallic electrode is used, the potential at the arc is usually lower, according to the nature of the work. In this process the metallic electrode is consumed, particles of molten metal from the electrode being deposited on the work being repaired. Thus the process has to be interrupted from time to time so that the operator can insert a new electrode. An important use of the metallic electrode is in repairing cracks in large work, for the operator can work on vertical or overhead surfaces. In fact, in locomotive repair shops, it is no uncommon sight to see an operator repairing a crack in a crown sheet from below, all of the metal consumed from the electrode being deposited on the work, thus there is no necessity of the operator stepping aside to dodge drops of molten metal. It is of special importance in using the metallic electrode that a uniform potential be maintained, for slight variations in the voltage disturb the arc and produce an imperfect weld.

From the above, it is evident that each of the two systems has its own application, and, in fact, in practically every shop different classes of work must be done, some demanding the graphite method and some the metallic.

While any source of direct current is suitable for this process of welding, it is at once evident that it would be uneconomical to employ the current from the ordinary supply networks, since for welding a potential of from 10 to 60 volts is required. Further means must be provided to always maintain the proper potential for welding, and finally it is important to protect the line against short circuits, which occur every time when contact is made before the arc is drawn.

All these considerations emphasize the necessity of having a special current supply for welding. Fig. 1 shows a portable 30-ampere multiple-unit welding outfit of the C and C Electric & Manufacturing Company. It consists of a motor (which is either direct- or alternating-current motor, accord-

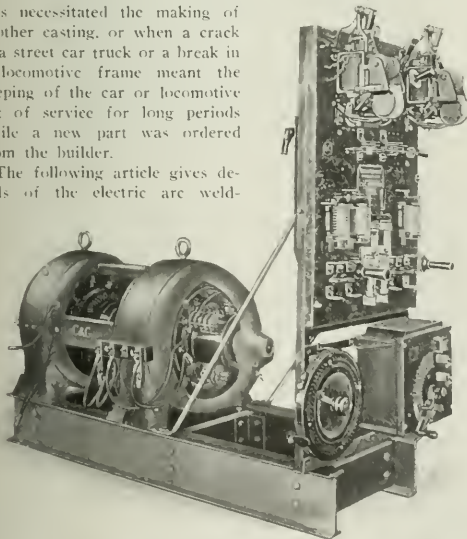


FIG. 1.—PORTABLE 300 AMPERE MULTIPLE UNIT WELDING OUTFIT

ing method and apparatus of the C and C Electric & Manufacturing Company in Garwood, N. J.:

The operation of arc welding is very simple. The usual practice is to connect the work to be repaired with one wire from the electric circuit and to attach the other wire to an electrode handled by the operator. The electrode is first brought into contact with the work, establishing the circuit, then drawn away from the work, forming an arc. The

ing to the system of supply in the plant) coupled to the generator which is of a specially wound variable-voltage direct-current type producing the welding current. This motor-generator set is connected with the switchboard which by the means of a single-throw switch permits the use of either the metallic or graphite electrode, alone, or both at the same time.



FIG. 3.—GRAPHITE ELECTRODE HOLDER. HAND SHIELD

It also contains the protective device, without which in the moment contact is made between the electrode and the work a short circuit would occur, and the generator would burn out or the motor circuit-breaker would be thrown. Thus it would never be possible to get the arc started. This is provided against in the C and C outfit by means of the automatic series relay. This provides that whenever the electric weld-

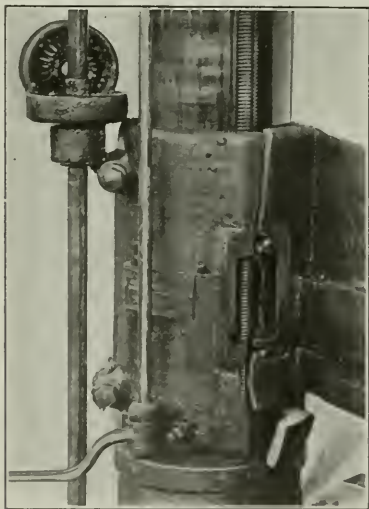


FIG. 4.—CAST IRON HOUSING SUPPORTING ARM OF RADIAL DRILL BROKEN IN THREE PARTS

ing circuit is broken a resistance is inserted in series with the electrode, and when the electrode is brought into contact with the work only enough current is allowed to pass to form the arc, and the minute that the arc is drawn this resistance is automatically cut out of the circuit and practically the full potential of the welding circuit is applied.

Fig. 3 illustrates an electrode holder adapted to foundry use where heavy cutting is done. It may be used continuously with the graphite electrode without the handle becoming overheated while welding heavy sections or cutting off large risers from steel castings, cutting up scrap, etc.

Fig. 2 shows a combination graphite and metallic electrode for use in connection with all classes of work, the metallic electrode being used for welding cast and malleable iron, brass, copper, aluminium, marine boilers, tanks, cracks in locomotive fire boxes, etc. This holder makes it possible to make repairs in the most inaccessible places and reduces the cost of the same more than 80 per cent. below the older methods employed. A great saving of time is also effected. Pressing the button in the combination electrode holder automatically reverts from graphite to metallic welding and vice versa.

A well-ventilated universal sighting hood of metal (Fig. 2)

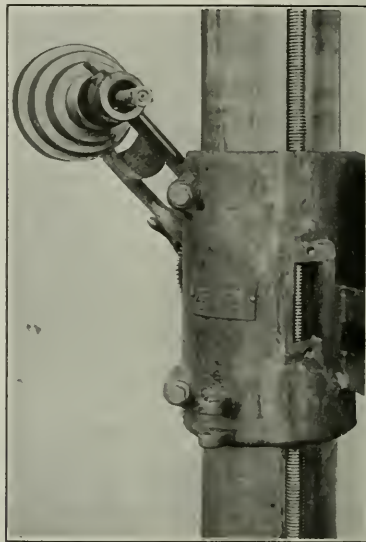


FIG. 5.—SAME AS FIG. 4 AFTER HAVING BEEN WELDED—DRILL READY FOR SERVICE

is provided, in which glasses of specially blended colors effectually protect the eyes of the operator while welding, leaving both hands free, and thus very materially expediting welding. For short jobs where one hand only is necessary the hand shield, shown in Fig. 3, is generally used.

The figures of cost in Table 1. (p. 469) should be of interest, as they are from practice. The column giving costs is figured out at 2 cents per kw. hour for current and 30 cents per hour for labor, but it can readily be figured out what the different operations will cost in any shop by using the figures in columns (1) and (2) and substituting proper costs for labor and power.

In general the same kind of material is used as a filler as the metal to be welded, but much higher efficiency will be obtained if specially prepared electrodes are used for welding various materials. When strength is not a very important factor, a soft iron rod or punchings may be used for welding steel. For repairing cast and malleable iron, a high-silicon iron should be used. Copper, brass, etc., are welded with like materials.

The system is not only applicable to welding, but also to cutting metal parts. As the electric arc produces the hottest flame known, the rapidity with which cutting is accomplished by concentrating the heat at a given point through the graphite electrode is unequalled. This is especially true of large sections which cannot be cut by gas systems.

TABLE I. FIGURES OF COST

	Time.	Average	
		KW.	Costs.
Gear case lugs.....	10 min.	6	\$0.07
Armature shaft (broken) 2".....	60 "	20-30	0.80
Dowel pin holes.....	5-12 "	4-8	0.07
Broken motor cases.....	2½-3½ hrs.	75-90	4.98
Broken lugs on a compressor cover, doors and grease-cup hinges.....	2-5 min.	1-3	0.03
Broken truck frames.....	30-60 "	20-35	0.63
Worn bolt holes in motors and trucks	5-10 "	3-5	0.05
Enlarged and elongated holes in brake levers	2-4 "	1½-3	0.03
Armature shafts, 2", worn in journals	2-3 hrs.	60-90	3.75
Armature shafts, worn in key-ways..	10-15 min.	7-12	0.10
Armature shaft, worn thread.....	30-60 "	10-15	0.24
Air brake armature shafts (broken)...	20-30 "	10-20	0.27
Leaking axle boxes.....	5-15 "	3-7	0.08

The system of arc welding has found introduction in locomotive shops of steam railways, street lighting, repair shops, steel and iron foundries, forge shops, shipbuilding and repair yards, etc. For years it has been the practice to braze the heads in tanks used for storing gas, acetylene, oxygen, etc. This was also the practice with air reservoirs used in connection with air brake apparatus. These heads are now welded in with the C and C method much more rapidly, and at a lower cost. Two heads may be welded on in forty minutes. This system is especially adapted to welding longitudinal seams, thereby cutting down the cost of the shell of tanks and boilers.

Figs. 4 and 5 show one repair illustrating the possibilities of electric welding. In Fig. 4 the cast-iron housing supporting the arm of a radial drill is shown broken in three parts. The casting was removed and welded with graphite electrode. Fig. 5 shows the casting after it had been welded, the drill being ready for service. Ordinary cast iron was used in making this repair. This radial drill has been operated in the shops of the C and C Electric & Manufacturing Company continuously since it was repaired in 1911.

Centrifugal Pumps

Their Proper Selection and Use

By H. De Huff

A well-designed, properly selected and carefully built centrifugal pump is a most satisfactory piece of apparatus, but it is important for its successful operation that it embodies all these qualifications. A centrifugal pump poorly designed, badly constructed or not suited to its requirements can be a most annoying source of trouble.

In the purchase of a centrifugal pump consideration should be first given to the type desired. Whether the pump is to be horizontal or vertical depends almost invariably on the local conditions. As a general thing it is very much better to install a horizontal pump than a vertical pump since easier accessibility to the working parts is possible.

The method of driving the pump is also determined by the available source of power. Where electrical power is obtainable at a low cost, the direct electric motor drive is the ideal method. Where steam is available a steam turbine may solve the problem.

Centrifugal pumps may be divided into two general classes, single-suction and double-suction, determined by the manner in which the impeller takes the water. The former division is further subdivided into single-stage and multiple-stage.

Single-suction, single-stage pumps are generally used where the quantity of water to be handled is comparatively small, or the head to be pumped against is comparatively low.

Double-suction pumps in general are suited for large capacities in low heads, or moderate capacities in high heads where the required head may be secured with one stage. Where the head is such as to require a number of stages, a succession of single suction impellers is generally used, contained in one

case, each impeller delivering its water to the adjacent impeller, thus producing a total head equal to the head produced by one impeller multiplied by the number of impellers.

The number of stages used depends upon the available speed. With the motor speeds ordinarily available running up to 1700 r.p.m., sizes up to 100 hp, a head of 150 ft. per stage is easily obtained, and in fact there are numerous instances where heads as high as 200 ft. are obtained.

One of the most desirable features in a horizontal pump is to have the case divided on a horizontal plane, passing through the axis of the pump, with the suction and discharge connections made to the lower half of the pump case; the two halves of the case being bolted together, removal of the upper half opens up the entire working parts of the pump. This is particularly necessary for multiple-stage pumps, for if the case is not bolted in this manner it is difficult to gain access to the interior of the pump. The larger the pump and the greater the number of stages, the more difficult this becomes.

Bronze impellers are usually most desirable, particularly where heads of 100 ft. and more are used. The mixture of bronze should be a high-grade one, such as to make a tough and very strong metal. The mixture, known as Government bronze, 88 per cent copper, 10 per cent tin and 2 per cent zinc is a very satisfactory metal for general use in centrifugal pump work.

All impellers should be fitted with removable bronze wearing rings where they run close to the pump case so that when wear comes on these parts it is not necessary to replace the entire impeller but merely provide new rings.

The pump shaft should be protected by removable bronze sleeves, so that the liquid pumped does not come in contact with it. These sleeves should extend through the stuffing boxes to prevent cutting of the shaft by the packing which might happen, depending upon the care given the pump.

Bearings should be entirely separate from the stuffing boxes and of the self-oiling type. They should be made with split removable shells babbitted so as to facilitate renewal.

The pump should be connected to its motor by a flexible coupling. The most satisfactory coupling for pump work is the steel pin and rubber bushing type. This allows of sufficient flexibility to care for a slight wear in the bearings and also allows the pump and motor shaft to move slightly endways independently.

From an operating standpoint the important features to be considered are accessibility of all parts and the ease with which the wearing parts can be replaced.

In installing centrifugal pumps they should when practicable be placed so as to reduce the suction lift to a minimum. However, a properly designed centrifugal pump will take care of a very good suction lift without affecting the capacity or efficiency. A lift of 22 ft. is not too much if the suction pipe is comparatively large and there are no foot valves to introduce an excessive amount of friction. Not all centrifugal pumps, however, are capable of operating on such high suction lifts. If not properly designed for such a lift there may be a considerable falling off in the capacity and efficiency of the pump.

Fig. 1 shows a D'Olier double-suction pump for 3000 gal. per minute, 110 ft. total head, at 1150 r.p.m., with an efficiency of 76 per cent.

All centrifugal pumps require priming and the case and suction pipe filled with water before they will start pumping, as they have no inherent ability to produce a suction lift unless water is passing through them. However, it is a simple matter to provide for priming. This can be cared for in the following various ways.

First, the pump may be placed below the level of the water from which it draws its supply so that when the suction valve is opened the case is filled.

Second, a foot valve may be placed in the bottom of the suction pipe or a check valve in the suction line below the suction water level so as to retain the water in the pump case and suction pipe. This is very satisfactory provided the de-

sign of the valve is such that it does not cause an excessive loss of head and provided the water is free from floating objects which might get into the valves and prevent them from properly seating. When first starting up the suction pipe must be filled from an outside source of supply.

Third, a steam siphon may be used to exhaust the air. This is efficient for small pumps and where the suction lines are not very large.

Fourth, small pumps may be primed by means of a hand-operated air pump.

Fifth, a separately driven vacuum pump may be installed to exhaust the air from the case and suction pipe. This is an excellent method for priming very large centrifugal pumps.

Centrifugal pumps should be carefully lined up when they are put in service. It is a comparatively easy matter to bend any bed plate by drawing up too hard on the anchor bolts. Attention given to this point will help to insure a smooth-running machine.

For most centrifugal machines a square graphite flax packing will be found satisfactory. This should be cut at an angle of about 30 deg. and the joints staggered. Packing should be first drawn up rather tight, then the glands eased up slightly. A mixture of heavy oil and graphite may be used in stuffing boxes with very good effect.

A properly designed centrifugal pump operates with very little noise and practically no vibration unless air or foreign bodies enter the suction pipe. The presence of air is indicated by a rumbling noise and by a falling off of pressure and capacity.

If for any reason the pump does not appear to deliver its full quantity of water when operating at its rated speed and pressure before entering a complaint to the manufacturer it is advisable for the user to open up the pump case, investigate and see if any foreign objects have become drawn into the

should be left to the pump manufacturer as the speed of it should be determined by the pump requirements and depends also on the available pump patterns and designs. Of course, there are conditions where the motor speed is practically fixed and in this case the pump manufacturer has to make his designs and number of stages conform to the available speed.

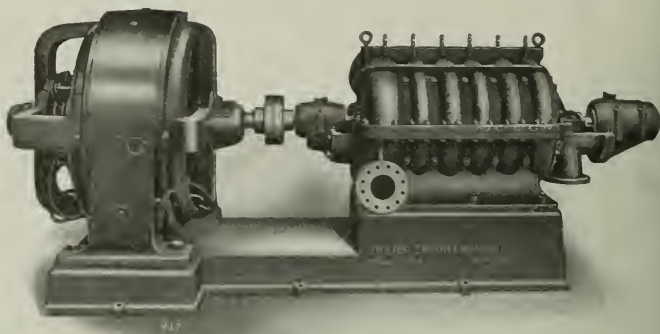


FIG. 2.—SIX-STAGE TURBINE PUMP DIRECT CONNECTED TO MOTOR

The horsepower rating of the driving motor should be sufficiently liberal to allow for a slight excess in capacity and pressure for the pump above contract requirements. It is generally advisable to allow 10 per cent above the power calculated from the given capacity, head and guaranteed efficiency. Consideration should be given also to the pump operating at pressure and capacity other than the rated condition.

By varying his designs the pump manufacturer can adjust more or less his characteristic performances to meet different conditions.

There are occasions when installing a pump when it is desirable to provide for considerable excess capacity at low heads.

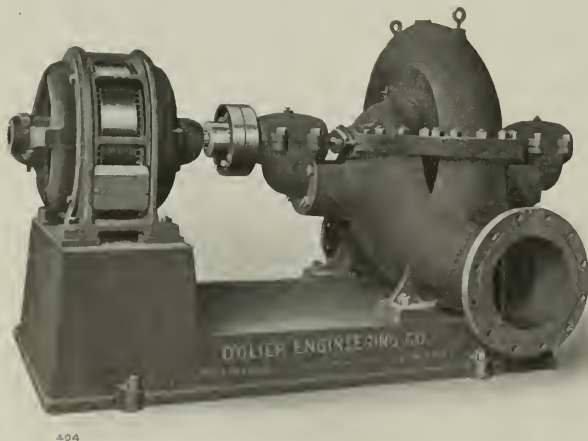
In such cases the motor should be of sufficient capacity to care for these conditions without placing a dangerous overload on it.

Centrifugal pumps are in general adapted to a wide range of conditions of service. They can be furnished for practically any capacity desired and for most pressures required commercially. The flow of water from the pump is smooth and entirely free from vibration and pulsation such as is caused by direct-acting pumps.

The centrifugal pump also has the advantage that it cannot damage itself by accidental closing of the discharge valve. The closing of the discharge valve merely results in an increased pressure of from 10 per cent to 40 per cent above the normal operating pressure, with the added fact, also, where the pump is taking its water under suction lift if run sufficiently long with the discharge valve closed it may drop its suction.

In all properly designed centrifugal pumps it should be possible to considerably reduce the pressure below rated conditions without dangerously overloading the driving motor.

Efficiencies obtained in centrifugal pumps depend on the capacity, pressure, speed and the nearness to securing ideal proportions in the impellers and pump case. Efficiencies are



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FIG. 1.—DOUBLE SUCTION PUMP FOR 3000 GALLONS PER MINUTE—110 FEET TOTAL HEAD AT 1150 R.P.M.

pump. Several cases have come under the writer's observance where such articles as tin cans, rags, rope, cotton waste and similar objects have been discovered in pumps when complaints have been made that they were not delivering enough water.

Selection of the motor to drive the pump is a matter that

obtained, however, running over 50 per cent in a 100-gal. pump to 85 per cent or even more in a 40-in. centrifugal pump. Very good efficiencies are obtainable in moderate capacity pumps. Under favorable conditions, and if the capacity is, say, 2000 gal. it is possible to obtain an efficiency of 80 per cent, although conservative manufacturers as a rule do not make guarantees so high.

The development of the centrifugal pump during the last decade to its present state of efficiency and wide range of uses reflects great credit upon these several pump manufacturers who, by reason of their appreciation of the possibilities of this type of pump and their enterprise, have brought it to its present position of importance.

The D'Olier centrifugal pump, manufactured by D'Olier Centrifugal Pump & Machine Co., of Philadelphia, two types of which are illustrated in Figs. 1 and 2 in this article, is an excellent example of the advance made in the development of the centrifugal pump.

New Catalog of Laboratory Supplies

Messrs. Eimer & Amend, 205-211 Third Avenue, New York City, have just issued the 1913 edition of their Catalog C of Chemical and Metallurgical Supplies and Assayers' Materials. It is a volume of 483 pages, 8 by 10½ inches, bound in red cloth, nicely printed and profusely illustrated. It should be welcome in every scientific, industrial and works laboratory.

A comparison of this latest edition with former editions of the well-known catalog brings out several interesting points, all of which indicate progress. The arrangement of the listed articles is, as before, alphabetical, but the classification has been carried through with greater consequence. For instance, all kinds of tubes are listed under tubes, which should assist in many cases the quick location of an article without referring to the index. The latter, however (17 pages), has wisely been retained.

Of greater interest, however, is a comparison, under the various headings, of the forms of apparatus listed in the present and former editions, since in the latest edition, as far as possible, all apparatus have been omitted which have been superseded by modern types of proven merit. Such a comparison, therefore, brings out forcibly the great advances made in laboratory apparatus design and construction in recent years and also emphasizes how successfully and energetically this old chemical and metallurgical laboratory supply house has kept up with the progress of recent years.

Personal

Mr. D. W. Brunton has returned to Denver from a visit to Greenwood, B. C., where his son, Frederick K. Brunton, is on the smelter staff of the British Columbia Copper Co.

Mr. John V. N. Dorr, of Denver, has returned from an Eastern business and pleasure trip. He attended the Boston meeting of the American Institute of Chemical Engineers and expects to go to Butte for the August meeting of the American Institute of Mining Engineers.

Dr. Rudolf Gahl, of Morenci, Arizona, is on a professional trip in the East.

Mr. H. W. Hardinge, of New York, is in Europe on a business trip.

Mr. Joseph A. Holmes, director of the Bureau of Mines, has taken a party of engineers to Alaska to investigate the Matanuska coal fields.

Mr. Charles W. Newton, general superintendent of the Butte & Ballaklava Copper Company and the Butte & Benith City Mining Company, of Butte, Montana, has gone East on business.

Mr. Benjamin B. Thayer, president of the Anaconda Copper Mining Co., went to Butte in the latter part of July.

Mr. Walter Harvey Weed has purchased the business of the late Horace J. Stevens, of Houghton, Mich., and will edit

and publish the Copper Handbook which, under Mr. Stevens' direction, became an authoritative work on the copper mining companies of the world. Mr. Weed's familiarity with the copper field insures the continuance of an excellent publication.

Mr. H. F. Wierum will have charge of the trial of the Hall process of sulphur recovery at the Balaklava smelter, California.

Superintendent B. L. Wolfe, Profs. E. C. O'Keefe and A. W. Young accompanied a number of the students from the Kansas School of Mines and Metallurgy to the mining section of southeastern Kansas and southwestern Missouri in the Joplin district, during the first week in July to give the young men an opportunity of inspecting the actual workings of the mines, smelters and mills of that district.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

REFINING METALS (Continued).

393,526, Nov. 27, 1888, Edward L. Smith, of Ansonia, Conn., administrator of Luther L. Smith, deceased.

Relates to the refining of metals and consists in placing a plurality of sheets of the impure metal in horizontal layers, separated by wood strips and sheets of cotton cloth in an electrolytic cell. The series of plates are so connected electrically that the top plate is the anode and the bottom plate the cathode, the intermediate plates constituting secondary electrodes, the lower side of an upper plate constituting the anode and the upper side of a lower plate, the cathode. During electrolysis any impurities, including gold, etc., collect on the cotton cloth, and may be removed at the end of the operation. The free acid liberated at the cathode being of lower specific gravity than the electrolyte, rises through the cotton cloth and comes in contact with the anode, thus automatically maintaining sufficient circulation.

405,452, June 18, 1889, Charles O. Yale, of Rome, N. Y., assignor of one-half to Moses M. Davis, of same place.

Relates to an apparatus for electrolytically refining base bullion. The impure lead is cast into anodes enclosed in muslin bags and suspended from a radial arm arranged to rotate above a vat. The anodes are arranged in concentric circles. Between the circles of anodes are stationary cylindrical cathodes. To remove the growths from the surface of the cathode a series of combs or scrapers is suspended from one of the radial arms, whereby the surfaces of the cathodes are scraped free from any spongy deposits which then collect at the bottom of the vat. The electrolyte is arranged to be circulated through an independent tank in which it is heated and maintained at a proper density.

405,604, June 18, 1889, Alberto Rovello, of Turin, Italy

Relates to the precipitation of copper from solutions resulting from the treatment of ores or the so-called "ashes" obtained in roasting copper sulfide in making sulfuric acid. When metallic iron and copper are immersed in such a solution an emf of about 0.60 volt is generated, and if zinc be substituted for the iron the emf increases to about 1 volt. This emf is utilized to generate a current by short-circuiting the electrodes, the current being utilized to deposit copper from the solution. A plurality of the electrodes are assembled in separate compartments of a cell, the several compartments formed by diaphragms, through the compartments containing the copper electrodes is conducted the stream of copper solution, while through the other is conducted a solution containing a similar acid salt of the metal constituting the electrode therein. For the electro-deposition of zinc, solutions of zinc are substituted for the copper solution and the electrodes connected to a dynamo, an emf of not above 1.40 volts being used. The apparatus consists of a plurality of sections of wood, generally U-shaped, separated by diaphragms of porous material and bolted together like a filter press.

Book Reviews

Metallic Alloys: Their Structure and Constitution. By G. H. Gulliver. Second Edition, revised, largely rewritten and greatly enlarged. 12mo, 409 pages, 310 illustrations; price \$3.25 net. London: Chas. Griffin & Co., Ltd.; Philadelphia: J. B. Lippincott Company.

This is a great improvement on the first edition, and as it now stands is perhaps the best elementary presentation in English of the theory of alloy structure. It is not a guide to the practical manufacture of alloys, but to the study of their minute structure and constitution. The author considers alloys as solutions of metals in each other, and applies in detail to them the thermodynamic theory of solutions. Is it accident, or the simple logic of the facts, which excludes the ionization theory of solutions, and even the word "ionization," from the book, although the whole subject is one of solution?

Not as criticism of the book, but as a comment upon the present state of information about alloys, it should be remarked that the book contains no definition of what constitute alloys. True, alloys of iron with carbon are mentioned, also of copper with phosphorus, but where alloys cease and ordinary chemical compounds begin is nowhere explained in the book.

Further, some of the alloys are being spoken of as forming chemical compounds, but the line between solution and chemical compound is drawn very loosely and, as the reviewer thinks, unsatisfactorily; in fact, it is the reviewer's personal opinion that progress in the future will probably lie in the direction of considering *all* solutions as chemical combinations in indefinite proportions, and the definite chemical compounds derived therefrom as being simply particular solutions in definite proportions, individualizing themselves under definite determining conditions.

* * *

Cyanide Practice in Mexico. By Ferdinand McCann. Octavo (15 x 22.5 cm.), 199 pages, 39 illustrations. Price \$2. San Francisco: Mining and Scientific Press. London: The Mining Magazine.

The eighteen chapters cover the cyaniding of gold and silver ores in Mexico, principally silver, and for this reason contain interesting information, because the cyaniding of silver ores has made greater advances in Mexico than in any other part of the world. Lack of a table of contents makes it difficult to summarize, but leafing through the book shows it full of valuable detailed practical information, up to date, expressed in good style, and illustrated by numerous photographs, flow-sheets and working drawings. The large detailed tabulation, by Mr. W. E. Hindry, of complete working details at sixteen plants, arranged under seventy-two columns, is a good and useful piece of work, and deserves special mention.

Most of the book was originally published by the author in the Spanish language.

* * *

Law of Conflicting Uses of Electricity and Electrolysis. By George F. Deiser, Member of the Philadelphia Bar. Octavo (15 x 23 cm.), 138 pages. Price \$2.50. Philadelphia: T. & J. W. Johnson Co.

The legal decisions on these subjects not being numerous as yet, the author has tried to include all that have so far been given, discussing fully those specially concerning electrolysis. The author has only partly carried out his program in that two-thirds of the book relates to the conflicting rights of telegraph, telephone and traction companies to use the streets or highways, and but thirty-eight pages relate to damage by electrolysis.

The latter subject is handled systematically, but most of the material is a recital of the facts and findings in the case of the Peoria Water Works Company vs. the Peoria Railway Company, in which corrosion of water mains was caused by the defendant's return current. It may appear a contradictory statement but the definitions of electrolysis and electrolytic

action given in these decisions are so faulty that some one (judge, lawyer, expert or whoever was responsible for them) needed very radical revision of his ideas. If justice was ultimately done in these cases, it was in spite of faulty definitions and not because of correct ones. The general tenor of the decisions, however, that is, as to their ultimate practicability and fairness, appears excellent, and Mr. Deiser has rendered a distinct service to electrochemists in compiling them and thus rendering them accessible.

* * *

American Mine Accounting. By W. H. Charlton. 367 pages, 250 forms and diagrams. Price \$5. New York: McGraw-Hill Publishing Company.

Generally speaking, any technical book of today in order to be one of value, must be a special treatise upon some particular subject.

This book is preeminently one of this type.

One does not have to be an expert accountant in order to appreciate its value to the mining public. Nor is its value restricted in any sense to mining and allied industries, as with slight changes and modifications the forms given therein may be adapted to nearly every line of industry. As scientific management is the path by which many business enterprises have been led from failure or indifferent results and brought to culminate in financial success, a study of the methods employed therein, shows that *detail* is given first consideration. "Small leaks undetected, soon drain the barrel."

Someone has said that 90 per cent of the value of modern education consists in knowing where to find information quickly, and to be able to apply it rapidly and accurately. Therefore to one compiling a system of accounting, where attention to *detail* is so necessary, this book will be found invaluable. Moreover it is thoroughly practical.

This book is also particularly valuable inasmuch as it is an American book and uses terms with which the American mine accountant is familiar. The author refers in his preface to three other books upon the subject; the one published in this country is rather elementary, restricting itself almost entirely to bookkeeping; the other two are English and Scotch books respectively, and hence necessarily contain much which is not readily intelligible on account of the foreign terminology employed.

* * *

Investigations of Methods of Analysis of Cane Products.

By William E. Cross, Louisiana Bulletin No. 35 (December, 1912). Agricultural experiment stations of the Louisiana State University and A. & M. College.

This bulletin of 83 pages contains a report of investigations on the methods of analysis of sugar cane products carried out in the Chemical Department of the Louisiana Experiment Station, Audubon Park, New Orleans, La., during the past fifteen months.

"It is universally recognized that accurate chemical control of a sugar factory and dependable analysis of the products turned out are essential to scientific progress in sugar manufacture, and it is therefore a matter of vital importance to have the methods of determination of such products of the highest accuracy and the greatest practicability. In this bulletin are submitted reports of experimental efforts in the direction of elaborating and perfecting such methods, as well as investigations on the analysis of sugar products in general."

The eight chapters of the bulletin deal with the following subjects: The determination of dry substance; the use of the refractometer in sugar-house work. The application of dry lead defecation to sugar-house analysis. A rapid method for the determination of glucose in juices. A modification of the Clerget method of determining sucrose in molasses. The effect of urea and betaine on the rate of inversion of sucrose by hydrochloric acid. The direct determination of sucrose in presence of reducing sugars. The acidity of raw cane sugars. Note on sour cane.

An appendix contains numerous tables. The bulletin on the whole is decidedly interesting and useful.

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The Montana Meeting of the American Institute of Mining Engineers

The 105th meeting of the American Institute of Mining engineers, held last month in Great Falls, Butte, and Anaconda, has turned out such a brilliant success as to leave no doubt of the beginning of a new era in the affairs of the Institute. The management of the meeting by the local committees, led by the officials of the Anaconda Company, was grand. The visits to the magnificent mines and smelters of the district were most instructive. The program of technical papers presented at the meeting was simply classical.

All the papers had been printed in advance. As the report of the meeting in this issue shows, the papers were of an exceedingly varied character, as they covered smelting and leaching, concentration and mining and geology. And yet they were a unit, as they all related to the Great Falls, Butte, Anaconda district. And they were all authoritative, being written by the responsible mining and metallurgical engineers of the district themselves.

Western hospitality is nothing new. But we doubt whether it ever manifested itself in a more glorious professional way than in giving to the world this set of papers on mining and metallurgy in Montana which will be the lasting monument of a great convention.

The Influence of South Dakota in Cyanidation

With the passing of time and the general adoption of successful processes and machines, the history of metallurgical development is likely to be lost to sight and current practice taken as a matter of course. And yet the historical feature of metallurgical progress is of surpassing interest, not only on account of its general bearing on the industry, but as showing how true it is that necessity is the mother of invention. In cyanidation, some of the most important steps have been evolved in obscure mills, where failure would have excited little more than local comment, but where success was of momentous importance to the operators. Subsequently the machine or method evolved has been recognized as fundamentally important, and its use has spread rapidly.

The closing article in the series on cyanide practice in South Dakota, appearing on another page, recalls the fact that the metallurgists of the Black Hills district have made many notable contributions to the art of extracting gold and silver from ores by the cyanide process. That district was the scene of some of the earliest attempts at cyanidation, by the American company which exploited the MacArthur-Forrest patents. The insuperable obstacles then encountered and the failures which ensued seem insignificant in the light of present practice; but they served to stimulate invention, and their record is easily forgotten in the consideration of the success which followed.

The importance of sand-slime separation was early recognized, for the first attempts at leaching unclassified pulp were decided failures. About 1899-1900, crushing in cyanide solution was introduced by John Henton, at Central City, in the first mill to operate continuously and successfully. Sand was separated from slime in spitzkasten or other form of settling box. The initial success of this step led to the erection of other mills on a similar plan, one of which for a long time held the record for low-cost operation in the Black Hills. Cones, spitzkasten, and settling boxes arranged for intermittent discharge of sand subsequently came into common use in cyanide mills for sand-slime separation.

In 1904 the separation of sand and slime was further improved by the invention of a mechanical classifier by J. V. N. Dorr who, in July of that year, operated the first of these machines in the mill of Lundberg, Dorr & Wilson, at Terry. A system of double-cone classification had been in use, but was inefficient on the kind of ore under treatment. It was impossible to obtain a leachable sand, free from slime, and failure of the mill appeared inevitable unless the separation could be improved. It was under such conditions, to avoid personal loss, that mechanical classification was developed and introduced into western cyanide practice.

It was in that day, also, that evolution in slime treatment began. Intermittent decantation had been standard practice, but about 1901 John Randall introduced continuous decantation with counter-current washing at the Lexington mill. Large cones were used, but were found poorly adapted to the purpose on account of irregular operation and the tendency of solid slime to build up on the sides. Despite the failure of the process at that time, it was recognized as embodying advantages over the old decantation process.

Although vacuum filtration of slime was not a contribution from South Dakota, the first Moore process plant to operate continuously and successfully was installed in the Lundberg, Dorr & Wilson mill in 1904. The slime was prepared for filtration by thickening in cones, but the inefficiency of this method again led to the development of a new machine. It was in 1906, at the Mogul mill, at Pluma, that Mr. Dorr invented and first operated a continuous, mechanical slime thickener. This machine has since found wide acceptance, not only in cyanidation where it has contributed to the success of continuous decantation, but also in concentration processes, particularly flotation.

Two other steps toward modern cyanide practice were first taken in the Black Hills, at the Homestake mills. For many years the metallurgical process employed here included only amalgamation and the leaching of sand with cyanide solution. Slime was a waste product. The loss represented in the latter was subsequently converted into profit through the inventions of C. W. Merrill, who devised the sluicing filter-press and the system of zinc-dust precipitation. The first large plant embodying these inventions was erected for the treatment of Homestake slime at Deadwood about 1906-7. Since that time the Merrill processes have enjoyed world wide use and have contributed to the success of the cyanide process.

The whole record speaks for itself. It is doubtful if any

single mining district in the world has contributed to cyanidation more methods and machines which have been as uniformly successful as those which had their inception in the treatment of Black Hills ores. Whether they were evolved of necessity to save their authors from financial loss, or to increase efficiency and add to profits already assured, they reflect great credit on a small district in a state of lesser rank among the mining commonwealths of this country.

The Metal Markets in the Hot Months

In the past six months we have witnessed several pronounced phenomena in the metal business. First of all, an increasing market demand for each metal in face of an increasing price; second, an inhibition of real demand coupled with a tightening of credit in the world money-markets; third, a decrease of production concomitant with decreased demand, and, fourth, a holding of prices at reasonable levels by reason of the closing down or restriction of certain large producers. All this is typical of the primary and secondary metals and has its proximate cause in the great ease and distinct certainty with which intelligence is diffused among all classes in the second decade of the twentieth century. It is a hopeful sign of prosperity for the next few months.

Copper exemplifies this well, and so does iron and steel, spelter and lead. The first mentioned is as necessary to the age inaugurated by Faraday and Edison as water is to animal life. The second is as necessary to the age of the great engineering activity as air is to animal life. On these two metals depends our wonderful modern civilization.

General business, which is closely tied up in this age with the metal markets, shows a marked correspondence with the metal business. The buying movement in the latter part of the winter spread through all lines of the industrial fabric and all metals reached new high-level records of production. Latterly, the Balkan imbroglio, the trouble in Mexico, the uncertainty about the tariff and currency in the United States, the prodigality of government and private expenditure, the social unrest and ferment, all have been restricting factors, and have put the check rein on credit all over the world, but especially in Germany. Probably the uprising in the Balkans has had much more to do with the tightening of credit than is superficially apparent. This strain on credit has been real in its existence and potent in its influence. Its source was the great extravagance so manifest in the United States and in all other places in the world.

Among business men, the point of interest is now in the crops. While, if these are of great size, a temporary strain on credit will be produced at the close of 1913, yet the banks have generally so put their houses in order that no serious trouble can ensue. On the other hand, if crops are meagre a sharp period of general liquidation will cause hard times comparable to those of 1893. At present writing a large winter wheat crop has been harvested and the promise for spring wheat, corn, oats, and cotton is good. Quotations for these staples in Chicago are gradually descending. However, it must be remembered that two weeks of hot, dry, windy weather in the Mississippi Valley would "send these prices kiting." But the general diffusion in agriculture of what is known as scientific farming has so improved

the methods of the practical farmer as to slowly increase the yield and quality of foodstuffs that the chances are better than ever for large and bounteous crops.

It can be laid down as a fact, therefore, that the metal business must find its lead from the farming business and that until we know what the crops really are nothing but a continuation of present conditions can be looked for. In general connection with all this let us state that there are good judges of business who state that there must be a general liquidation and readjustment before any period of improvement or prosperity can start. Liquidation that should have come in 1908-1909 was tempered by certain powerful interests in a spirit of what we believe to be mistaken altruism. These same judges believe that we are starting on an era of descending prices following a fifteen-year period of ascending prices. Such a readjustment, though of benefit to all, will not be regarded as acceptable by the younger generation of energetic business men. All this is for the future. Right now a spirit of caution is the right one to have until the crops are gathered into the barns, and until Congress has produced something satisfactory or at least definite in the shape of a tariff bill and a currency bill.

The Output of Rolled Iron and Steel

Statistics of the output of rolled iron and steel furnish a more accurate index to our production of commercial material than do the more frequently cited statistics of steel ingot output. In the first place, the steel ingot is not an article of commerce, and indeed the production of steel ingots is not really the production of steel at all, but merely the production, by casting, of ingots. In the conversion of ingots into merchantable forms of steel much scrap is produced, which goes through, rather than into, the steel-making process again. In the second place, wrought iron is practically interchangeable with soft steel for many uses, and at any rate it is a closely cognate product, and in studying the tonnage output of rolled steel the tonnage of rolled iron may conveniently be added, furnishing an aggregate of the commercial material produced in rolled form.

Comparing the tonnage of steel ingots with the tonnage of rolled steel, one finds a fairly constant loss of approximately 24 per cent. This was the case in 1904 and 1905, back of which the present statistics of rolled steel do not go, and was also the case in 1912, the rolled iron and steel statistics of which year have just been presented. In a few of the intervening years the loss dropped to 22 or 21 per cent., but these were years of light production when ordinary proportions were disturbed.

Practically none of the material thus subtracted is actually lost. A relatively small portion of the steel is converted into scale, useful in the blast furnace and in some other directions, while by far the major part becomes new steel scrap, used in the open-hearth furnace. Before the general adoption of the basic open-hearth furnace such scrap was customarily charged in the Bessemer converter, being used in varying proportions to cool the heat and thus make up for variations in silicon content of the blast furnace metal. As steel works have found the better outlet for this material through the open-hearth furnace, the Bes-

semer department has been deprived of it, and the blast furnace is required to work more closely to silicon than formerly though the mixer has helped in this direction.

While the proportion of Bessemer steel to total steel has been decreasing constantly, it was nevertheless one-third in 1912, and with a loss of 24 per cent. from the ingot to the finished rolled steel, and further production of scrap by processes subsequent to the final hot rolling which marks the point at which a given material in course of manufacture is returnable for the statistics of "rolled steel," makes a total of scrap produced which it can readily be seen is quite large in proportion to the output of open-hearth steel. Thus a large percentage of "scrap" can be used in the open-hearth process without a great deal of old material being consumed. There is a somewhat popular misapprehension that the open-hearth steel industry consumes a large tonnage of old material, which is not an accurate conception. Detached open-hearth plants do use a large percentage of old material, but the basic open-hearth plant which draws direct metal from its blast furnaces through the mixer, and also draws scrap produced in rolling Bessemer steel from an attached plant, can get along without any outside scrap, and only employs it when it can be secured at inviting prices.

The following table shows, for the past seven years, our output of steel ingots (not including castings), rolled iron and rolled steel, in gross tons:

	Steel ingots.	Rolled steel.	Rolled iron	Total rolled
1906.....	22,624,431	17,401,911	2,186,557	19,588,468
1907.....	22,559,477	17,664,736	2,300,186	19,864,322
1908.....	15,667,027	10,889,744	1,238,449	11,828,113
1909.....	23,298,779	17,935,250	1,709,431	19,644,660
1910.....	25,154,087	19,281,123	1,740,156	11,623,379
1911.....	25,029,479	17,578,556	1,460,615	19,039,171
1912.....	30,284,682	23,019,259	1,637,582	24,656,841

The reported output of rolled steel represents the entire commercial material produced from the steel ingots, each weighed after its last hot rolling (including necessary shearing), thus including skelp rather than pipe, rods rather than drawn wire, sheets and black plates rather than sheet bars, and so on. For 1912 exported slabs, billets, etc., are included, but for earlier years this desirable inclusion was overlooked.

The decadence of rolled iron is striking, the year 1907 having furnished the maximum tonnage in recent times. Prior to the general adoption of mild steel there was a slightly larger tonnage of rolled iron, the entire industry being so small, however, that the maximum tonnage seems to have been about 2,500,000 tons, in 1890.

There is a very different alignment in the relative tonnage of different rolled products than obtained only a few years ago. The output of rails, for instance, reached its maximum tonnage in 1906, since when there has been an absolute decline in the tonnage and, of course, a still greater decline in the relative tonnage. Rails in 1912 comprised less than 14 per cent. of the total rolled product, merchant bars comprising 15 per cent., while the tonnages of rods, plates, sheets (including black plates for timing) and structural shapes all lay between 10 and 14 per cent., and skelp fell just short of 10 per cent. Thus we have seven products, rods, rails, shapes, plates, skelp, sheets and merchant bars, whose tonnage is not greatly dissimilar. These seven products in 1912 comprised 83.3 per cent. of the total rolled iron and steel output of the country.

Readers' Views and Comments

Formulæ for Calculation of Alkali Hydroxides and Carbonates

To the Editor of Metallurgical & Chemical Engineering:

Sir:—The analytical chemist is often called upon to determine the amounts of alkaline hydroxides and carbonates present in a sample of NaOH or KOH. The method which calls for the least amount of time is the following: Double titration in the cold with phenolphthalein and methyl orange according to the method of R. B. Warder (Treadwell & Hall, Quant. Anal.), together with the determination of the sodium and potassium salts according to the indirect analysis method.

It can be said of the indirect analysis method that it is not extremely accurate except in case the sodium and potassium salts are present in about equal amounts (or say 20% of one to 80% of the other), but it must be remembered that we are not here concerned with the determination of small amounts of one in presence of the other.

Under the conditions mentioned these methods are accurate and are shorter by far than any others as far as the mere operations are concerned, but the calculations are long and tiresome in the extreme. It was to shorten the latter that the writer worked out the following formulæ. These have been checked against hypothetical mixtures and will be found to be accurate within 0.01%.

ALKALINE HYDROXIDES AND CARBONATES ON BASIS OF ONE GRAM CHARGE FOR ANALYSIS

$$\% \text{ NaOH} = \frac{0.0561 (T - t)}{0.02805 + 0.02199 \left(\frac{N}{M} \right)}$$

$$\% \text{ KOH} = \frac{2.1993 (T - t)}{\left(\frac{M}{N} \right) + 0.7840}$$

$$\% \text{ Na}_2\text{CO}_3 = \frac{0.1831 (t)}{0.03455 + 0.02709 \left(\frac{N}{M} \right)}$$

$$\% \text{ K}_2\text{CO}_3 = \frac{5.4176 (t)}{\left(\frac{M}{N} \right) + 0.7840}$$

Where,

T = No. cc. half normal acid to phenolphthalein as indicator.

t = No. cc. half normal acid to methyl orange as indicator.

M = Weight NaCl found in indirect analysis.

N = Weight KCl found in indirect analysis.

H. A. MERENESS,

Knoxville, Tenn.

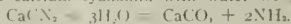
The Ostwald Process for Making Nitric Acid from Ammonia and the Production of Ammonia from Calcium Cyanamid

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The article in the August issue on this subject (page 438) is of great interest to me. It is true that the process of making nitric acid from ammonia produced from calcium cyanamid is new for England as stated there. But exactly this same process has been and is in successful commercial operation in a plant in Belgium though nothing has yet been published about it.

The following notes on the process should, therefore, be of interest. Naturally certain important details are being kept secret and must be withheld from publication.

If we mix calcium cyanamid with water we get ammonia.



To mix cyanamid with plain water is impractical for in-

dustrial purposes for the reason that not all of the nitrogen can thereby be extracted in form of ammonia. In order to obtain all the nitrogen as ammonia, we must use alkali and carry on the process along the following lines which have been developed in many years of hard work and experimental research.

If we mix the correct quantities of CaCN_2 , H_2O , Na_2CO_3 and NaCl in one dish and mix well, acetylene will be evolved. This is due to the fact that cyanamid contains 0.1 to 0.3 per cent of carbide of calcium. After the carbide has thus been removed the mixture is transferred to an autoclave and the steam turned on and the pressure raised to 6 or 8 atmospheres. All the nitrogen will then distill in the form of ammonia gas.

From my own experience I know that this distillation goes so well and so quickly that in ordinary daily operation one autoclave will distill in about one-half hour the ammonia from 1000 kg. cyanamid which contains 20 per cent of nitrogen. The extraction is nearly complete, as will be seen from a few analyses of the cyanamid before and after the distillation.

% Nitrogen in Calcium Cyanamid			
Before	After	Before	After
18.50	0.27	19.50	0.26
18.75	0.27	19.75	0.34
19.00	0.32	20.00	0.40
19.25	0.34	20.25	0.38

If the operation in the autoclave is carried out properly, the ammonia and steam come out entirely clean without carrying any cyanamid dust. Nevertheless, the practice is to pass the ammonia and steam through a cleaning apparatus for the purpose of catching any dust which might be carried along.

After that operation ammonia and steam are passed together into a column which cools and condenses the steam and lets pure ammonia gas enter into the gasometer.

Ammonia obtained by this process is very pure, the purity being 99.8 per cent or more.

The entire process is easily carried out and is a large source of ammonia which may be transformed into many chemical substances. We can transform ammonia nitrogen into nitric nitrogen from which we can get very easily ammonium sulphate or liquid ammonia, but the most interesting transformation process is the transformation into nitric acid and ammonium nitrate.

If we let the ammonia pass through tubes containing a catalytic agent (platinum) we can transform ammonia H_2N into NO_2 and with further oxidation in oxidation towers into nitric acid of 36 deg. B. This acid we can transform with the same ammonia into nitrate of ammonium and we can continue concentrating further to highest concentrations.

The process is exceedingly interesting and very simple and in the writer's opinion it has a great future.

A. B.

The Western Metallurgical Field Refining Sulphur in Wyoming

About three miles west of Cody, Wyo., is a large deposit of native sulphur, varying in thickness from 60 to 75 ft., and lying from 1 to 7 ft. below the surface. Recently the Northwest Sulphur Co. has secured control of 1,400 acres of this sulphur-bearing ground, and has tested the deposit by means of 150 drill holes. The results of drilling indicate the presence of about 500,000 tons of crude sulphur.

A part of the deposit has been uncovered, and sulphur is being mined from open pits. A small refinery has been erected on the property, and sulphur is being refined at the rate of about 12 to 15 tons per day. The system of refining

is extremely simple, and the equipment required is not expensive either in first cost or in operation. The plant consists essentially of steam boilers, retorts, retort-cars and bins. Figs. 1 and 2 give views of parts of the plant.

The crude sulphur is dumped into the retort-cars which are run into the retort and subjected to the action of steam at 60-lb. pressure. During this process molten sulphur runs from



FIG. 1.—RETORT-CAR, RETORT AND BOILER

the retort and into storage bins, and when a charge has been exhausted the car is withdrawn and dumped. The refined sulphur is reported to be 99.8 per cent pure.

During the month of July, 1913, the refinery produced an average of 12 tons refined sulphur per day, at a cost of \$8.40 per ton.

As the product sells for about \$20 per ton at Cody, Wyoming, the operation is decidedly profitable. In the month of August the capacity of the refinery was doubled, bringing it up to about 25 tons per day. Fuel cost is low, the company has about 200 acres of coal land from which a regular supply is drawn.

Jigging Black Sand and Gravel

Experiments now being conducted in Denver point to the possible use of the Richards pulsator jig in the retreatment of black sand and dredge gravel. The tests on the former were made with material collected in sluice boxes, and showed remarkable results. A small laboratory jig was used, bedded with gravel from $\frac{1}{8}$ to $\frac{1}{4}$ in. diameter, and arranged to yield spigot, side-gate and tailing products. The black sand contained considerable clay, which apparently coated the gold particles and prevented amalgamation. Careful panning of the different products after each test showed that practically all the gold was obtained in the spigot discharge. In one or two instances a color was obtained by panning the side-gate discharge, but none was observed in the tailings. A further interesting point was that the gold thus obtained was readily amalgamable, whereas before jigging it was not. Most remarkable of all, perhaps, was the complete recovery of very fine gold dust. It was to be expected that small particles of appreciable size would be recovered without difficulty, but it was surprising to note that the very finest dust also was collected in the spigot discharge, and that the other products were practically free from gold.

The application of the jig to dredge gravel is probably of greater importance. Tests on material passing a $\frac{1}{2}$ -in. screen suggest the possibility of jigging this size product, and obtaining a spigot discharge which can then be retreated on a second jig, with practically complete recovery of the gold. The arrangement of the jigs on the dredge boat should not be a difficult matter, as the machines are small, and the necessary power and water would be at hand.

Flotation Litigation at Butte

Mention has previously been made in these columns of the fact that the Butte & Superior Copper Co., at Butte, was using a flotation process under the patent of Mr. James M. Hyde. Recently the Mineral Separation Co. brought suit against the Butte & Superior Company, alleging infringement of the flotation process owned by the former. The case was heard and decided in favor of the Minerals Separation Co., and a decree was ordered as demanded by the plaintiff for treble damages and an accounting before a master in chancery. It is understood that the Butte & Superior Co. will appeal, and in the meantime operations will continue as in the past.

The history of litigation over flotation processes, as outlined in Mr. T. J. Hoover's book on "Concentrating Ores by Flotation," suggests the futility of trying to settle technical matters in courts of law. The record is one of decisions and reversals and compromises. As the author himself suggests: "All this mass of litigation is inconclusive. No one of these suits would involve a decision on all the patents. In fact, if all the law suits already started were brought to a final decision it is doubtful whether the public would know whether they could use the processes of successful litigants without fear of further litigation. If the whole situation could be placed before a competent court to decide on merit it would be far preferable to the engineer, but it might be less acceptable to the legal profession, for obvious reasons."

New Smelting Plant for the Granby Company

The Granby Smelting Company, operating in British Columbia, is constructing a new 2000-ton smelting plant at the Hidden Creek mines, Granby, B. C., at a cost of about \$3,000,000. According to a statement by Mr. Jay P. Graves, vice-president and general manager for the company, construction is progressing rapidly, and on July 1 there were 1125 men on the pay roll. Structural steel for the smelter buildings is on the ground, a brick plant is practically completed, and railway connections have been established. With no unforeseen delays the plant should be ready for operation about January, 1914.

One of the features of the plant is the construction of the underground ore-storage and crushing plant, connected with the smelter by tunnel. This arrangement will effectually over-



FIG. 2.—SULPHUR RUNNING INTO BIN FROM RETORT

come delays and difficulties of operation incident to the severe winter weather.

The company is establishing a model town and will build and maintain a recreation hall and gymnasium for its employees. A 40-room modern hotel is completed and a hospital will be built, comprising six private wards and one large ward, with operating room and every facility for caring for the ill and injured.

Labor Strike in Michigan

At the call of the Western Federation of Miners, the first general labor strike in the Lake Superior copper country occurred on July 23. This has been anticipated for a year past, during which time it has been understood that Federation workers were steadily organizing the miners of the district. Without other formality than the presentation of a protest against the one-man drill and a request for shorter hours and more pay, the strike was precipitated, and in a few days all mines and mills were closed. As a result of disturbances the state militia was called out and placed in charge of the upper peninsula of Michigan.

The specific demands of the Federation included recognition of the union, a minimum wage of \$3 per day of 8 hours, and the abolishment of the one-man drill. To this request the operators made no reply, and since the strike was called they have steadfastly refused to recognize the Federation or treat with its officers, even at the request of Governor Ferris. They have, however, indicated their willingness to meet their men as employees; and in defence of their attitude have cited their consideration for the men during past years and their success in adjusting differences between employees and the companies. It is believed that the firm decision of the operators to ignore the Federation is taken mainly on account of the past unsavory record of that organization in other mining camps where similar strikes were inaugurated.

At last reports the strike was still on, with little hope of an early settlement of differences. Pumps are operating in practically all of the mines, but no underground work is attempted. Although there is considerable iron mining in northern Michigan, it is considered unlikely that any attempt will be made to spread the strike into the iron country, as the Federation is not strongly organized there.

Company Reports

Utah Copper Co.—The twenty-first quarterly report for the second quarter of 1913, states that the production of copper in concentrates was 31,785,448 lb., an average of 10,595,149 lb. per month. This was recovered from 1,910,214 tons of ore, of which the Magna plant treated 55 per cent and the Arthur plant 45 per cent. The mills averaged 21,000 tons per day. The average grade of the ore was 1.2807 per cent copper. The grade is below normal, due to the necessity of mining very low-grade and partially stripped areas. The average cost per pound of copper, after allowing for smelter deductions and without crediting miscellaneous income, was 8.933 cents. If miscellaneous earnings are applied the cost was 8.16 cents. While the cost per pound of copper was high, the cost per ton of ore treated was reduced to a figure lower than had been attained in any previous quarter. The cost of mining and milling, including charges of every nature, was 90.3 cents per ton. The total net profit for the quarter was \$2,218,753, computed on the basis of 15 cents per pound for copper. Construction expenditures of consequence will have ceased by the end of the current quarter.

Nevada Consolidated Copper Co.—The sixteenth quarterly report, covering operations during the second quarter of this year, shows that the mill treated 762,880 tons of ore of an average grade of 1.76 per cent copper. The cost per pound of copper produced was 8.95 cents, as compared with 10.61 cents for the preceding quarter, and 8.33 cents for the last fiscal year. The net credit to undivided profits was \$98,422, after paying the fifteenth quarterly dividend and making charges for depreciation and ore extinguishment. Earnings were computed on the basis of 14.392 cents per pound for copper. The contract existing between the Steptoe Valley S. & M. Co. and the Giroux Consolidated Mines Co. for the treatment of the ores of the latter company, has been cancelled by the Giroux organization and will become effective in June, 1914. The contract was for five years with the privilege of cancellation by either party on one year notice.

Ohio Copper Co.—A report made public under date of July

22, states that the construction of the third section of the mill has been completed and that the plant is now treating between 71,000 and 75,000 tons monthly, with a total cost of mining, milling and transportation of 85 cents per ton. The copper content of the ore has averaged 1.015 per cent. The earnings have not been satisfactory and the company is contemplating the use of flotation instead of the standard process now installed. With this in view, Mr. M. W. Atwater has been retained to make investigations.

The Iron and Steel Market

Production of finished steel was heavy in August, at between 85 and 90 per cent of full rated capacity of the mills, substantially the same rate as in July. As outputs are normally decreased by hot weather, the showing for the two months is very good from a commercial standpoint. Except for the normal stocking of wire products against the fall movement to distributors, all the steel produced is being shipped and as there is no suggestion that buyers are stocking—the tendency is rather to reduce stocks—the indication is that actual ultimate consumption has proceeded at a heavy rate.

Strictly new buying was somewhat heavier in August than in July but as it remained far below the depletion of mill order books proceeded farther and finally reached the stage, which it was certain several months ago would be reached at one time or another, of mills looking for additional business of importance at cut prices. At the close of August the steel market in general is in a transitional stage, prices tending to decline to seek the level at which buyers will take hold more freely and actually fill up the mills. How far the readjustment will proceed cannot possibly be predicted, as it is a question of restoring a balance between elements which are not well understood. A great deal, too, will depend upon how the mills determine to shape their policy, whether to seek simply enough business to round out the year, delaying the greatest test until later, or conclude to book enough business to furnish a good foundation for running through the winter and into next spring. This is largely a matter of judgment and decision among a relatively few large steel producers.

Pig Iron

The pig iron buying movement came to a close early in August. In southern iron it was heaviest late in June and early in July; in northern iron it centered in July. The long decline, lasting from late in December until July, was clearly arrested, but only by the great help afforded through a material decrease in production, which decrease was spread over a period of several months, and amounted to a total of perhaps 20 per cent among the merchant furnaces. Had the buying movement lasted a little longer a definitely advancing market would undoubtedly have been inaugurated, by buyers being encouraged to purchase for deliveries farther ahead, but as it was most purchases were only through the third quarter, though a few were made through the half year, particularly in southern iron. The average level of quoted prices has been advanced, through the movement, but not over 25 cents, but taking cognizance of the fact that a number of important sales were made at lower figures than openly quoted the advance might be considered close to 50 cents. At the close of August the market is stagnant, but with the operating furnaces comfortably filled for two or three months, buyers being equally well covered. We quote: Southern No. 2 foundry at Birmingham, \$10.75 for small prompt lots, with \$11 quoted usually for such delivery, and firmly maintained for fourth quarter; No. 2 N foundry, Philadelphia, \$16.65@16.75; No. 2 X foundry, f.o.b. Buffalo furnace, \$14; No. 2 foundry delivered Cleveland, \$14.50@14.75; No. 2 foundry at Chicago furnaces, \$15; at valley furnace (no cents higher, delivered Pittsburgh) No. 2 foundry, \$14; malleable, \$14.25; basis, \$14@14.25; Bessemer, \$15.75@16; forge, \$13.50.

Steel

The fore part of August was very dull in unfinished steel, with former quotations maintained on billets at \$26.50 and sheet

bars at \$27.50, while rods dropped from \$29 to \$28. After the middle of the month sheet bars began to be offered at cut prices by the smaller mills while larger mills offered to make sliding scale contracts, based on the average price of basic pig iron to be disclosed monthly, which would net the sellers less than \$25. At this writing the large mills seem to be reaching the conclusion that they will have to accept much lower prices and it seems probable that in the early part of September billets can be obtained at \$23 or \$24 and sheet bars at \$24 or \$25, maker's mill, Pittsburgh or Youngstown, while rods are likely to remain at about \$28, Pittsburgh.

Finished Steel

As already indicated there is a distinct declining tendency in finished steel prices, touching a large number of commodities, if not the majority. The tendency is not well shaped at present, and may involve only slight reductions, or may eventually involve very considerable reductions. Meanwhile we quote former prices on plates, bars and shapes, though some of these products have already been rather openly shaded. Sheets have definitely declined further, in that prices formerly quite exceptional are now made much more generally. Wire products have declined \$1 a ton. Butt weld sizes (3-in. and under) of steel pipe have been reduced one point or about \$2 a ton. We quote prices as follows, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Common iron bars, 1.60 cents, Pittsburgh; 1.25 cents at eastern Pennsylvania mill; 1.35 cents, delivered Philadelphia; 1.40 cents, Chicago.

Wire nails, \$1.65, base; plain wire, 1.45 cents.

Sheets, blue annealed, 10 gage, 1.65 cents; black, 28 gage, 2.15@2.20 cents; galvanized, 28 gage, 3.20@3.25 cents; painted corrugated, 28 gage, 2.35@2.40 cents; galvanized corrugated, 28 gage, 3.25@3.30 cents.

Merchant steel pipe, 80 per cent off list for 3/4 to 3-in.

Steel boiler tubes, 3 1/2 to 4 1/2-in., 60 per cent off list.

Standard railroad spikes, 1.70 cents, Pittsburgh; 1.75 cents, Chicago.

Button head structural rivets, 1.90@2 cents; cone head boiler rivets, 2@2.10 cents.

The Non-Ferrous Metal Market

Since our last report metal prices have tended generally higher, and business has been transacted on a fairly large scale. The labor strike in the Lake copper country and threatened strike in the lead district of southeast Missouri have reacted in favor of higher prices. Spelter has been scarcer on account of the closing of some western properties at the time of the recent slump in the market, and prices for this metal also have been raised.

Copper.—Since the Lake strike, prices for Lake copper have been nominal and sales have been on a very small scale. The electrolytic market has been firmer and this brand of copper is becoming scarce for immediate delivery, although no premiums for spot metal have yet been asked. Lake is quoted nominally at 15 7/8@16 cents and electrolytic at 15.50@15.60 cents.

Tin.—Domestic demand has been increasing and large orders have been placed abroad. As a result spot metal is commanding a premium of about 1/2 cent and the quotation for August tin was 41 1/2 cents.

Lead.—Prices for this metal have advanced since our last report, and early in the month the market was active and strong. Later it was quiet and without special feature, with quotations at 4.37 1/2@4.40 cents, St. Louis, and 4.45@4.50 cents, New York.

Spelter.—A good demand has been in evidence and higher prices have been paid. Production is still below normal, owing to reduced output from western mines. The last quotations are 5.45@5.50 cents, St. Louis, and 5.60@5.65 cents, New York.

Other Metals.—The aluminium market is rather dull, and sales are only for immediate demands. Current quotations tend lower, being 22 3/4@23 1/4 cents, New York. The antimony market also is dull, with sales of small volume. Various brands are quoted at from 7.25@8 cents. Business in quicksilver is fair and prices are unchanged at \$40 per flask of 75 lb., New York, and \$39.50, San Francisco.

Eugene A. Byrnes

Dr. Eugene A. Byrnes, of Washington, D. C., a distinguished patent attorney and expert, died on August 1st at Haven, Maine, whither he had gone for his vacation. He died of acute peritonitis following an attack of appendicitis.

Dr. Byrnes was born in New York State in 1862. He graduated from the University of Michigan, and in 1884 came to this city, where he was for a time a teacher in the Central High School. Subsequently he entered the Examining Corps



THE LATE EUGENE A. BYRNES

of the United States Patent Office, and by his industry and ability rose to the position of Chief of a Division, being meanwhile graduated from the Law School of Columbia, now George Washington University, and was admitted to the Bar. He also took a post-graduate course leading to the degree of Ph.D. in electrochemistry and physics. In 1901 he resigned his official position, and entered upon the practice of his profession, taking as a partner Mr. C. P. Townsend, and later, also Mr. J. H. Brickenstein, the three constituting the well-known firm of Byrnes, Townsend & Brickenstein.

From the resolutions passed by the Patent Law Association of Washington on the occasion of Dr. Byrnes' death we quote the following:

"Modest and unassuming, but of a studious nature and with a fondness for scientific investigation, Dr. Byrnes was promptly accorded high standing among his professional brethren, by reason of his ability and attainments. He occupied the unique position in the patent profession of being at once a lawyer of ability and an expert chemist and physicist, whose opinion and advice were sought in matters of this character by leaders of the profession, and unhesitatingly accepted.

"Death claimed him in the prime of vigorous manhood, and with scant warning. Those whose privilege it was to come into contact with Dr. Byrnes, whether in a business or a social way,

bear grateful testimony to his unfailing courtesy and kindness, and to the fairness and candor of his nature. In his death this community loses a worthy citizen, the profession and this Association a fine example, and his business associates a loving comrade, and a strong and able co-worker. Words cannot express the loss to his wife and relatives."

This journal loses in Dr. Byrnes a good friend and ever faithful contributor. Right from the start of this journal in 1902 Dr. Byrnes has prepared the "Digest of United States Patents" (prior to 1902), arranged according to subject matters in chronological order. A little more than a week before his death we received from him his last manuscript. His Digest, a work of love and a by-product of his professional activity, survives him and to his many friends in the electrochemical fraternity who have used it and will use it, it will serve as a pleasant remembrance of the fine personality of Dr. Byrnes.

William James Evans

In the death of Mr. William James Evans, on July 23, New York chemists and the Chemists' Club have suffered a serious loss. Although Mr. Evans had been ill for some time he had partially recovered and his many friends in the Chemists' Club were delighted to see him able to visit the Club again. His death followed an operation which suddenly became necessary owing to a return of his former trouble.

Mr. Evans was born in Port Wilson, Ontario, Canada, August 31, 1852. In July, 1880, he entered the employ of Messrs. McKesson and Robbins, and was actively engaged as



THE LATE WM. J. EVANS

a salesman for some years; later he took up detail work among the physicians and druggists of New York City. Mr. Evans was a member of the American Chemical Society, of the American Pharmaceutical Association, of the Society of Chemical Industry, and of the Lotus and Drug and Chemical Clubs. He was the past president (1910-1911) of the Canadian Society of New York. He was an active member of the Schools and Universities Club of New York, and was the ever jovial president of the Fendosophs.

Mr. Evans joined the Chemists' Club on March 13, 1908, and was elected treasurer the following December. He served as treasurer during the years 1909, 1910, 1911, and 1912, and was particularly active in putting the club on a sound financial basis during its most critical period before and after the removal to the new building in Forty-first Street. In his own quiet and jovial way Mr. Evans has done an immense amount of good for the Club that shall never be forgotten.

From resolutions adopted by the Board of Trustees of the Chemists' Club we quote the following:

"The kindly and friendly interest which Mr. Evans took in the Club and all its functions, his sincere and cordial meet-

ing with all its members, and his efficient and unflagging co-operation in forwarding the interests of the Club made him not only a member of unusual usefulness but also a warm personal friend of each of us."

Everyone who knew him—and this means every New York Chemist—knows how true this is.

Mr. Evans leaves a widow, one son, and two daughters.

Personal

Dr. Richard Beck, president of the Freiberg Bergakademie, of Freiberg, Saxony, will be the guest of honor at a dinner of the Old Freibergers in America at the Engineers Club in New York City. Further particulars may be obtained from O. L. Bryden, 701 Jefferson Avenue, Scranton, Pa.

Mr. J. E. Johnson, Jr., has opened an office at 52 William Street, New York, as consulting engineer and metallurgist. He will devote himself to investigations and reports on iron and steel lines as well as to design and operation problems and to expert work in patent causes. Mr. Johnson has been connected for a good many years with iron and steel works, involving large power and metallurgical operations and has carried on important investigations. The more recent of these aimed at determining the causes of differences in quality of pig irons of the same analysis. Mr. Johnson will be associated with Messrs. Sanderson and Porter, engineers and contractors.

Mr. Milton M. Kohn has returned to New York City, from his European trip.

Mr. Philip J. Kroll has been appointed sales agent of the International Oxygen Company, manufacturers of oxygen and hydrogen generators of the I. O. C. system, for the Pittsburgh district. Mr. Kroll's headquarters will be at Room 1023, Park Building, Pittsburgh.

Professor Albert Sauveur of Harvard University has been awarded by the Franklin Institute of Philadelphia the Elliott Cresson Gold Medal, the highest award in the gift of the Institute "in recognition of his numerous and important contributions to the science of metallography and the influence he has exerted in bringing this science into practicable and exceedingly useful application in the iron and steel industry."

Mr. Edson O. Sessions has opened an office as consulting engineer at 5022 Sheridan Road, Chicago.

Mr. Frederick W. Sperr, Jr., until recently chief chemist of the by-product coke plant of the Tennessee Coal, Iron, and Railroad Company at Fairfield, Ala., has resigned to take charge of the blast furnace and by-product laboratories of the Inland Steel Company, at Indiana Harbor, Ind.

The First Mining Show and Industrial Exposition, to be held under the auspices of the American Mining Congress, will be held at the Horticultural Hall, Philadelphia, Pa., from October 17 to 25. Full information relating to the Exposition may be obtained from Richard L. Humphrey, Director, 1031 Land Title Building, Philadelphia, Pa.

The Multiple Unit Electric Company announces their removal from 136 Liberty Street, to 133-7 East Sixteenth Street, New York City, which is necessitated by an increased volume of business. In the new quarters will be installed improved labor and time saving machinery, which will greatly facilitate the production of their electric furnaces and heating appliances. While the company's products have received the flattering approval of the technical public, owing to their simplicity, durability, and efficiency, the company realizes that it was possible to improve upon their product, and one of the first moves in this direction was to air-cool the electrical connections and place them in position far removed from the heat zone. This means that the furnace proper need never be disturbed, and a simple removal of a heat unit produces a practically new furnace. There has been a change in the personnel of the company, and Mr. O. A. En Holm has been appointed general manager with complete supervision over office and manufacturing departments; Mr. M. M. Kohn being no longer connected with the company.

Evolution in Methods of Handling Slime.

VI.—Rhodesia

By H. N. Spicer

Rhodesia is probably one of the oldest gold-mining countries in the world, history proclaiming that King Solomon drew some of his gold supplies from that region. Nevertheless, today the mines of Rhodesia are profitable and far from exhausted;



FIG. 1.—GENERAL VIEW GLOBE AND PHOENIX 40-STAMP MILL

and from a metallurgical point of view the country is considerably more up-to-date than any other in South Africa.

Until within the last few years, Rhodesia has been regarded as a country for the prospector and the small company content to operate a small mill on a comparatively high-grade ore. More recently, however, owing to the development of some of the mines, and to the fact that several of the large Transvaal mining houses have interested themselves in the country and have acquired what appear to be the best of the mines, work has been undertaken on a large scale.

Mills of 1000 to 2000 tons daily capacity are now in course of erection.

Influence of Australian Practice

The development of cyanidation in Rhodesia has been greatly influenced by Australian practice, due to the fact that some years ago metallurgists and engineers from the latter country found their way to Rhodesia. Naturally they brought with them the ideas which they had developed and used successfully in Kalgoorlie and other Western Australian camps.

As a whole, Rhodesia is a dry country, though not nearly so dry as Western Australia. As a consequence the water problem is serious; and one of the main factors in successful milling is the use of a minimum of water and the conservation and re-use of that element. The Australian metallurgist was well qualified through long experience to tackle the problem in Rhodesia, making extensive use of plate-and-frame presses and exhaust-steam condensing plants. The present use of these devices in Rhodesia points to the prior education of the men who installed them.

Globe and Phoenix

The premier gold mine of Rhodesia is the Globe and Phoenix, from which handsome profits have been derived for a number of years. For a long time this property was considered the one exception to the belief that the ore-bodies in Rhodesia, though rich, were small and did not extend to any great depth. This mine is now being worked at a depth of nearly 2000 ft., and the published report of the company states that the ore-body continues to hold its own in width and value.

The Globe and Phoenix company has two mills: one for the treatment of ore directly from the mine, and the other for the

retreatment of the accumulated tailings from the first mill.

In Fig. 1 is given a general view of the first mill. In the left background is the building for water recovery by steam condensation. Directly in front of that is the 40-stamp mill, with the power-house at the right.

Forty 1250-lb. stamps crush the ore in water through 20-mesh screens, averaging about 6 tons per stamp in 24 hours. The average gold content of the ore milled is 1 oz. per ton. The crushed ore is first amalgamated and then ground in twelve 5-ft. Forwood-Down Australian grinding pans. The pan discharge is roughly classified into sand and slime in small spitzkasten; the sand is concentrated on Record vanners, making a product representing about 4 per cent of the total ore treated. Both sand and slime are passed over blankets to recover coarse gold, after which they are elevated by means of a double-acting 12-in. vertical plunger pump and delivered to a series of large wooden spitzkasten for final classification.

Slime Is Weathered Before Cyaniding

The underflow from the spitzkasten flows to four 30 ft. by 8 ft. leaching vats where the sand is drained and treated with cyanide solution. The slimy overflow from these vats is combined with the overflow from the spitzkasten and sent to dams for settlement. Water is recovered and returned to the mill. The slime has a value of \$4 per ton; but owing to the presence of a small amount of antimony it has not been possible to treat this material by cyanidation without first allowing time for oxidation by exposure to the atmosphere.

After the dumps have weathered for six months, during which time they are plowed and harrowed, they are treated by standard intermittent decantation at the rate of about 100 tons a day. The slime is mixed with cyanide solution in a vortex mixer, about 5 tons of solution being added to 1 ton of slime. Lime also is added at the mixer, at the rate of 4 lb. per ton. No other agitation is given than that received during the mixing operation, and the pulp is sent to two 35-ft. slime collectors. This slime does not settle readily, although 60 per cent of it is of a sandy nature. Two treatment vats are provided for each collector, and the usual decantation process is applied as described in earlier notes on Rand slime practice.

Concentrates Roasted and Cyanided

The concentrates from the vanners, amounting to 8 or to 10 tons a day, contain an average of 4½ oz. gold per ton. After settling and draining, this product is treated in a Merton roaster and then ground to slime in Forwood-Down pans. The ground



FIG. 2.—SAND RETREATMENT PLANT, GLOBE AND PHOENIX

pulp is thickened to a consistency of 1 part solution to 1 part solids, agitated in paddle agitators, and pumped to a Dehne plate-and-frame press. This treatment has proved very efficient, and an extraction of 90 per cent is obtained. Zinc shavings are used for precipitation.

Continuous Decantation for Reground Sand

The extraction made in the sand-leaching department is not good, owing to the presence of antimony in the ore, and the tailings are discharged with an average value of \$4 per ton.

This material has accumulated until the dumps contain approximately 250,000 tons of sand, worth say, \$1,000,000. It was for the retreatment of this product that the second mill was designed and erected. The plant is interesting, both from a metallurgical point of view, and because it is the first in South Africa to employ continuous counter-current decantation. It is now treating 350 tons of reground sand per day.

A general view of the plant is shown in Fig. 2. It comprises

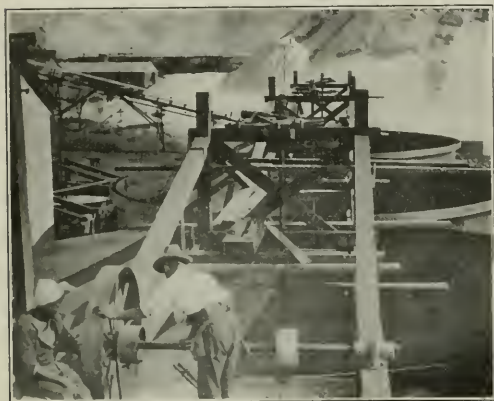


FIG. 3.—THICKENERS AT RETREATMENT PLANT, SAND DUMP AND BELT CONVEYOR IN REAR

three duplex Edwards roasting furnaces, duplex vertical 16-in. plunger pump, spitzkasten, twelve 5-ft. Forwood-Down grinding pans, blanket tables, cones, two 22 ft. by 5 ft. 6 in. tube mills, one 35 ft. by 12 ft. and four 40 ft. by 12 ft. Dorr thickeners, eight 10 ft. by 5 ft. paddle agitators, and centrifugal pumps. In Fig. 2 the roasters, plunger pump and one Dorr thickener are at the right, and four Dorr thickeners at the left. In the center is the structure housing the centrifugal pumps for the continuous decantation system. In the middle foreground are four of the agitators, the other four being behind the building. Fig. 3 gives a closer view of the line of thickeners, and shows also a part of the main sand dump, with inclined belt con-



FIG. 4.—ROASTERS AT GLOBE AND PHOENIX RETREATMENT PLANT

veyor for transporting the sand from the dump to the roaster bins. Fig. 4 gives a view of the roaster buildings, with the conveyor and bins at the right, and plunger pump at the left. A closer view of the paddle agitators is given in the foreground of Fig. 5.

Sand Roasted to Remove Antimony

As already indicated, the sand is removed from the dump and elevated by belt conveyor to an 800-ton steel bin at one end of the roaster building. Push conveyors feed the sand to

the roasters, where it is heated to a temperature sufficient to drive off the antimony. An interesting point in connection with the operation of the roasters is that the fuel is coal brought from the Wankie fields near the Victoria Falls on the Zambesi river. Although the coal has to be hauled by rail a very great distance, say 500 miles, it is found cheaper than wood, for the local timber has been removed for many miles around.

The roasted ore is carried by push conveyors to a sump where it is mixed with the solution overflowing the second thickener of the decantation system. This pulp, having a consistency of 5 parts solution to 1 of sand, is raised by the plunger pump to a large spitzkasten. The underflow is ground in pans and passed over blankets to remove gold that is too coarse to be recovered by cyanidation. The overflow from the spitzkasten passes to cones, the underflow of which is ground in tube mills. Finally, all the pulp from pans and tube mills, which is practically minus 150-mesh, gravitates to the pump and again passes through the classifying system.

Thus only the overflow from the two tube-mill cones passes to the continuous decantation system. The pulp is first thickened in the 35-ft. tank, then passes through the agitators to effect solution of the gold, and finally through the 40-ft. thickeners where it receives counter-current washing and decanta-

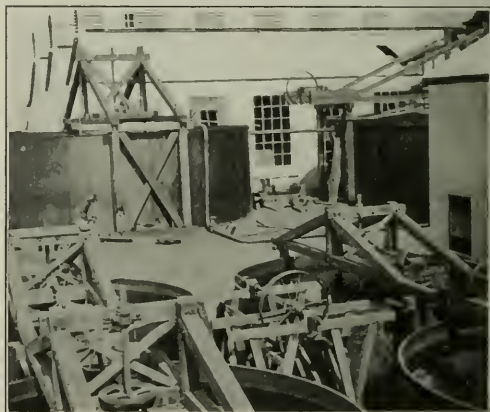


FIG. 5.—PADDLE AGITATORS IN FOREGROUND, GLOBE AND PHOENIX RETREATMENT PLANT

tion according to standard practice as used in similar plants in the United States.¹

All the thickeners are fitted with spring-alarm devices, which are of considerable advantage in the treatment of material which settles readily and which might accumulate so rapidly as to interfere with the movement of the rakes. The thickened pulp is discharged through 1-in. nozzles. All of the pumping from tank to tank is done by belt-driven centrifugal pumps placed in a central building. An extraction of from 88 per cent to 90 per cent is being obtained in this plant, and the operating costs are very low.

Lonely Reef Mill

About 60 miles north of Bulawayo is a comparatively small mine, the Lonely Reef, which has been a consistent dividend producer for a number of years. A general view of the mill is given in Fig. 6. The equipment comprises 15 stamps, to which 5 are now being added; three tube mills, amalgamating plates, four 20 ft. by 10 ft. Dorr thickeners, four 22 ft. by 8 ft. Pachuca agitators, and four Dehne plate-and-frame presses. Practically all of this equipment is shown plainly in the view of the plant, the stamps being housed in the large building in the background, and the filter presses in the build-

¹Continuous Decantation has been described in this journal: July, 1911; January, 1912; January, 1913; August, 1913.

ing at the right. A closer view of the thickeners is given in Fig. 7.

The ore is crushed in cyanide solution, through $\frac{1}{4}$ -in. screens, and classified in cones. The cone underflow is reground in tube mills and passed over amalgamating plates. The tubes have silex lining, but hard rock of local origin is used for pebbles. A closed circuit is established between the tube mills and cones, and the pulp is returned to the latter after amalgamation. The cone overflow is very finely ground, and from 90 per cent to 92 per cent will pass a 200-mesh screen. This product is thickened to about 50 per cent moisture, in which condition it is elevated by plunger pumps to the Pachuca tanks which are connected for continuous agitation. The treatment is finished by filter pressing, and the tailing is trammed to the dump as shown in Fig. 6. Presses are used in preference to



FIG. 6.—GENERAL VIEW, LONELY REEF MILL, RHODESIA

other forms of filter on account of their greater efficiency in recovering water.

Excellent metallurgical work is done at this mill. The ore contains an average of 1 oz. gold per ton, and the quantity treated daily is 150 tons. With the addition of five stamps the daily capacity will be proportionately increased.

Concrete Slime Collectors at Bushtick Mill

An interesting feature in connection with operations at the Bushtick mill, is the use of concrete slime vats, 85 ft. diameter and 7 ft. deep. A view of one of these vats is given in Fig. 8, which shows the cracking of the concrete due to the settling of the vat. Various materials have been tried for filling and patching these cracks, but pitch has been found the best.

The ore at the Bushtick is low-grade, averaging about \$5 per ton. Twenty ordinary gravity stamps and six Nissen stamps crush in cyanide solution, through screens of $\frac{1}{2}$ -in. mesh. The stamp product passes directly to tube mills, and after being reground is run over blankets which retain a rich

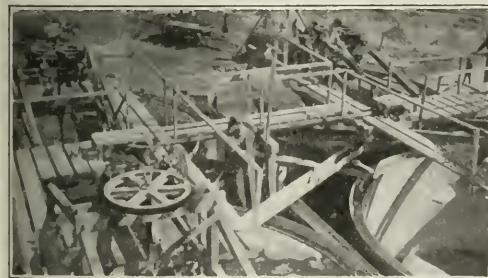


FIG. 7.—SLIME THICKENERS AT LONELY REEF MILL

concentrate representing a value of about \$1.50 per ton of ore treated. The tube mills work in closed circuit with pointed-box classifiers, the overflow from which passes to concrete settling pits. Here the sandy portion of the pulp settles and the slime overflows with solution, gravitating to a series of four 85-ft. concrete treatment vats, referred to above. The slime is treated

by the ordinary system of intermittent decantation, the pulp being transferred by centrifugal pumps.

All of the power used at this mine is generated by gas engines, direct-connected with electric generators. Gas is produced from charcoal.

A report of operations for the month of May, 1913, shows



FIG. 8.—85-FT. CONCRETE SLIME COLLECTORS, BUSHTICK MILL

that the mill ran 25 days, crushing 6,106 tons of ore. The total yield was 1,308 oz. fine gold, of which 1,068 oz. was recovered by cyanidation of sand and slime. The duty per stamp per 24 hours was 10.13 tons.

Crushing in Water at the Gaika Mill

There are several features of interest at the Gaika mill. Fifty per cent of the ore is a steatitic rock, very greasy and producing considerable slime. Trommel screens of $1\frac{1}{4}$ -in. mesh precede the stamps. The undersize, amounting to about 30 per cent of the ore treated, by-passes the stamps, and the oversize is crushed in water. The mortar screens also are $1\frac{1}{4}$ -in. mesh, which is the upper limiting size of the feed to Chilean mills. After being reground to 30-mesh in the latter machines, the ore is passed over amalgamating plates and blankets.

A view of the cyanide department at the Gaika is given in Fig. 9, showing the elevated sand-leaching vats at the right, Dorr thickener in the center, and elevated slime storage tanks at the left. Behind the latter is a structure housing the vacuum



FIG. 9.—SAND AND SLIME DEPARTMENT, SARKA MILL

filtration plant. A closer view of the top of the thickener is given in Fig. 10, which is reproduced to show the crank and pulley occasionally used to operate the thickener by hand. It sometimes happens that the power is cut off, in which event the machine is easily operated by hand at normal speed by a Kaffir.

Intermittent Discharge from Thickener

The Chilean mill product is classified in cones, and the sand is treated by leaching. The slime consists of 12 parts water and 1 part solids. It gravitates to the thickener which is discharged intermittently, say once an hour, to provide charges for the filter. Operated in this way the discharged pulp contains only 30 per cent moisture. It is then mixed with cyanide solution and agitated by circulation through a centrifugal pump connected with the filter storage-tank.

The filter was the first of the vacuum type to be used in South Africa, having been built in Australia and transported to Rhodesia. It is of the stationary type and does good work, but owing to the nature of the slime it is impossible to treat cakes of more than $\frac{3}{4}$ in. thickness.

New Mills Under Construction

As already intimated, Rhodesia has been and still is principally a country of small and rich mines; but the future mining prosperity of the region will depend more on the successful treatment of the larger deposits of low-grade ore. For this purpose several large mills are now being built.

The largest of these is at the Shamva mine, where a mill of

Flotation of Gold-Copper Ore

A thoroughly modern mill is now being built at the Falcon mines to treat daily about 400 tons of gold-copper ore. After crushing in Nissen stamps, the ore will be concentrated on tables to remove coarse mineral. The tailings will be separated into sand and slime, and the former reground in tube mills. The combined reground sand and slime will then be dewatered and concentrated by flotation, using the Minerals Separation process. This will be the first flotation plant in South Africa, and is suggestive of the recent advances in the concentration of slime and finely ground sand.

A number of other mills, mostly of small capacity, are in course of erection or design. The reader who has followed these notes will readily perceive the modern character of Rhodesian milling plants, and the up-to-date methods of handling slime for cyanidation and flotation, as compared with those in vogue in some other countries. The Rhodesian metallurgist is progressive, and is to be congratulated on results already obtained under adverse conditions, and on the prospect of solving further problems in ore-treatment in that country.

Denver, Colorado.

Carborundum Refractories

By F. J. Tone

Carborundum, while long recognized as a substance of remarkable refractory properties, has never taken an important place among the materials at the disposal of the metallurgist or furnace builder. This has not been due to the material itself but rather to the methods of manufacture into refractory forms.

The first to recognize the refractory properties of carborundum was its discoverer, Acheson, who describes it in his original patent as a substance possessing the properties of great refractibility.

The first patent describing the manufacture of brick and other refractory articles was Acheson No. 615,648, which describes the manufacture of carborundum crucibles, bricks and other forms by the use of highly ferruginous clays and other iron compounds. This was followed by Talbot, with a process relating to the production of neutral furnace linings using a bond of coal tar, by using especially amorphous carborundum. Other early patents were those of Engels, covering the coating of firebrick and furnace walls with carborundum and the manufacture of carborundum muffles; and the patent of Fitz Gerald, who invented a process of recrystallizing highly refractory and solid articles without any foreign binder.

Two types of carborundum refractories are known; one containing a vitrified or ceramic binder, the other consisting entirely of carborundum. This latter type is made according to the Tone patents, Nos. 992,668 and 1,013,700, which in general cover the silicidizing of carbon forms and their conversion into silicon carbide by subjecting the carbon to silicon-containing vapors at a high temperature. Patent No. 992,668 describes a preferred method in which silicon carbide grains are molded with an addition of carbon. When these bricks are heated in a mixture which gives off silicon-containing vapors, as, for example, in the ordinary carborundum furnace, the carbon is converted into carborundum, forming a dense, interstitial binder or matrix for the silicon carbide grains.

U. S. Patent No. 1,013,700 covers the conversion of pure carbon forms into carborundum, the resulting material being in all respects equivalent to the silundum described in Bülling's German patents. The latter process is applicable only to those forms of small cross sections. Silicon vapors cannot, without great difficulty, penetrate a pure carbon article to a depth exceeding $\frac{1}{4}$ in.

This type of refractory is absolutely unattacked by acids and is inert to the influence of practically all other chemical agents. It is, however, extremely expensive to produce and its use is limited to those places where the ordinary brand of carborundum refractory is not suitable. Thus for certain parts



FIG. 10.—THICKENER ADAPTED TO BE OPERATED BY HAND

2,000 tons daily capacity is nearly ready for operation. The ore will be crushed by Nissen stamps in cyanide solution; preliminary classification will be effected in cones, and the sand will be reground in tube mills. After final classification the slime will be thickened in Dorr machines, agitated in Pachuca tanks and filtered in a Butters vacuum filter. Zinc-dust precipitation will be used, according to the Merrill system.

At the Cam and Motor mine a large dry-crushing mill is being built. The ore will be dried, crushed in ball mills and roasted in furnaces of the Edwards type. The roasted ore will be mixed with cyanide solution and classified in cones. The sand will be reground in tube mills, and after reclassification of the pulp, the sand will be leached and the slime filtered by vacuum.

On account of the great scarcity of water in parts of Rhodesia, dry-crushing is likely to find wider application, and several mills of this type are being erected at such mines as the Antelope and Planet-Arcturus.

of electric furnaces, as, for example, for heating furnaces of the carborundum-resistance type, it is the only material suitable for placing in contact with the carbon resistor. It also has advantages in constructing electric furnaces of the Hoskins carbon-plate type, where, owing to the resistance to oxidation of the silicon carbide coating, the life of the carbon plates is increased five to ten times. It also has remarkable value for the construction of roofs of electric steel furnaces and furnaces of a similar type.

The vitrified carborundum refractory has, owing to its lower cost of production, a much wider application to the refractory industry. It consists of suitably chosen carborundum grains bonded with a special form of refractory clay, mixtures of grits and methods of molding being employed to insure a very dense body of low porosity. The brick are fired in kilns of a new design, giving a range of temperatures never before attained in commercial kilns used for the production of refractories. The clay binder is completely vitrified so that only a small quantity is necessary to bind the highly compacted mass.

Carborundum is a material unique among refractories due to the following distinguishing physical properties:

- (1) High softening or decomposition point.
- (2) High thermal conductivity.
- (3) Low coefficient of expansion.
- (4) Hardness and mechanical strength.
- (5) Resistance to acids and other chemical agents.
- (6) Comparatively low density.

High Softening or Decomposition Point

According to the most recent determinations crystalline carborundum forms at 1,840 deg. C. and dissociates into its elements at 2,240 deg. C., the silicon volatilizing off as a vapor, the carbon remaining as graphite. No fusion or softening occurs below the dissociation temperature, as is shown by the sharp and perfect forms of graphite pseudomorphs or "skeletons" left when carborundum dissociates.

Carborundum refractories retain their high mechanical strength practically intact at high temperatures, which distinguishes it from other refractories since most refractories undergo deformation especially when subjected to mechanical stress at high temperatures. Thus, for example, silica brick, chrome brick and fireclay brick soften at a temperature several hundred degrees lower than their fusion point, which is not the case with carborundum refractories.

When bonded with a clay binder carborundum brick have very great refractory value. However, it is recognized now that the ideal condition to be obtained is that the amount of binding material used be the least consistent with the requisite mechanical strength and that the vitrification temperature of the binder be as high as is commercially practicable.

Failure to realize these essential conditions has been the cause for the indifferent success of carborundum refractories where their use first appeared most promising. In general the refractoriness of a conglomerate refractory mass is a direct function of the amount and character of the least refractory constituent, and the recognition of these facts is what now permits the manufacture of carborundum refractories which are giving such satisfactory results.

Thermal Conductivity and Capacity

Carborundum refractories have a thermal conductivity five or six times that of fireclay. Wolgodine has determined the conductivity of various refractories expressed in gram calories per second per cubic centimeter per 1 deg. C.

Carborundum brick	0.0231
Magnesia brick	0.0071
Chrome brick	0.0057
Fireclay brick	0.0042
Silica brick	0.0020

We have been accustomed to regard heat insulation as a property in firebrick equally desirable with resistance to fusion, but a close study of furnace work reveals numerous locations

where heat conductivity is the chief desideratum. These are muffle furnaces and indirect-heating furnaces of all types, such as zinc, retorts, partition walls in by-product coke ovens, pottery saggers, bats, etc., kiln floors, recuperators, regenerators, etc.

The transmission of heat depends on the diffusivity which is given by the relation

$$D = \frac{K}{W'S}$$

where D = diffusivity.

K = heat conductivity in gram calories per cubic centimeter per 1 deg. C.

W' = density.

S = specific heat.

Based on a temperature of 1000 deg. C., the value of D for carborundum refractories is 0.0578, whereas for firebrick it varies from 0.0067 to 0.0094. That is, carborundum brick show an efficiency of six to nine times that of firebrick in the transmission of heat. Further, for a given time and for a given temperature of the atmosphere of furnaces more heat will be absorbed and a higher temperature reached than with fireclay.

The specific heat of fireclay brick is 0.2243, that of carborundum brick about 0.19. The density of a carborundum brick is about that of fireclay. The heat capacity is less and consequently the absorption and transmission of heat is correspondingly greater.

Low Coefficient of Expansion

Carborundum has a coefficient of expansion of 0.0000047 per degree from 200 to 900 deg. C. That of firebrick varies from 0.00000567 to 0.0000161; that of fused alumina is 0.00000866. Carborundum brick are practically indestructible from changes in temperature. White-hot bricks can be thrown into water without breaking. This property is made use of in building furnace structures where leakage, due to alternate contraction and expansion is an important factor. This indestructibility of carborundum refractories under differential heat treatment is also aided by the extremely high thermal conductivity which aids in dissipating the heat.

Hardness and Mechanical Strength

Since carborundum refractories have almost no coefficient of expansion, no porous structure is necessary and a high density is easily obtained. A porosity of 15-20 per cent is obtained against 25-30 per cent for ordinary fireclay brick. This results in extraordinary mechanical strength and combined with the great hardness of carborundum which is 9.0 on Moh's scale of hardness (diamond 10, corundum 9) gives the material a remarkable resistance to abrasion. Locations where this property is of paramount importance are the floors of coke ovens subject to the wear of charging and discharging apparatus, the bottoms of Lehr furnaces for making glass, the floors of heating furnaces and furnace parts subject to the attrition of rapidly moving gases.

Another factor influencing the mechanical strength at high temperature is the molecular rearrangements at certain critical temperatures. Tests show that magnesia refractories may fail at 1550 deg. C., chrome brick at 1450 deg. C., from this cause. Silica changes into tridymite at about 1200 deg. C. and is consequently subject to deformation from this cause.

Carborundum is not subject to any molecular changes of this character having been formed at 1820 deg. C. in a crystalline state and having no allotropic modifications.

Resistance to Chemical Action

Carborundum is absolutely unacted upon by any acid when in the liquid or gaseous state; the only active acid reagent being a mixture of hydrofluoric and nitric acids. It is, however, decomposed by alkali, but only when subjected to a high temperature. When used in metallurgical structures it develops a brownish maroon glaze which prevents any subsequent action of slags, fused ashes, etc., and also practically prevents

any oxidation taking place. This glaze results from the interaction of the carborundum and the binder and as it is developed in the body of the refractory is extremely tenacious and impervious.

Density

Carborundum has a specific gravity of 3.1 to 3.2; when made up into refractory form its density is 2.1 to 2.5. This comparatively low density lessens the mechanical stress developed at high temperatures owing to the small weight of the refractory structures.

The Carborundum Company, Niagara Falls, N. Y., has placed on the market the two brands of refractories mentioned above. The name "Carbofrax" brand has been given to that formed of carborundum with a citrified binder and the name "Refrax" brand to that formed of carborundum entirely. Both varieties can be furnished in any regular or special forms, as regular, arch and key, brick, retorts, crucibles, base blocks, muffles, tubes, etc.

Hydrometallurgy

Joy of Its Theory, Woes of Its Practice

By Regis Chauvenet

Hydrometallurgy is a broad term if it be held to include every process in which the metal to be won is, at any stage of the operation, in solution.

The present article does not deal with any one process, whether successful or the reverse, nor shall we attempt to describe operative details, further than to indicate their general scope or chemical significance. We wish to point out why some well-heralded processes are relegated, not exactly as "have-beens" but rather as "never-weres," and why it is to be feared a similar fate awaits more than one attempt now in the course of evolution, i. e., in the hopes of its stockholders.

The difficulties met with in solution processes are many. For the most part they are not based upon theoretical error. The troubles encountered, a few of which we shall present, are partly due to manipulatory obstacles, partly to underestimates of expense. We cannot specify individual cases, whether prospective or operative, for obvious reasons. Nor are the difficulties to be described peculiar to any one process, or even to any allied group. We hope to show in what directions miscalculations have been common, as a warning both to over-enthusiastic investors and to prospective investors.

Every metallurgical process involves the operation of chemical laws. This may be generally recognized today; possibly not always in times past.

"In Spite of Science"

In a by-gone day the writer, then in St. Louis, Mo., was visited by an official of the Missouri Tin Company, a corporation formed for the purpose of extracting tin from porphyry. In justice to the promoters, they did not so state it in their prospectus. The functionary aforesaid, after much earnest protest against the assertion that there was no tin in the porphyry, a statement for which the writer was responsible, wound up with the following amazing dictum: "Why, we know as well as you do that the tin in our ore isn't susceptible of chemical detection."

It was almost prophetic—for did not the renowned "Metallurgical Murphy" years later, issue his immortal report: "The ore is of metallic contents, but basic constituents. Its values, while susceptible of chemical detection, are not capable of metallurgical extraction!"

Chemical Ignorance

We have said that difficulties encountered in actual practice were for the most part not based upon theoretical error. However, this is hardly true of many processes which are still in the purely prospective stage.

By "ignorance" of chemistry we do not mean total ignorance of text-book knowledge of that subject. It is a frequent happening, however, that the inventor of a chemical process knows

just enough chemistry to harm him. He has read of, possibly practiced, certain reactions, and tries them under conditions wholly favorable to their success. Of reactions in complex solutions, or the influence of mass; of the solvent powers of his own solutions for supposedly "insoluble" substances—of a hundred other complications, familiar to all operative chemists, he knows little or nothing. We divide the style of ignorance too commonly found in attempted chemical metallurgy, into "scientific, manipulatory and financial." In the following examples of the "left-wing" cases (i. e., frauds pure and simple), we divide our criticism between the fakir and the dupe; for the latter ordinarily has ample means and opportunity to discover in advance the false or fraudulent grounds upon which he is asked to invest.

Frauds, Fakirs and Fables

In the above classification we have not included the ignoramus "in toto." We nevertheless shall give one frightful example of him. King Solomon has disposed of him in a well-known phrase, which we may put into modern technical language by saying that one may pound him to forty mesh in a ten-stamp mill without changing his mental characteristics. Most metallurgical men will recall without difficulty several cases of attempts at the impossible. We have at the outset to dispose of the ever-recurrent metallurgical fraud—fool he may be, fraud he certainly is—whose mission is to create matter, or in brief, to extract metal, not from where it is, but from where it isn't!

"Ven ve are dead," said poor old Rip van Winkle, "how soon ve are forgotten!" And these recurrent spasms of alchemistic foolery, which appear

"Each in their turn, to make the vulgar stare,"

how soon they are forgotten in this busy world. Yet, within the memory of thousands of living men, we have had three schemes of continental fame, and half a dozen of more local celebrity, all claiming the production of gold, some under one, some under another disguise, as to method; but all in fact, if not in form, claiming the rediscovery of the art of transmutation.

(Strange, by the way, that all of these "transmutors" or what-ever-you-may-call-em's, never discovered any transmutation except into gold. Possibly one or two of them may have included silver.)

For the strictly technical reader, it may be a waste of time to read over accounts of mere trickery. Nevertheless some of these now historical transactions are reproduced. "To what end?" may be asked. Well, as a very slight contribution to the science of psychology. There is no appeal to science in these wornout juggler's tricks, but if we venture to classify them, as we think we can, and show that they fall under certain definite forms of appeal to ignorance, prejudice, credulity or greed, we may possibly indicate to some readers the ear marks by which he may recognize a fraud, without having too closely to investigate the alleged technical details.

"Classified Ignorance"

These hydrometallurgical attempts then, in so far as they try to get the general public interested, are divided into:

- (1.) Appeals to total ignorance.
- (2.) Appeals to anti-scientific prejudice.
- (3.) Assumed appeals to scientific principles, under a further assumption that the inventor (and he alone) has learned how to apply these profound truths to practical ends

First: Appeal to sheer ignorance. The best example that occurs to us is that of the "Wynne" process, started something over a decade ago in Denver, Colo. The promoters so far succeeded that they actually sold stock in the process to no small amount. The "recipe" was kept secret, but the results were apparent. Prospectors and mine owners were invited to send in samples of their ores, and did so. Invariably the "return" was made showing values far in excess of report made by any other assayer. But the new concern went further than this, in many instances it actually turned back to the sender of the ore sample the gold "extracted."

Not the "ignorant class" if you please, but some men of prominence and of presumable education were taken in by this particularly stupid trick.

"Of course," you will say, "there was some attempted explanation of the phenomenon?" No. Not so far as the public who were invited to invest were concerned. On its naked, self-asserted production of gold from rock, barren or nearly barren, the delusion found many believers. It would seem that the return of the metal in a few selected cases was productive of "results."

The "inventor" died. His stockholders were heirs to his chemical "recipe." They tried it, reporting, with refreshing naïveté, that it wouldn't work." No wonder. It is before us now, a silly mess of potash, nitric acid, ammonia, lithium carbonate and other ingredients. Possibly a fair bug poison, but fit for nothing else that we can imagine. The stockholders pocketed their loss and the public forgot about it in a fortnight.

Second: Appeal to anti-scientific prejudice. We should like to annex the proper name to this little bit of local history, but we regard the "feelin's" of some possible readers, who may still be a little sore. We shall call the camp by what it is in fact, not by its map name. Busted Camp, then, had "ore" and had also many enthusiastic promoters. The "formations" were identical with those of Cripple Creek. (Of course, for the discovery was made in the days of first development of the wonderful riches of the latter famous camp.) Everything was lovely at Busted, except that there was no gold in the ore. However, the settlers were true "Boosters" and weren't going to let a little thing like that discourage them. They invented a new assay method, which they called a "color method," but we shall not try our reader's patience with a description of the fool thing. Suffice it that the great argument used was based upon the fact that no "scientific" man or "regular" assayer or chemist, could find the gold here. That, it seems was the main point of value!

For, there is no doubt about it, there is an element in every community who have deep-seated convictions that "science" is a fraud. They could not define "science" to save their dollars—but as a noted foreign agitator said as he landed in New York, "whatever your government is, I'm against it," so there are certainly those who say "whatever science says about this, I don't believe it!" And of these were the investors in Busted Camp: the "poor but honest" and later, bankrupt people, who invested in their own ignorance—and lost.

The perpetrators of this fraud knew, of course, what they were doing. They also knew how to calculate upon ignorance and prejudice. That is the characteristic of fraudulent promoters everywhere. They are shrewd enough to gauge the prejudices of those with whom they deal. They descend to their level in their solicitations. Finding people who were already prepared to believe in a thing in the *precise ratio in which it had no scientific bearing*, they play upon that particular form of ignorance, and by skillful sneers at "scientific" work, induce the belief that work which is *not* "scientific," must be correct!

Is the logic apparent? It was at least successful, and many a "share" was sold. It was another proof of the prevalence of scientific illiteracy. We may roughly state the proportion of scientific illiterates at ninety per cent of the population. The only connection of this with "hydrometallurgy" is found in the fact that the assay method by which they found the gold that wasn't there, was a "wet" method, and that the proposed method of extraction was also "hydrometallurgical."

The Ocean a Gold Mine

Third in our classification comes the assumption of scientific principles. The "Electrolytic Salts Company" had a wonderful history. Mr. Jernegan issued circulars stating among other things that it was "well known" that sea water contained gold. The whole statement was a mixture of falsehood and of semi-truth. The object of the enterprise was not to extract sun-beams from cucumbers, but gold from the ocean.

It is indeed a "well-known" fact that for many years a

paragraph has been floating around the world stating that sea water contains a definite proportion of gold. One metallurgist has taken the trouble to trace the statement to its source, with the result that the trail "ran up a tree." No authoritative statement of the contents of gold "per ton of ocean" could be discovered. But it was enough to start the remarkable exploitation now briefly to be described.

The inventor stated the amount of gold to be one grain per ton of sea water. Possibly there is some infinitesimal trace of gold there. Certainly it is not that much—but the amount matters very little. The ingenuity of the attack (on the public) consisted in the declaration that the inventor was well aware that he couldn't handle a ton of material for four cents, which is about the value of one grain of gold. He added however, that if we could persuade "Nature" to do the job for us, that little difficulty would be overcome.

Nature has so shaped the Bay of Fundy that tides of extraordinary altitude characterize its upper reaches. "Here we shall place our mill," said the inventor.

The tanks or reservoirs to hold the sea water are to be so placed that they will fill at high tide, and empty themselves at low tide. Here is Nature's handling of the material. But how about the extraction of the gold? Why, that was so simple that "it was to smile."

Here we have a gigantic battery of one fluid (the sea water) and one cell (the reservoir). There were, of course, "plates" for the deposition of the gold. In flows the water, down goes the gold. The only "work" is to scrape off the beautiful stuff—and there you are.

It was indeed a lovely scheme. The hokus of the original statement aided by the pocus of the "battery" needed only the locus (Bay of Fundy) to bring it to a focus (subscriptions and stockholders). And the subscriptions flowed in: the mill was actually built. The scoffers scoffed, of course, but suddenly they were put to shame. Lo, a dividend! A real, fifteen-thousand-dollar dividend, paid in hard cash. With it, perhaps the most ingenious statement of all, viz.: "The yield was a great disappointment, being but one-fourth of the total gold content. From this trivial portion the dividend has been declared. But the cause of the small yield has been discovered, and the matter is easily rectified," and so on.

Now indeed, money came in fast. There had been just enough apparent appeal to scientific principles to satisfy those to whom electrolytic operations are a fact sufficiently well known, but who have never bothered themselves as to any details. Doubts were dispelled by the money. The stale old trick of a dividend paid from stock subscriptions had worked once more.

Then came the closing of the mill, the flight of the promoter to a more congenial climate, and other highly uninteresting details. It is said that the fugitive was caught abroad, but we are not able to give the closing chapters of this miserable history.

Enough of these flights of metallurgical fancy which are pardon a mixed simile, but sample bricks (gold bricks) from many an edifice of fraud, testifying at once to the credulity and scientific ignorance of a supposedly enlightened public. It was not to discuss such baseless frivolities that we started to pen this brief essay. There are plenty of cases outside of mere trickery, and to some of these we now turn our attention. Many processes are predestined to failure whose theory is fair, and whose intent is honest.

Trouble Begins at the Mine

We wholly omit the cyanide process. It constitutes a class by itself. This being understood, we must add to save misapprehension, that we do not undertake to criticize any one process. Our object is to point out certain difficulties inherent in most operations in which "wet" methods are in use, characterizing no single case, that is, no one as apart from most of the others.

Most "wet" methods may be said to be attempts to repeat on a large scale the operations of the analytical laboratory. Remotely, they exhibit, in their experimental stage, the same

relation to the commercial plant that a model does to the actual machine.

It is a matter of common remark among mechanical engineers that the perfect working of a model by no means insures the successful operation of the "real thing." We may say the same regarding the working of a chemical process, albeit the reasons are not quite the same for failure in the latter case.

The fallacy is not always apparent at first glance. There is usually an experimental plant in miniature, in which everything works nicely, for we are now assuming that the prospective working plant is based upon actual results obtained in a small way. This again, implies that the chemical principles are correct. The reasoning which carries us to the larger operation is exceedingly simple. Here is a result obtained with one ounce—no matter of what. Multiply by thirty-two thousand, and we have the outcome of the same operation on one ton. But we are seeking the erection of that favorite unit of the metallurgical inventor, a "hundred-ton mill." Very well. Multiply the last figure by one hundred—and there you are!

In this little model plant, solutions are made in vessels quite impervious to the action of chemicals, whether solvents or precipitants. Precipitation and filtration proceed rapidly and regularly. When you need a hotter solution, all you have to do is to turn on a gas jet, the cost of which (on a few liters of solution) is too trivial to take note of. Your little battery, too, how small a bother it is. Now and then an anode may wear down, but you have a box of new ones handy.

An unfortunate necessity of most hydrometallurgical processes is uniformity in the composition and tenor of the ore treated. Hence our caption, "Trouble begins at the mine."

In smelting ores, there may be a large margin of variation, within which it is no great trouble or expense to change treatment according to the laboratory reports. Take lead smelting. The furnace has been running on ore with little or no zinc. That metal now appears in something more than the trace or the very low percentage which has been the rule. The difficulty is met on the charging floor, with a slag calculation regulating the charge: an increase in the iron, a decrease in the lime, probably an allowance for less total silica. The intractable element is fluxed off, not quite as easily as the ordinary constituents, but certainly without visible interruption of current operations.

Not so in our "hydro" scheme. We may be getting our metals into solution by direct action, or possibly by previous chemical preparation (ex. gr. chloridization), but when a new constituent, or a marked change in proportions of old ones comes along, the changes involved are complex. Later we shall return to this point in another form, showing the necessity of too much "expert" work in "scientific" plants.

The differences in treatment due to change in composition, form a theme of too much detail to be taken up here. All we want is to present such changes as one of the "Woes" of hydrometallurgy. A great custom smelter can throw all sorts on its bedding-floor, and come out with a fair average. Rare indeed would be the instances in which this could be done by a wet-treatment works.

One class of material for wet treatment consists of ore too low for smelting and (if gold) not fit for amalgamation. The first question arising is whether the raw material can be directly treated or whether some roast or other preparation should precede. If the latter, we have our first trouble, before the method comes into play at all, for the roasting of low-grade ore means the employment of most of the fuel for keeping the big barren gangue heated.

Whether roasting has or has not preceded solution, we have as difficulty number two, the repetition in another form of the fuel waste involved in roasting a low-grade ore. Treatment of the ore with any solvent means the application of that solvent to all of the barren gangue as well as to the ore proper. This brings in a question to be met more in detail a little later, viz. the trouble arising from too great a dilution of solutions.

The difference between large and small operations in the analytical methods of ore treatment are barely indicated in the

above preliminary remarks. In fact, the trouble recurs throughout, and we cannot stop at every step to call attention to it. Like the poor, it is "always with us."

"But cannot we concentrate the ore before treatment?" This is a suggestion often heard, and a favorite one with promoters of wet methods. Let us look at it for a moment.

These initial difficulties are not really overcome by concentration. It is true that so far as cost of extraction per unit of metal is concerned, the concentrate is far ahead of the raw ore. Suppose a ratio of twenty into one. We have met with more than one enthusiastic inventor who would figure on his extraction cost on a concentrate, but who became very shy when mining cost was inquired into. But the fact that ninety-five tons had been mined for the five he had treated stood there all the same, like a giant with a big club, guarding the very entrance to the process.

So the operating expense killed the treatment of the raw ore, and the mining expense killed the concentration proposition. Our critic, standing at our elbow remarks, "sarcastic like," that the mining expense kills it in either case. True. But we are trying to be kind to the process, and grant that if it worked on the concentrate, "it was the mine's fault" if the outcome was bad.

Of the many preparations of minerals for solution, we select chloridization, not because it has been much practiced but because certain operations following the initial treatment illustrate the difficulties of chemical treatment in several necessary proceedings. "Chloridization" by the way is a somewhat new term. As it sounds affected, as against the simpler "chlorination," let us here explain that we shall use it to mean the treatment of the ore by "dry chlorine," a very different thing, chemically, from "chlorination" of a wet pulp.

Chloridization has been tried in England, Germany, Australia and the United States. That is, it has been experimented with, though it can hardly be said to have come to a commercial basis. That fact does not exclude it from either scientific or practical investigation. We are not greatly concerned just now with the commercial failures or success, we wish to show the nature of the obstacles which will have to be encountered.

Finally, we wish to emphasize, in order not to be misrepresented, that the difficulties in chemical manipulation on the large scale which we shall partially detail, are not presented as peculiar to the chloridization process. They inhere in any scheme in which it is sought to separate metals from one another starting with a solution containing all of them; in short as already indicated, any scheme which approaches in its details, the analytical method for complete separation of all the constituents of a substance under analysis.

The operation starts by treating the ore (usually in a revolving cylinder) with dry chlorine. This probably has been tried out more in England than in any other country. In the United States we have two plants in the experimental stage, one at Helena, Mont., the other at Georgetown, Colo.; there may be others, but we have not heard of them. Although a little aside from the scope of this article, we may note that several methods of getting the chlorine have been proposed. The most obvious is by electrolysis from common salt, $\text{NaCl} = \text{Na} + \text{Cl}$. The sodium, however, in the presence of water becomes caustic soda (NaOH), and hydrogen is liberated. The reason for mentioning this part of the operation is to remark that it makes quite a difference in the net cost of the product, whether the current at the plant is cheap or dear, also whether there is or is not a ready market for the caustic soda. If there is not, can it be made available in the precipitation of some element in another part of the scheme?

Cost of chlorine produced in this way, is always a somewhat uncertain figure. "One cent" is a common claim; it may be doubted whether this has ever been realized. Two cents a pound is nearer to actual experience. If the caustic soda is used in furthering the process, its value (if it is to be subtracted from the gross cost of chlorine production), is an uncertain factor, greatly depending, we should say, upon the point of view of the promoter.

Another source of chlorine has been zinc chloride, the zinc of course to be regained ultimately, or indeed, directly from the original electrolysis. The metallic zinc thus obtained is again put into solution, being used for the precipitation of metallic lead, i. e., $PbCl_2 + Zn = ZnCl_2 + Pb$. This would be a queer proceeding, economically, were it not for the fact that the relative weights of lead and zinc are at 207 to 65.

Not to anticipate however, let us consider some of the reactions following solution of the chlorides after the reaction in the cylinder is complete. Take the proposed sequence in an early attempt at Helena. It included at one point the complete separation of the iron, as ferric hydroxide. The reagent to be employed was nothing more than the caustic soda made on the spot, and tapped daily from the batteries which evolved the chlorine. Of course, the chemistry is quite correct. Now as to handling the precipitate. We have a hundred-ton mill, and our ore contains let us say 10 per cent of iron. Who shall face the handling of this terrific precipitate? The ten tons of iron make nearly twenty tons of the hydroxide. Have you ever precipitated one gram of metallic iron as ferric hydroxide? It needs about a liter (say a quart) of water for its treatment. Allow for filtration and washing a total of only half as much solution as indicated above, and we would have a daily handling of six hundred thousand gallons of water, and cold water would not suffice. For decently quick work in precipitation and especially in filtration we need hot solutions. No wonder that the genius of more than one inventor has been at work to modify or remove this bugbear. It has in fact been met in a later process at Helena, and a far better procedure adopted. Elsewhere, we are told, this hydroxide precipitation is still proposed albeit not in this connection.

Inevitable Losses

Going backward for a moment in the operation, it must be remembered that at every stage we shall have losses. These are not necessarily due to careless operations. Take the original solution of mass from the cylinder. Let us suppose that we have lead, zinc and iron as main constituents, and 50 per cent gangue. Lead is present as chloride. Wash it out. You may be pretty sure of this, that when the lead is washed out all the other chlorides will be also. (We are neglecting silver for the moment.) But this washing uses hot water, as $PbCl_2$ is insoluble in cold water.

In the Helena process, heat is applied long before this, i. e., to the cylinder itself, in order to promote the decomposition of sulphur chlorides.

The absolute necessity of heat at various stages of solution processes is an expense much overlooked. Few who have not tried it are aware of the enormous gain in time and clean work, in precipitating and washing, of hot solutions ever cold. Yet water persists in holding on to its high specific heat, and few low-grade processes are there but that will go down, economically, under this expense alone.

But those losses? When we have washed out a total of 90 per cent of assay value we have done very well. For it will not do to assume that our metals are wholly converted to soluble forms. Again when we have precipitated and filtered a big tonnage of ferric hydroxide, we shall be doing not badly if our filtrate contains 95 per cent of theory, for we cannot wash as would an analytical chemist. We shall have other precipitations. At every one, another deficit. After two operations (in addition to original solution), we may figure something after this fashion, "Ninety per cent of ninety-five per cent and ninety-five per cent of the latter weight." That would be about eighty-one per cent. These successive deficits in work of this description are to be overcome only by very clean and hot precipitations, and enormous amounts of wash water. But the latter brings about a new source of expense, not alone in heating the water, but in the unbearable dilution. Tanks upon tanks and then more tanks. There is hardly an operation of the kind that is not "up against" some such questions all the time, ex. gr., whether to leave a constituent behind, losing it, or wash it out at an ultimate expense greater than its value, when all of the extra installation, time and handling is accounted for.

We said above that much ingenuity had been called into play to avoid the necessity of handling the enormous iron precipitate necessitated in this and other similar operations. We shall pass over this rather quickly.

As to methods for disposing of this great bulk, one is simple enough in principle, being "not to produce it at all." This is effected in the Malm process by roasting out the chlorine from the iron chloride, thus regaining chlorine which is simultaneously used for attacking fresh ore. The iron oxide now left behind is insoluble. As an economic procedure this shows one striking advantage, for if we precipitate hydroxide with caustic soda, we get the chlorine back in its original form of sodium chloride, that is we have obtained an expensive reagent combined in the form from which it came.

Another method is to use as precipitant a substance which yields a much less bulky product than by caustic soda. Zinc oxide will do this. It has also been proposed in this connection to use zinc ore direct, calcined carbonate or even silicate¹. The topic is an enticing one, but we decline to be tempted aside by it, further than to say that the last named proposition seems to offer a new set of difficulties, we hope not too serious for economic outcome.

This case is presented merely as a type. Iron hydroxide is not the only precipitate which may have to be dealt with. If we have shown that precipitation, filtration and washing, the commonest of operations in analysis, are by no means such simple matters as often assumed in treatment of heavy tonnage, we have accomplished one object of our essay.

As for the general position that the methods of the analytical laboratory are not fitted for large operations, so that arguments fit for grammes are very poor ones as applied to tons, we have already mentioned it. We warn any one who may be invited to aid in projects of this general nature to look to this feature. The argument against a solution process which is based upon no other grounds is insufficient, *taken by itself*.

Having then indicated (1) the difficulties encountered in treating precipitates, (2) the necessity of applying heat at many stages, (3) the losses to be figured on at each stage, we may pass on. In dropping these points we may add one more little note of warning, viz: nearly every inventor figures upon a higher final product than experience warrants. There is a very familiar old argument, still used and still doubtless to be used for many a year, against shipping ore to a smelter. The argument reduced to its fundamental basis is about this: "Do not pay other people for the treatment, but treat your own ore. If they can pay for the ore and make a profit out of the treatment, that last is money that should be in *your* pocket, not in theirs." Strange that this argument should have been many times so misapplied that certain mining enterprises have actually lost money for years on it, yet we believe it to be demonstrable that such has been the case. The general (i. e., the apparent financial) argument has overcome the really technical one. The two points for comparison are, (1) What will treatment per ton cost you, (2) What will be your percentage of production, compared with assay value?

Then as to the "other fellow," who offers to treat your ore for you, the same questions with hardly any variation, viz: (1) What will he charge you per ton for treatment, (2) What will be your actual yield?

Stripped of all the details of freight and other charges this is what it comes to. Aside from treatment-cost look at the element so often lost sight of, the fact that smelting regains the highest percentage of any process, and then make a very simple calculation, the object of which is to find out whether after all it doesn't "cost more than it comes to."

Some may challenge the statement that smelting shows the highest yield of any process. Of course we mean that for a statement holding true of average cases and average conditions. That there are ores so conditioned by constitution, grade and locality that smelting is impossible, is well known. But we know that ores have been locally treated for years which might

¹It is proposed to use this scheme at Helena.

have been shipped for custom treatment at better profit.

A few chemical difficulties remain to be disposed of. One is the troublesome "entertaining" of an element of value in the precipitation of another. This may be avoided in the analytical laboratory to a great extent. The means used is usually dilution. Hot solutions as a rule are less liable to this "trick." It cannot be lost sight of, being indeed only a special form of the "losses" already mentioned.

Incident to the question of the handling and washing of great masses of material, comes the ever present question of the treatment of very dilute solutions. Here again nothing can be settled except under discussion of the special conditions. There may be two reasons for thorough washing of a precipitate, one being to secure a pure product, the other the presence of an element too valuable to waste in the washings. Both of these reasons may apply in a given case. Nice figuring is called for, or it may be necessary to appeal to actual experience to decide when a solution is past the point of profitable treatment and may be better sent "down the creek" than further bothered with.

Continuous Operation an Essential

A very broad question (and difficulty) arises in the sequence of operations. It is assumed in any operation involving several steps, that each and every one delivers its daily or hourly quantity to the next one in order. The original treatment sends along its solution for precipitation let us say, the second one delivers its precipitate on the one side and its solution on the other, each for further treatment. Here however, occurs some stoppage. No matter what the nature of the hindrance, the general happening is now this, viz: all operations "precedent" come to a stand in a very few hours, i. e., until the obstruction is removed. We do not pretend that such troubles are peculiar to hydrometallurgy, but we believe that they cost more here than anywhere else.

A stoppage of this kind of which we were a witness arose from the sudden discovery that a certain element was present in solution which had no business there. It was small, it was of no value, but its presence would certainly give an "un-merchandiseable" color to the next product which happened to be zinc, in one or another form according to demand—oxide, carbonate, chloride. How did the objectionable element get there? Here is another of the small miseries of detail. Possibly the solution, containing many reagents and compounds derived from the ore itself, constituted a solvent for the troublesome intruder, which should have been (but wasn't) wholly included in a previous precipitation. No matter how—no matter why. It was impossible to send along a "dirty" solution. The processes ahead stopped for want of material. The processes behind stopped for lack of space to work in. The "experts" (we shall pay our compliments to them presently) have to quickly determine how best to rectify the mistake.

Batteries, Efficiency and Losses

This is delicate ground, on which the professional electrician may challenge almost any statement. We shall not assert that the cost of a given current is commonly underestimated, but when it comes to the annual round-up, which must of course include wear, tear and repair, leakage (both the "juice" and occasionally the chemical solutions), and net chemical product—in short net efficiency, the results are often disheartening.

Many loose assertions are afloat regarding electrolytic decomposition. Take a simple case, based on the original law of Faraday, copper chloride versus cuprous chloride (CuCl_2 and Cu_2Cl_2). "The same current that decomposes the one decomposes the other, so that if we reduce the copper solution to cuprous condition we get twice as much metal deposited for the same current." Certainly. The theory is perfect. The "kation" is copper, the "anion" is chlorine. And the kation is twice as weighty in the cuprous salt. You would indeed get twice as much copper for the same current, but for the fact that you don't. You will do very well at 70 per cent increase instead of 100.

Then there is a thing called resistance. Take the very case we started with, decomposition of common salt. Into the

theory of its progressive decomposition we do not even attempt to enter. It is well known that as the operation proceeds the current required for a given net evolution of chlorine (and corresponding formation of caustic soda) increases for each unit produced.

For every such decomposition there will be a practical limit, determined for any given works by experience alone. In the case under consideration it may be somewhere between 60 per cent (of total decomposition) and 85. Besides the battery question we have here the additional one of the very impure caustic we have to use, i. e., if it is to be used in subsequent operations.

Incidentally, we have never seen this feature squarely and fairly considered in pre-calculations, i. e., when made by the projectors. There is also a sort of scheme always indicated—perhaps a little dimly—to regain all of this salt at the "final end" of the process, i. e., when the last of the valuable metals has been extracted. As a rule, however, these final dilute salt solutions will "go down the creek."

How Many Experts to the Ton?

No organization of a works running on chemical principles but that would include the scheme of chemists—analytical, operative and supervisory, in its employ. But how many? Most schemes include "wages" in their estimates. There is more call and more constant call for the skilled part of the work in hydrometallurgy than in any other line we can think of. The laboratory force would be at work unceasingly, as a matter of course. But is that all? Far from it. Ordinary labor cannot be depended on in such operations. There is "hurry work" from the time a defect is discovered to the time it is rectified, and, as indicated above, a stoppage of many operations while the remedy is sought and applied. In fact a chemist on each and every landing of the mill would not be amiss. We need an "expert" to detect the trouble, another one to locate it, and a third to make the necessary analysis, possibly all three have to call in number four whose function it is to quickly apply the remedy, whether medical or surgical.

The dream of a plant of this kind running automatically and smoothly, taking in a given weight of ore daily, turning out the appropriate amount of metal ready for shipment, is beautiful; but alas, it is a dream. That is, unless you have a very large force of human experts with both brains and hands to assist the "automatic" running.

Consider the complexity of the case (independent of the particular process) which must always include so many different functions that their mere enumeration is tedious. And of all these functions there is not one that does not demand chemical knowledge, quick perception and skillful manipulation.

You must know that your original solution is practically complete. You must know the same of each precipitation. You must ascertain the purity, as you go, of each product, whether final or intermediate. You must carefully avoid any excess in the use of chemicals.

On the mechanical side, exercise an eternal vigilance for leaks whether liquid or gaseous. Keep your batteries always at efficiency of 100, if you know what that is. Don't let anything get cold, or you will check the sequence of everything. But keep your fuel down to just the necessary point so as to get the right amounts of heat in the right places, with nothing going off by radiation or through the chimneys. (All this is SO EASY!)

Then, as a final little addition to all your duties, modify your daily procedure according to the variations in your raw material. That last is a mere trifle, of course. It doesn't add more than 40 of 50 per cent to the "regular" schedule.

The "Endless Chain" of Reagents

We shall end our survey with the general statement that nearly every process calculates on the regaining of reagents and very few ever realize that ambition. There are chemical, physical and financial reasons why this is impracticable. In the hypophosphite process, largely practiced in Mexico, and far from unknown in the United States, which is one of the most successful of all wet processes, neither the hypo nor the sodium

sulphide is regained entirely, although the chemistry of the process would indicate the full restoration of the former. The point is always reached at some stage, when a careful estimate will show that a given solution cannot be "regenerated" at a profit. In the case of salt, it is too cheap a chemical to waste money on, and it rarely pays to evaporate a salt solution merely to regain the solid.

Finally, we have not written this sketch to discourage the attempts at metallic extraction by wet methods. Our object has been rather to indicate in a very broad way, that some of the prevailing fallacies have been considered rather in an economic than a scientific sense.

There have been far more failures than successes. Throwing out all of the mere "tricks" and all of the processes which, although honest were based upon false chemistry, i. e., fundamentally false, there remains a number of failures, owing their downfall chiefly to matters of detail, which a more careful study of conditions might have foreseen. We believe that we have indicated the nature of most of these, and we are quite sure that many processes have failed in practice for lack of figuring. It is one thing to put through a chemical equation or a series of them, in the laboratory. It is quite another to carry out these transformations on the commercial scale, and bring time, space, apparatus, supervision and product into "one harmonious whole."

The Advancement of the Mercury Arc Rectifier

By Harry F. Perkins.

The extensive use of the mercury-arc type of rectifier has caused numerous articles to be published in this country and abroad, describing the characteristics and application of this form of rectifier. With the arc contained in a glass tube, the rectifier has been designed to operate series arc lamps at voltages up to 7500 direct current, with current over 6 amp, and for such other low-voltage direct-current purposes as charging storage batteries and operating small direct-current motors, where the current does not exceed 50 amp.

Within the last few years much work has been done toward increasing the capacity of the rectifier, and articles giving in part the results of this work have appeared from time to time in the *Elektrotechnische Zeitschrift*, Nov. 7, 1912; the *London Electrical Review*, Dec. 27, 1912; *Bulletin de la Société Internationale des Elec.*, January, 1913; *A. E. G. Zeitung*, June, 1913; *Electrical World*, and other journals.

The object of this treatise is to show a recent form of iron tube which is used in place of glass in the development of the rectifier for high power, and which has been reduced to an extremely simple and stable state.

In Fig. 1 a rectifier capable of delivering 150 amp at 350 volts direct current is shown, combined with its vacuum pump on a common base. The cut shows practically all there is to the apparatus. The anodes, which are of steel, are insulated by heavy porcelain from the arms of the tube, and the pressure on their gasket surfaces, which prevents leakage and allows of expansion, is kept fairly constant by means of springs on their retaining bolts. A window is provided in a convenient place so that the interior may be viewed during operation.

It has not seemed advisable so far to do without the vacuum pump, for though the rectifier shows no great amount of leakage over long periods of time, the pump is conveniently available in case of any repairs, while it does not need to run during operation of rectifier. Only one pump would be required to a station, as it is only used occasionally and could handle several tubes at a time in an emergency. This pump requires about 1/3 hp to operate.

The rectifier carries the load previously stated without artificial cooling of any kind; it is started in practically the same manner as all low-potential tubes, as will be described later.

The particular tube shown in cut has been in operation for

several weeks in a public garage, charging the batteries of a number of electric vehicles at a time. The charging takes place mostly at night, though occasionally some charging is

done at noon, and over-charging is done once a week. A tube similar to this one has been in operation for the past three months. The weight of the tube is about 350 lb.

In Fig. 2 a night view of the garage gives an idea of the rectifier load, the light spot on the tube shown in the background indicating that the tube is working. Some of the vehicles in the picture have lead batteries and some are equipped with Edison batteries.

The wiring connections are shown diagrammatically in Fig. 3, the rectifier being of the three-phase, three-arm type. A bank of three delta connected standard transformers step the line potential of 2200 volts, 60

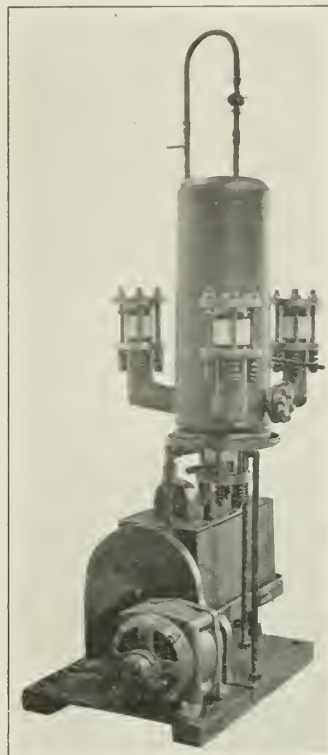


FIG. 1.—BATTERY CHARGING RECTIFIER FOR 150 AMPERES

cycles, down to 236 volts, which is applied to compensators connected in Y to give a neutral point, which is the negative side of the direct-current circuit.

The compensators also allow of voltage variation by means of taps. Close voltage regulation may be obtained by varying the tap connection one phase at a time. The slight un-

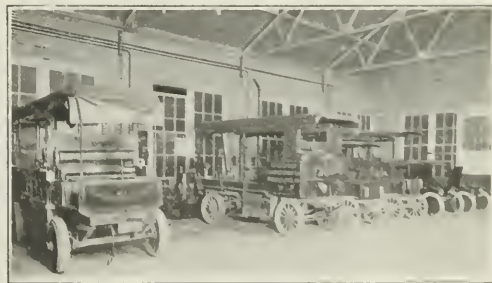


FIG. 2.—RECTIFIER CHARGING BATTERIES ON SIX VEHICLES

balancing of phases thus produced does not affect the rectifier in the least. Each battery has a standard charging rheostat in series which gives the desired individual control. This gives to the rectifier all the regulation qualities of a direct-

current generator, without the rheostat losses. Very fine regulation can be obtained by using a standard induction regulator.

The arc rectifier is an apparatus having characteristics which are entirely different from either a generator, with its moving parts, or an alternating-current static transformer. There is no friction, windage, nor appreciable iron loss, nor I^2R losses of the usual kind in the rectifier; the loss is represented by a practically constant voltage drop, most of which is at the anode and cathode.

While it has often been stated that the voltage drop is constant, it is not found to be accurately so, but on the contrary, it is seldom steady. The variation is negligible, however, except in cases of very low voltage or small currents.

It is found that an increase of pressure in a tube will cause an increase in the voltage drop, and the general design of the

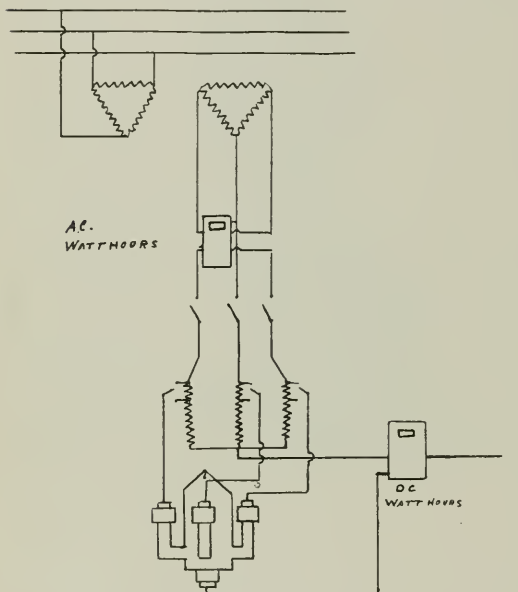


FIG. 3.—CONNECTIONS FOR RECTIFIER IN FIG. 2

tube has a considerable effect on it. The arc drop reaches 30 volts in some cases, while in moderate low-voltage, small-current apparatus, it is approximately 15 volts. Therefore the efficiency of a tube on 110 volts direct current ought to be about 85 per cent, with a decided improvement at higher voltages. That is actually the case.

A comparison between the efficiency of a motor-generator set and a low-voltage rectifier is shown in the curves of Fig. 4 (the slight gradual drop in rectifier curve is due mainly to drop in compensator). The advantages of the rectifier over the motor-generator are at once apparent, as the rectifier maintains its efficiency at all loads, and it is free from loss when no current is being taken. With rectifiers for higher voltages an efficiency of 97 per cent can be obtained.

The power factor created by the rectifier on the power circuit depends largely on the transformer design. The three-arm type of tube has a tendency to produce a unidirectional magnetic flux, which in turn distorts the current wave, and thus the power factor of the circuit is affected. By the proper arrangement of transformer a good power factor can be obtained and in the case of both single and polyphase rectifiers a power factor well above 90 per cent has been obtained.

This leads us to say something about the wave shape of current and voltage connected with rectifiers. Figs. 5 and 6

illustrate the pulsations delivered by the rectifier in Fig. 1. If the supply frequency is multiplied by the number of arms,

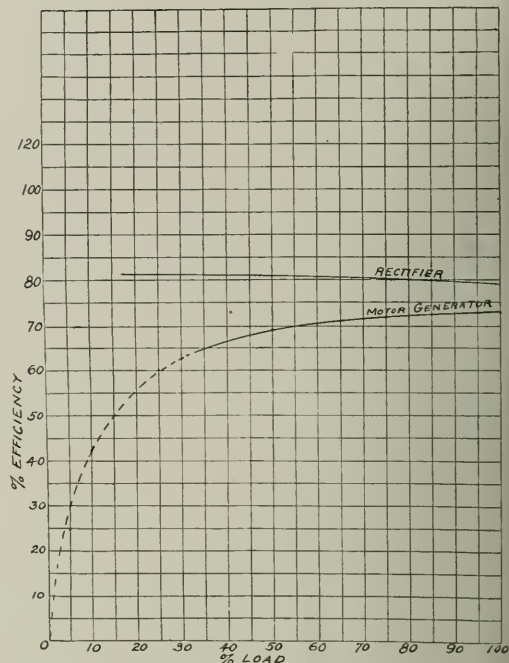


FIG. 4.—EFFICIENCY CURVE OF RECTIFIER IN FIG. 1 COMPARED WITH THAT OF MOTOR GENERATOR SET

the pulsations per second will be the result. Thus if six arms were supplied from a three-phase source, the hollows between the waves here shown would be filled each by a similar wave, and a current practically equivalent to that from a commutator can be obtained.

Figs. 7^a, 8 and 9 are fair samples of the rectified current from single phase with and without direct-current reactance, and of quarter-phase and six-phase rectifiers without addi-

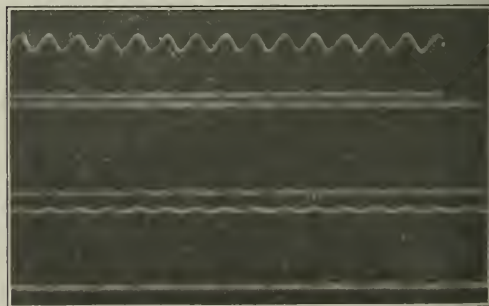


FIG. 5.—UPPER CURVE RECTIFIER VOLTS. MIDDLE CURVE BATTERY VOLTS. LOWER CURVE D. C. AMPERES

tional reactance. Fig. 7^b shows the wave shape when running in multiple with a rotary converter.

While the starting and operating of the mercury arc rectifier is fairly well known, on account of its extensive use in small units, a detailed explanation may not be out of place here.

Taking the case of a single-phase tube (Fig. 10), it is noted

that there are two main anodes, a cathode and a starting anode. The main anodes are connected one to each end of the transformer winding, the cathode is connected to the positive (+)

anode opposite to the one connected to the starting anode, when the potential is reversed.

It is true that the arc will not start without ionization and

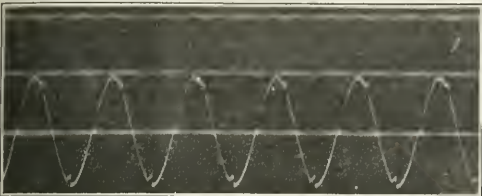


FIG. 6.—UPPER CURVE BATTERY VOLTS, LOWER CURVE LINE VOLTS ON ONE PHASE

side of the direct-current circuit to be supplied. The negative (—) side of the direct-current circuit is connected to the middle point of transformer. The starting anode is connected through resistance to one of the main anodes.

When it is desired to start the tube, the starting anode is

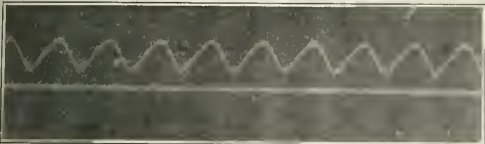


FIG. 7A.—SINGLE PHASE WITH REACTANCE IN CATHODE LINE

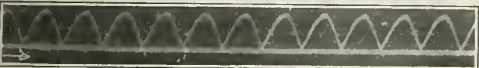


FIG. 7A.—SINGLE PHASE WITHOUT REACTANCE

brought into contact with the cathode. This allows an alternating current to flow, limited by the resistance mentioned above and the auxiliary load. When the contact is broken an arc will start and its direction will be determined by the potential direction at the instant of break. If, however, the starting anode happens to become cathode at the instant of break, the arc dies out when the potential reaches zero, and therefore lasts only a portion of half a cycle.

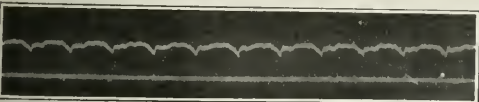


FIG. 7B.—SINGLE PHASE IN PARALLEL WITH ROTARY CONVERTER, CURVE OF D. C. VOLTS

Should the contact break when the potential is in the proper direction an arc is started with a cathode spot in the pool of mercury forming the cathode. This cathode spot, during its brief existence of less than half a cycle, is usually sufficient to ionize enough mercury vapor to start an arc from the main

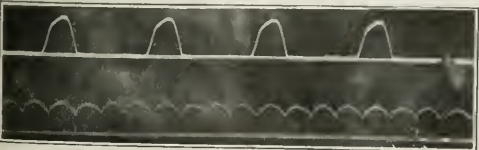


FIG. 8.—QUARTER PHASE, UPPER CURVE, ANODE AMPERE ONE ANODE LOWER CURVE, D. C. AMPERES



FIG. 9.—SIX PHASE VOLTAGE CURVE

that it will go out the instant that this ionization ceases. Therefore, as the arc will supply its own ionization while the voltage remains in the proper direction, it is important to have some means of continuing this ionization while the potential is falling below that necessary to maintain the arc, through

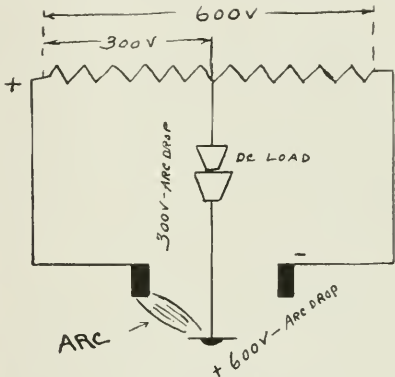


FIG. 10.—DIAGRAM OF CONNECTIONS

zero, to a potential that will start the arc from the other anode.

In the case of rectifiers supplying variable loads where the current may be off for short periods, as, for instance, a railway load, this carrying over of the arc is accomplished by

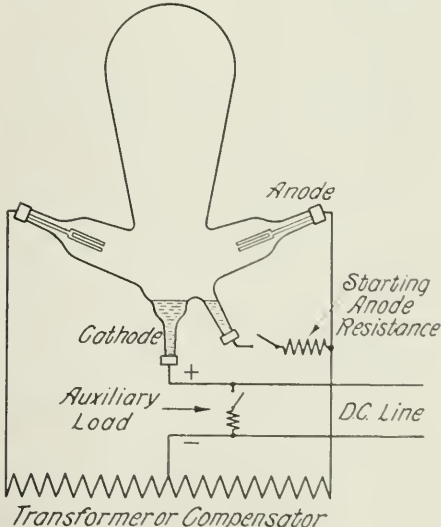


FIG. 11.—DIAGRAM OF CONNECTIONS

supplying the starting anode from a separate direct-current source, or by inserting two small auxiliary anodes which can form part of a separate rectifier circuit supplied by an addi-

tional coil from the main transformer, or by a separate small transformer. This auxiliary rectifier can be designed to consume a minimum current at low voltage and can thus be made self-sustaining with small loss.

In rectifiers carrying a steady load this carrying over can be done to best advantage by placing in the direct-current circuit an air-gap reactance.

The main arc flows in one direction only when the cathode is excited and when the potential at the anode is positive with respect to the cathode, but no current will flow without excitation until a potential is reached that will cause a Geissler discharge, for the vacuum in these tubes is of a very high degree and ordinarily it would take several thousand volts to cause a discharge without a cathode spot. This is one reason why the current does not flow in both directions.

Upon examination of the diagram, Fig. 11, it will be seen that current flows in only one-half of the transformer winding at a time, and the potential of the direct-current load is that of one-half of the average (not the r.m.s.) transformer voltage, minus the arc drop; as the cathode is only separated from the working anode by the arc drop, we have practically all the potential of the transformer applied between the cathode and the idle anode, which is now negative with respect to the cathode.

That the rectification is perfect can be understood when it is shown that many hundreds of tubes are in service with over 10,000 volts between anodes.

Research Laboratory, General Electric Company, Schenectady, N. Y.

Quantitative Spectrum Analysis

By G. A. Shook

I. Colorimeters

There is to-day unfortunately no satisfactory explanation of the ultimate nature of light. In all probability, however, light is some sort of a transverse wave motion, but the character of the medium through which the waves are propagated and the mechanism by which the waves are produced are questions still unanswered. Nevertheless the experimentally derived laws in this field are so well established that the lack of a fundamental theory is not felt practically.

When white light passes through blue glass it becomes blue. Whether the white light contains, as one of its components, the blue color or whether the blue glass manufactures the blue light out of the white, we are not able to decide with so much confidence as the ancients or the philosophers of Newton's time. Yet the fact remains and there is never any exception to it.

We are still farther from satisfactory theories when we come to the case of colored solutions, due to the numerous exceptions to all the simple hypotheses that have been proposed. These hypotheses are probably correct, but due to such disturbing influences as incomplete ionization and chemical reaction they cannot be universally verified. Nevertheless as far as practical colorimetry is concerned, experimental errors are generally greater than errors that might arise due to the departure from fundamental colorimetric laws.

When a colored compound is dissolved in a colorless solvent the extent to which the solution is colored depends upon the concentration, or the mass of the solute per unit volume of the solvent, and there is a very simple relation between the two factors. A thickness of 10 cm. of a 20 per cent solution will produce the same light absorption as 20 cm. of a 10 per cent solution of the same compound, etc.

If the molecules of the dissolved substance are not dissociated and if furthermore the solvent exerts no influence upon the color, then this simple relation holds rigorously. However, if the dissolved compound is ionized or dissociated then there are several possible results.

1. If either the positive ion or the negative ion alone is colored and if it has the same color as the molecule then the

law still holds even with partial ionization. The negative ions Cl , NO_3 , SO_4 , etc., are colorless and the copper salts formed with these ions are all blue. The positive ions K , Na , Br , etc., are colorless and the chromates formed with these metals are all yellow.

2. If the molecule and the colored ion do not have the same color then the rule would not hold. A very concentrated solution of copper chloride is green, but on dilution the color changes to blue and remains blue as long as the dilution continues. Consequently the law would hold if we considered only dilute solutions.

3. In the case of two colored ions, neither ion being colored the same as the molecule, for example H_2 , Fe , Cu , the law would not hold except for dilute solutions, i.e., complete ionization.

The color of the ion may be different for different electrical charges; for example, the iron ion in the ferrous condition is green, while in the ferric condition it is yellow.

It might seem at first sight that any method for determining the concentration of a salt by means of its color would be unreliable, but in practice all these difficulties are not met with at once. In most cases we are dealing with dilute solutions or at least with a limited range of concentrations. It will be subsequently shown that in refined work it is not difficult to determine just how far the law holds for any particular case.

Law of Light Absorption

Whenever light falls on any absorbing medium, for instance, a piece of colored glass or a cell containing a colored solution, part is reflected, part absorbed and the remainder transmitted.

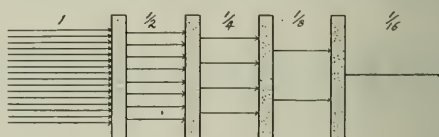


FIG. 1.—GRAPHICAL REPRESENTATION OF THE LAW OF ABSORPTION OF LIGHT

Let the intensity of the light incident upon a layer of unit thickness of an absorbing medium be I and the intensity of the transmitted light be I_1 . If the coefficient of transmission is a , we may write

$$\frac{I_1}{I} = a \text{ or } I_1 = I a$$

a is that fraction of the incident light which is transmitted by unit thickness.

Now if this transmitted light traverses a second layer of unit thickness it will again be diminished in the same ratio, hence

$$I_2 = (I_1 a) a = I a^2$$

For three layers

$$I_3 = (I a^2) a = I a^3$$

Therefore, if the light traverses a thickness of h units the transmitted light, I_h , has the value

$$I_h = I a^h \quad (1)$$

For example, if the coefficient of transmission a is $\frac{1}{2}$, then the intensity of the light transmitted through one layer of unit thickness is $\frac{1}{2}$ the intensity of the incident light. If this transmitted light traverses a second layer, $\frac{1}{2}$ is again transmitted so that we have left only $\frac{1}{4}$ of the original amount of light. If the light passes through three layers we have $\frac{1}{8}$ left and for four layers only $\frac{1}{16}$ or $(\frac{1}{2})^4$ remains, etc. (Fig. 1.)

For a second absorbing medium

$$I_h' = I a'^h$$

and if the transmitted light is the same in each case $I_h' = I_h$, whence

$$a^h = a'^h \quad (2)$$

Now let b represent the transmission coefficient for a unit layer of a colored solution of unit concentration (1 g. of

solute dissolved in 1 cc. of solvent). We may now write

$$I_1 = I b,$$

and if the concentration is doubled, i.e., if the number of absorbing molecules is doubled, then

$$I_2 = (Ib) b = I b^2.$$

Therefore, if the light passes through a unit layer of concentration c it follows that

$$I_c = I b^c \text{ or } \frac{I_c}{I} = b^c \quad (3)$$

For a different concentration c' of the same dissolved substance we may write

$$I_{c'} = I b^{c'}$$

Since we have unit thickness in each case, $h = h'$, whence

$$I_c = I b^c = I a$$

and

$$\begin{aligned} I_{c'} &= I b^{c'} = I a' \\ b^c &= a \text{ and } b^{c'} = a' \end{aligned} \quad (4)$$

From (1) it follows that

$$\frac{I_h}{I} = a^h = (b^c)^h = b^{ch}$$

This is known as Beer's Law.

Combining (2) and (4) we obtain the relation

$$b^{ch} = b^{c'h'}$$

whence

$$c h = c' h' \quad (5)$$

Equation (5) forms the basis of the theory of all colorimeters. Light of the same intensity is caused to pass through two suitable vessels; one contains a solution of known concentration c' and the other a solution of unknown concentration c of the same solute. The heights h and h' of the two columns are varied until the transmitted light through each is of the same intensity and then the heights are measured. The unknown concentration may then be determined by means of the simple relation

$$c = \frac{c' h'}{h}$$

There is another class of instruments which is used for determining concentrations, known as spectrophotometers. A spectrophotometer simply compares the intensity of two light sources for some particular color. From (3) it is seen that if the ratio of the transmitted light to the incident light is known, the concentration may be estimated. Equation (3) may be put in the form

$$\log \frac{I_c}{I} = c \log b = k c \quad (6)$$

where k is a constant for any particular wavelength of light and any particular substance in solution.

Equation (6) simply means that the logarithm of that fraction of light transmitted by a unit layer of a colored solution is directly proportional to the concentration of the solution. This unit thickness may be 1 cm. or any other convenient width of containing cell.

For simplicity we will call the negative logarithm of the ratio $\frac{I_c}{I}$ the *extinction coefficient*, so that

$$e = -\log \frac{I_c}{I}.$$

The logarithm of a fraction is, of course, negative, hence, by prefixing the negative sign, e becomes a positive quantity. Equation (6) may now be written

$$e = A c$$

We will moreover call the constant A the *absorption ratio*.

The form of the expression for the determination of e depends, of course, upon the type of instrument used, and this will be fully considered in a succeeding issue of this journal.

The extinction coefficient is sometimes defined as the reciprocal of the value of the thickness of a layer such that the incident light is diminished to one-tenth its value. This, however, is an unnecessary complication since all that is required is that we may be able to express the ratio of the intensity of the transmitted light to the intensity of the incident light.

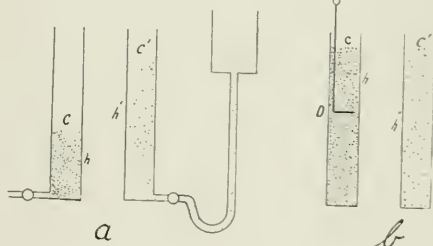
There are five essential parts to a good colorimeter, viz.:

1. Containing vessels for the standard and unknown solutions.
2. A method for adjusting the height of one or both of the two liquid columns.
3. An accurate means for measuring the heights of the two columns.
4. A device for bringing the two fields of view close together in order to determine when equality of brightness is obtained.
5. A source of illumination.

The simplest type of colorimeter, but one which is too crude for accurate work, consists merely of two cylindrical glass jars or graduates placed upon a white mat surface, to give uniform illumination. One graduate is filled to a convenient mark with the unknown solution and a standard solution is poured into the other graduate until the two top surfaces of the liquids appear equally bright. Knowing the two heights the unknown concentration may be readily calculated.

Of course, the standard solution may be filled to a given mark in one cylinder and the unknown poured into the other cylinder until a balance is effected.

Sometimes one jar is filled to a given mark (100 cc.) with an unknown solution and the other jar is filled with a stronger solution of known concentration to a given mark (10 cc.). The known solution is then diluted until both jars appear to have the same intensity of color and the required amount of



FIGS. 2A AND 2B.—SIMPLE TYPES OF COLORIMETERS

dilution necessary to effect this balance can be estimated by observing the volume of the standard solution.

In such cases the diameter of the two cylinders must be equal, but when we are concerned only with the heights of the liquid columns this is not necessary.

A slight improvement upon this simple type of colorimeter is to provide the jars with stop cocks for drawing off the solutions. If one of the jars, say the one containing the known solution, is connected to a reservoir, Fig. 20, which can be gradually raised and lowered, it will greatly facilitate the operation of making an intensity balance.

A still better arrangement is a tubular reservoir provided with a glass plunger, in which case the solutions come in contact with glass only.

Instead of changing the height of the liquid a white disc D , Fig. 2b, may be lowered into one jar, thus changing the effective length of the liquid column.

Another scheme which is often used is to fill a number of jars, all to the same mark, with standard solutions of varying concentrations. A similar jar is then filled to the same mark with the unknown solution and its concentration may be roughly estimated by comparing the intensity of its color with the standard solutions.

The chief objection to all of these methods is that only a rough estimation of the equality of brightness can be made due to the fact that the two fields of view are not close enough together. This objection is overcome in the following instruments:

Duboscq Colorimeter

This instrument is shown in schematic form in Fig. 3. The standard and the unknown solutions are placed in two glass

jars, A and A' , into which dip two solid glass cylinders, B and B' . Each cylinder is actuated independently by means of a rack and pinion, and to each pinion is attached a fiducial mark F which moves over a graduated scale. The heights h and h' may be read directly from the scales.

Daylight is diffusely reflected from an opalescent mirror M to the jars A and A' , is then transmitted through the cylinders B and B' and by means of the reflecting prisms P and P' finally reaches the eye.

By means of these two prisms, which are cemented together, the two fields of view are brought in close contact so that the observer can make an accurate intensity balance. The faces of the prisms, the ends of the cylinders and the bottoms of the jars are all optically plane, so that the field of view is very clear and uniform, which is essential, if accurate work is required.

The eye is not able to make any estimate whatever of the relative brightness of two illuminated fields, but it can accurately determine when two fields are equally bright if favorable conditions obtain.

The two fields must be uniformly illuminated throughout and must not be separated by a line of any appreciable width. When the eye is required to move from one field to the other, as in the case of the crude colorimeters, it loses the impression of one field before reaching the other and the accuracy of the judgment is greatly impaired.

In some forms of colorimeters both eyes are used, but such a method is unreliable.

It is moreover desirable to have just sufficient light enter the eye to prevent fatigue. If the field is too dark the eye is constantly under a strain due to an abnormal dilatation of the iris and if it is too bright an abnormal contraction is necessary.

To make an accurate intensity balance, means must be provided for varying the intensity of one or both of the fields in a continuous manner. One field should be made alternately brighter and darker than the other, diminishing each time the difference in intensity until no difference can be detected. Care must always be taken in making a setting, but the operation of making one field brighter than the other must not be prolonged until the field appears blurred, for then the eye has lost its maximum sensitiveness.

A colorimeter similar to the Duboscq, due to Stammer, is sometimes used for beer analysis. The standard solution is here replaced by a colored glass and the containing vessel is moved instead of the dipping cylinder. This instrument, however, is not applicable for all adulterations.

Müller has designed a similar apparatus for water analysis. It has a 10 cm. tube which is filled with water, to which Nessler's reagent has been added. The balance is made by means of gold-colored glasses 1.5 mm. in thickness. The equivalent effect of the glasses is given in the following table:

1 glass is equivalent to.....	.02	mg. of NH_3 100 cc. of H_2O .
2 glasses are equivalent to.....	.05	
3 glasses are equivalent to.....	.07	
4 glasses are equivalent to.....	.10	
5 glasses are equivalent to.....	.15	
6 glasses are equivalent to.....	.20	

Wolff Colorimeter

The essential difference between the Wolff colorimeter and the Duboscq is that the former is not provided with the dipping cylinders. In order to bring the two fields to the same intensity, liquid is drawn off from either vessel.

It is somewhat easier to calculate the concentration if one jar is filled with the weaker solution up to the 100 mark and the stronger solution drawn off. This arrangement has the disadvantage that one cannot make a fine adjustment of the two fields as in the Duboscq. In order to repeat a setting it is necessary to pour some more solution back into the vessel or to disturb the solution which is at the 100 mark.

Since the accuracy of the concentration, finally determined, depends primarily upon the accuracy of making a good photometric balance, this is a weak point in this type of apparatus. It does, however, have this slight advantage, that the jars do

not have to be made as long as the ones used in the Duboscq instrument since there is no liquid to overflow.

In some types of the Duboscq colorimeter, this difficulty is overcome by making the upper part of the jars of much larger diameter than the lower part.

In the Duboscq instrument, the light entering the two vessels does not come from the same part of the mirror and, therefore, it is necessary to bring the two fields to the same intensity before the solutions are poured into the vessels. In the Wolff instrument, however, the light which reaches the two vessels is reflected, by means of a Fresnel double prism, from approximately the same region so that the two fields are practically always the same in intensity.

A smoked glass is sometimes used to diminish the intensity of the field, thus enabling the observer to make a better balance. The bottom caps of the two glass tubes may also be removed for cleaning.

The above types of colorimeters involve the essential features, although there are a number of other colorimeters which

FIG. 3.—DUBOSCQ COLORIMETER

employ different methods for changing the intensity and different devices for producing good fields.

The most sensitive photometric screen is undoubtedly the Lummer-Brodhun. The optical system for bringing the two fields together, when this photometric screen is employed, is shown in Fig. 4.

L is the Lummer-Brodhun prism and C is a compensating cube, which is inserted in order that the path of each beam may have the same optical length. This arrangement could of course be used on any colorimeter employing two vertical vessels.

Extinction Colorimeters

There is another class of colorimeters sometimes employed for turbid solutions. Instead of comparing the intensity of the light transmitted through a standard solution with the

light transmitted through an unknown solution the light through a single column is made extinct by varying the length of the column.

A turbid solution of known concentration is placed in a suitable vessel and the observer views a standard candle or a lamp of constant intensity through this column of absorbing material. The length of the column is increased by pouring in more of the turbid solution until the light just becomes extinct and then the length of the column is noted. If the same

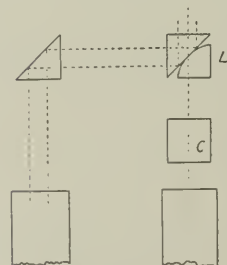


FIG. 4.—LUMMER-BRODHUN COLORIMETER

operation is carried out with an unknown solution its concentration may be easily calculated.

Writing Beer's Law for the two cases we obtain

$$\log \frac{I_h}{I} = b^c h$$

and

$$\log \frac{I_{h'}}{I} = b^c h'$$

But the fraction of the incident light which reaches the eye in each case is approximately the same, hence

$$\log \frac{I_h}{I} = \log \frac{I_h'}{I'} \quad (7)$$

$$c h = c' h'$$

Absolute darkness is not, of course, produced in each case when the light disappears, consequently I_h is a finite quantity, though very small.

This method has the advantage that no standard solution is required and the necessary apparatus is much simpler than that which is necessary for the comparison of the intensities of two fields.

Colorimeters based upon this principle cannot, however, be used for accurate work since one is not able to reproduce the extinction point with the same accuracy that one can reproduce the equal-intensity point.

Moreover, the personal equation must be reckoned with, since different eyes vary greatly in this respect.

Again, the standard lamp must be kept constant, which is not always possible without complicated auxiliary apparatus.

This method has been used extensively for the rough estimation of the amount of sulphate in water, urine, coal, cement, etc., and in fact in any case where a fine precipitate could be easily formed.

The early investigators using this simple method observed that the product of the concentration and the height of column necessary to produce extinction is a constant, but aside from this they gave no theory of the method. Some of them state that it is necessary to keep the light constant and others state that this precaution is unnecessary. The column of liquid must always be lengthened until the lamp just disappears, but manifestly a strong light will require a greater length of absorbing material than a weak light.

Evidently it is not necessary to restrict this class of analysis to the extinction colorimeter since any of the determinations may be made by means of a spectrophotometer with much greater accuracy. The colorimeter is not so well adapted to this sort of work as the spectrophotometer, due to the fact that the former requires a much longer layer of absorbing medium and the consequent loss of light is too great.

There are a number of colorimeters on the market which are designed for special purposes, but the fundamental principle involved in most of them is the same and consequently one instrument, with slight modifications, can be used for most all purposes.

There is no one colorimeter which embodies all of the best features. The best method for bringing the two fields together is that shown in Fig. 4, while the best method for varying the intensity and determining the heights is that used on the Duboscq colorimeter.

The containing vessels should be easily removable for refilling and cleaning.

Since we are concerned only with the thickness of the absorbing layer and not with the volume of the solution, it is not necessary that the two vessels have the same diameter nor that either has a uniform cross-section.

Metallic scales and delicate optical parts should be well protected from the action of salts.

Before taking any observations upon a solution one should make sure that both fields of view are equally illuminated. They can be brought to equality by shifting the colorimeter slightly or by rotating the illuminating mirror.

In the next article of this series spectrophotometers will be considered.

University of Illinois,
Urbana, Illinois.

Plumbago is the chief mineral product of Ceylon, and the United States is the principal consumer of the exported mineral. The mined product is sorted by hand and classified into "lump," "chip," "dust," and "flying dust." These sorts are then graded according to quality.

A Metallurgical Problem

Concentration and Reverberatory Smelting of a Second-Class Copper Ore

By F. W. Traphagen

One of the most interesting of all metallurgical calculations is that involved in the treatment, by reverberatory smelting, of a second-class copper ore.

With proper attention to details and a careful consideration of what occurs in good practice, one can calculate with great precision what will take place in the handling of such an ore from mine to marketing of finished products; and can make also a very close estimate of costs and losses, hence of the commercial promise of each particular deposit.

These estimates involve a careful study of local conditions and a close comparison with practice in various places, but when made with care one can justly have great confidence in the results of his calculations. Of course a critical analysis of local conditions and a study of the results of experimental work, especially of concentration, are very important considerations.

In the particular problem taken for illustration of the methods for preliminary calculations, an attempt is made to secure fairly close approximations to average conditions. The data used will fit Montana practice more nearly than that of any other section, but an attempt is made to avoid, as far as practicable, the localization of data.

Just where the line of demarcation between first and second-class ore shall be placed depends upon three principal factors: the market price of copper; the respective losses in concentration with roasting and reverberatory smelting, as contrasted with that in direct blast-furnace smelting; and the costs of the respective treatments.

With a high market price and with large concentration losses, the percentage of copper marking the line of demarcation is lower.

Assumed Metallurgical Conditions

In our calculations we will assume the following:

Second-class ore for concentration, roasting, reverberatory smelting and converting treatment will contain: Copper, 4 per cent; iron, 11 per cent; sulphur, 14 per cent; silica, 58 per cent; silver, 2 ounces per ton; gold, 0.015 ounce per ton, and alumina and other non-metallic gangue material, balance, approximately 13 per cent.

Reverberatory slag: Silica, 40 per cent; ferrous oxide, 45 per cent, and matte, 1 per cent, the precious metals being assumed to be present in the slag in the proportions in which they exist in the matte carried by it.

Converter slag: Silica, 29 per cent; ferrous oxide, 58 per cent; copper, 2 per cent; silver, 1.2 oz. per ton; gold, 0.005 ounces per ton. Inasmuch as it is the universal custom to retreat converter slag, either in blast furnaces or in reverberatory furnaces, with a high recovery; and also as this slag is a good basic flux for the blast furnace charge, we will in our calculation assume a final loss in converter slag of only 20 per cent of the values present, making no charge for recovery of metallic constituents.

Concentration losses will be assumed in this problem to be 15 per cent of all metallic constituents, including the associated sulphur. The alumina and undetermined gangue will be assumed to follow the silica in ratio to the amounts of each per cent in the original ore.

Second-class ore of the same composition as that given above will be used for "lining ore," i.e., for use in the converter to supply the necessary silica for the oxide of iron formed from the matte during converting. No charge will be made for the treatment of this ore other than for its mining and freight to the smelter.

It is assumed that a 40 per cent matte is required and that the blister copper contains 99.2 per cent copper.

In order to avoid unduly complicating the problem, volatilization and dust losses are not to be taken into account.

Assumed Costs and Prices

The following costs of treatment will be assumed:

Mining, including development, administration and general depreciation, per ton ore.....	\$4.00
Freight from mine to smelter, per ton ore.....	.20
Concentration, per ton ore.....	.60
Roasting, per ton concentrate.....	.50
Smelting, per ton calcine.....	3.00
Converting, per ton matte.....	4.00
Freight, refining and marketing, per ton blister copper..	26.00

The market value will be taken as 15 cents a pound for copper, \$20.67 an ounce for gold and 60 cents an ounce for silver.

Required Data

The following information is required:

1. Composition and weight of tailing produced, *i.e.*, copper, per cent; gold and silver, ounces each per ton.
2. Composition and weight of concentrate, especially percentage of silica to be left with the concentrate, for the direction of the foreman in his work, and concentration ratio.
3. Composition and weight of calcine, and particularly percentage of sulphur to be left in calcine for production of matte of required copper tenor.
4. Composition and weight of reverberatory slag, and losses in pounds of copper and ounces of gold and silver.
5. Composition and weight of matte available for concentration, *i.e.*, weight of total matte from calcine minus that carried by reverberatory slag.
6. Weight of "lining ore" required to convert matte. This involves a calculation of the available silica, *i.e.*, the silica in excess of that required by the iron of the "lining ore" which goes into the converter slag in the same ratio as the iron from the matte. The weight of copper, gold and silver contributed by this ore to the total must also be determined and added.
7. Composition and weight of converter slag to determine losses of copper, gold and silver, 20 per cent of which is to be deducted.
8. Composition and weight of blister copper produced, and of the three metals, *i.e.*, per cent copper and ounces per ton gold and silver in blister, and total pounds copper and ounces gold and silver in this final product of the smeltery.
9. Costs of treatment and returns from the marketing of products.
10. Costs for production of a pound of copper, with and without elimination of gold and silver values.

One of the simplest methods of calculation is to base all weights and all costs on the original ton of ore.

The results of successive calculations will be tabulated, *assumed values being given in italics.*

Solution of the Problem

The first step consists in finding the weights of the metallic constituents of the ore left with the concentrate and tailing respectively, on the basis of the concentration loss. In this case 85 per cent will be in the concentrate, the remainder in the tailing. This is calculated in pounds.

In order to determine the amount of silica in concentrate and also in tailing, the quantity of iron available for fluxing purposes must be determined. This quantity varies with the grade of matte required. The higher the grade the less iron will be present in the matte; hence the more iron for fluxing purposes and the greater quantity of silica to be left in the concentrate.

In our example we have taken a 40 per cent matte. The copper is present as cuprous sulphide, Cu_2S , and the remainder is calculated as ferrous sulphide, FeS , neglecting the small amount of gold, silver and other elements which would make a very small percentage of the total. This assumption is not strictly true, for there is frequently an excess of iron over the assumed FeS ; but it is sufficiently close for ordinary work.

In our calculations we will use, instead of the exact atomic weights, the nearest round numbers. Taking copper at 64 and sulphur at 32, Cu_2S would be 80 per cent copper and 20 per cent sulphur; and the 68 pounds copper recovered in our concentrate would require 17 pounds sulphur to form Cu_2S for total matte.

This 85 pounds of cuprous sulphide constitutes 50 per cent of a 40 per cent matte, *i.e.*, 40 per cent copper plus 10 per cent sulphur, to make Cu_2S , leaving a balance of 50 per cent for the ferrous sulphide, which in this case is equal to 85 pounds.

Ferrous sulphide, FeS , contains 56 parts of iron and 32 parts of sulphur in 88 parts of the compound, or in simpler terms 7/11 iron and 4/11 sulphur. This is equal to 54 pounds of iron and 31 pounds of sulphur.

Fifty-four pounds of iron being required for matte, the difference between this figure and the weight of iron in the concentrate gives us the amount available for fluxing silica:

$$187 - 54 = 133 \text{ lb. iron for flux.}$$

The reverberatory slag selected consists of 45 per cent ferrous oxide and 40 per cent silica. The 133 pounds of iron will form 171 lb. FeO , thus:

$$56 (\text{Fe}) : 72 (\text{FeO}) :: 133 : X, \text{ or } 171 \text{ lb. FeO.}$$

The amount of silica required by this iron is

$$171 (\text{FeO}) : X (\text{SiO}_2) :: 45 : 40, \text{ or } 152 \text{ lb. SiO}_2.$$

This is the amount to be fluxed, and hence is the weight of silica in concentrate, calcine and reverberatory slag, the same weight being present in each.

The weight of alumina, etc., in concentrate would be:

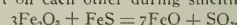
$$\frac{1160}{\text{SiO}_2 \text{ in ore}} : \frac{152}{\text{SiO}_2 \text{ in conc.}} :: \frac{26}{\text{Al}_2\text{O}_3 \text{ in ore}} : \frac{X}{\text{Al}_2\text{O}_3 \text{ in conc.}}$$

or 34 lb. alumina, etc., in concentrate, which is also the weight in the calcine and reverberatory slag.

Professor Peters' takes the view, justly I think, that iron in

¹Principles of Copper Smelting, p. 31.

calcine is present as ferric oxide and ferrous sulphide, but not as ferrous oxide. Making this assumption, these two iron compounds react on each other during smelting thus:



In modern reverberatory smelting the atmosphere of the furnace, so far as its influence on the charge is concerned, is neutral, for the charge spreads out at once on the surface of the matte left on the hearth and is covered or surrounded by the slag left from the previous skimming and is not affected by the gases above it.

There may be, of course, a small amount of magnetic oxide present, but the amount is generally so small that it may be safely ignored in our calculations. The ferrous oxide of the slag then is derived from the calcine in which it exists, not as ferrous oxide but 6/7 in the form of ferric oxide and 1/7 as ferrous sulphide. This, with our figures for other constituents, gives us the weight of calcine.

The 152 pounds of silica constitutes 40 per cent of our slag, the total weight of which is

$$40 : 100 :: 152 : x, \text{ or } 380 \text{ lb. slag.}$$

Assuming the matte lost in slag to be 1 per cent, we have in slag 3.8 pounds of matte which we distribute among its different constituents. Subtracting these results from the weights in total matte gives us weights going to the converter.

Our lining ore contains considerable iron which goes into slag in the ratio established, just as the iron of the matte does, and uses up a proportionate amount of silica. To determine the available silica of this lining ore we must first deduct the silica required by its own iron, which is 11 per cent, equivalent to 14.14 per cent ferrous oxide. According to assumptions made for the converter slag this will be:

$$\frac{58}{\% \text{ FeO}} : \frac{28}{\% \text{ SiO}_2} :: 14.14 : x, \text{ or } 7.07 \% \text{ silica.}$$

Deducting this amount from the 58 per cent silica in the ore, we have 50.93 per cent available silica.

We have 52.79 pounds of iron in our available matte which must be oxidized and fluxed. This is equivalent to 67.87 pounds ferrous oxide, and will require 33.935 pounds of silica,

TABULATION OF RESULTS
(ASSUME VALUES IN ITALICS)

Material	W'ght, Lb.	(ASSUMED VALUES IN TABLE)																			
		COPPER				IRON	Ferrous Sul- oxide, Lb.	FERROUS OXIDE		Ferric Oxide, Lb.	SILICA		SULPHUR		ALUMINA		SILVER		GOLD		
		%	Lb.	%	Lb.	%		Lb.	%		Lb.	%	Lb.	%	Lb.	Oz. per Ton	Oz.	Oz. per Ton	Oz.		
Second class ore	2000	4	80	11	220					58	1160	14	280	13	260	2	2	0 915	0 015		
Concentrate.....	679		68		187					22.4	152		238		34	5	1 7	0 037	0 01275		
Tailing.....	1321	0.9	12		33					1008	42		226		34	0 595	0 3	0 00446	0 06225		
Calcare.....	548.8		85			114.8			163	152	107		42				1 7		0 01275		
Matte—total.....	170	40	68		54								48			20	1 7	0 15	0 01275		
Matte—in slag.....	3.8		1.52		1.21								1.07				0 038		0 000285		
Matte—available.....	166.2		66.48		52.79								46.93				1 662		0 012465		
Reverberatory slag.....	380	0.4	1.52			46	171			40	152		34				0 038		0 000285		
Lining ore.....	66.6	4	2.66	11	7.33					58	38.63	14	9.32	13	8 66	2	0 0666	0 015	0 0005		
Converter slag.....	133.2	2	2.66			58	77.28			29	38.63					1 2	0 08	0 0005	0 000333		
Blister copper.....	69.16	99	2 68.61													49.54	1 713	0 37	0 0129		

100
or $\frac{100}{50.93} \times 33.935 = 66.6$ pounds of ore.

BALANCE SHEET

To Mining 1 ton ore @ \$4.....	\$4,000	By Cash from sale of metals:	
Freight 1 ton ore @ \$0.20.....	.2000	68.61 lb. copper @ \$0.15.....	\$10.2915
Mining 66.6 lb. ore (lining).....	.1332	1.713 oz. silver @ \$0.60.....	1.0275
Freight 66.6 lb. ore (lining).....	.0067	0.129 oz. gold @ \$20.67.....	.2666
Concentrating 1 ton ore @ \$0.60.....	.6000		
Roasting 679 lb. concentrate @ \$0.50 per ton.....	.1679		
Smelting 548.8 lb. calcare @ \$3 per ton.....	.9232		
Converting 166.2 lb. matte @ \$4 per ton.....	.3324		
Marketing 69.16 lb. blister copper @ \$26 per ton.....	.8991		
Total cost.....	\$7 2625		
Profit.....	4.3231		
	\$11.5856		\$11.5856

Cost per lb. copper, $\$7.2625 \div 68.61 = 10.58$ cents.

Cost per lb. copper, crediting value of precious metals $(\$7.2625 - \$1.2941) \div 68.61 = 8.7$ cents.

This ore will contribute gold, silver and copper, and these amounts must be added to the total.

Our 170 pounds of total matte contains 1.7 and 0.01275 ounces silver and gold respectively, hence the slag will contain:

$170 : 3.8 :: 1.7 : x = 0.038$ ounces silver, and

$170 : 3.8 :: 0.01275 : y = 0.000285$ ounces gold which must be deducted to give gold and silver going to converter.

From the 66.48 pounds copper in matte going to converter, plus 2.66 pounds copper furnished by "lining ore," is subtracted 20 per cent of the copper carried by the converter slag. The converter slag weighs 133.2 pounds, 2 per cent of which is 2.66 pounds; and 20 per cent of this last figure is 0.53 pounds, leaving 68.61 pounds of copper resulting from the treatment of one ton of ore, plus the required lining ore. In like manner the gold and silver are obtained, the results being given in the table.

One ton of ore produces 679 pounds of concentrate, hence the concentration ratio is $2000 : 679 :: 2.94 : 1$.

The gold and silver are calculated to points beyond the possibility of assay, for the reason that these small fractions become of great consequence when multiplied by several thousand, and have a marked bearing on the sum total of profit.

Summary of Procedure

1. From percentage composition calculate pounds per ton of each constituent. Gold and silver, of course, in ounces per ton.

2. On basis of concentration loss, calculate weights of metals and of sulphur, in concentrate and tailing respectively.

3. From weight of copper in concentrate calculate weight of iron required to form matte.

4. Subtract iron required for matte from total iron in concentrate and calculate weight of silica required to flux this iron in ratio of ferrous oxide and silica in reverberatory slag. This gives weight of silica to be left in concentrate.

5. Calculate weight of alumina in concentrate on basis of ratio existing in ore.

6. The total weight of concentrate is now known. From this weight and the weight of silica present the percentage of silica can be determined.

7. The composition of the tailing can be determined by subtracting the weights of the constituents of the concentrate from those of the ore. The total weight of tailing is obtained in a similar manner, and from these data the percentage of copper and ounces per ton gold and silver can be calculated.

8. The weight of reverberatory slag is obtained from the weight and percentage of silica. From this weight we calculate the weights of copper, iron, gold and silver carried by the matte in the slag, and these weights subtracted from the corresponding weights in total matte give metals going to converter.

9. The weight of lining ore required is determined by first finding its available silica and dividing the weight of silica required to flux the iron in matte going to the converter by the percentage of available silica and multiplying the quotient so obtained by 100.

10. From the weight of lining ore used the weights of copper, gold and silver contributed to the total can be obtained.

11. The weight of converter slag can be calculated from either the iron or silica, the weights and percentages of which are known. From this weight and the determined recovery by retreatment of converter slag, losses of copper, gold and silver are calculated.

12. The above calculations give us the weights of the various products treated and the final recoveries of copper, gold and silver, from which the cost of treatment and value of products are obtained.

13. The weight of blister copper is determined by dividing the weight of copper by its percentage and multiplying by 100.

14. Ounces per ton gold and silver are obtained by dividing ounces present in blister copper by the weight of blister and multiplying quotient by 2000.

Denver, Col.

Successful copper smelting in blast furnaces at great altitudes requires that the capacity of blowers and air conduits, and the area of tuyeres, be increased with the lowering of atmospheric pressure. For an atmospheric pressure of 8.75 lb. per sq. in., these factors would have to be increased 70% in order to burn the same quantity of coke as in a furnace designed for sea-level conditions. Other features of smelting at high altitudes are (1) reducing radiation losses from crucible and jackets; (2) the use of hot blast, and (3) the use of back-pressure on the furnace in such a way that the fuel is burned under the most desirable pressure for its economical working. These facts are deduced from a mathematical consideration of the subject by VINCENTE PAZOS Y SANCIO, in the July *Quarterly* of the Columbia School of Mines.

The Elmore vacuum flotation process at the Sulitelma mines, Norway, produced 850 tons of copper concentrate in the month of July, 1913.

Cyanide Practice in the Black Hills, South Dakota.—III.

By H. C. Parmelee

Lundberg, Dorr & Wilson Mill

On July 31, 1913, the 100-ton cyanide mill of Lundberg, Dorr & Wilson, situated near Terry, S. D., definitely ceased operations after a long, profitable and interesting career. The plant was closed on account of the exhaustion of the ore supply in the firm's mines. Since January, 1904, this mill has been in continuous operation, with the exception of two periods of idleness during the "eight-hour" and "closed-shop" labor strikes in 1907 and 1910. During this period it has been used for the treatment of ore from the firm's Buxton and Bonanza mines, and has been operated also as a custom mill for other mines in the vicinity.

A general view of the property is given in Fig. 1. The Buxton mine is located on the north side of Fantail gulch, below the town of Terry, and the Bonanza is across the gulch on the south side. The mill was located on the Buxton side, and ore from that mine was trammed directly into the ore bins. The Bonanza ore was delivered to the top of the mill by a Bleichert aerial tramway, with buckets carrying 1200 lb. ore. There is only a single span of 910 ft. in this tramway, which has a capacity of 15 tons per hour and can be operated by one man.

History

The mines were opened in the early eighties, and as no recovery could be made by amalgamation or concentration, the first ore produced was shipped by wagon and railroad to the smelter at Omaha. The freight and treatment charges amounted to \$40 per ton. In 1892 a smelter was built in Deadwood and the cost of treatment was reduced at first to \$16 and gradually during the next 10 years to \$8 per ton. After producing ore to the value of over \$1,000,000, the Buxton Mining Co., owning

The mill is of interest historically as having passed through the whole period of American cyanide practice. When designed, leaching was the standard method of treatment. Slime was the *bête noir* of the millman, both on account of the difficulty of separation from sand and the recognized loss in decantation, which was the only method of slime treatment in use in this country.

Development of Sand-Slime Separation

The ore handled at this mill produced a great deal of slime, with the attendant difficulties in treatment. A system of double cone classification was originally installed, but failed utterly to make a satisfactory separation of sand and slime. The frequent changes in the nature of the ore aggravated the conditions and it was impossible to produce a leachable sand or a slime free from coarse sand. The constant attention demanded by the cones added also to the expense of operation.

The difficulties thus encountered were finally overcome by the invention of the Dorr mechanical classifier in July, 1904, and this machine proved responsible for the ultimate success of the mill. The original classifier was replaced in 1908 by a Model B machine, and the latter was in continuous operation from that time until the mill closed. The total cost for repairs during that period was not over \$5, and the classifier is apparently in condition to run another five years.

The use of this machine and of other mechanical classifiers has since spread over the whole country and into most of the cyaniding districts of the world.

Another item of interest is the fact that this mill was the pioneer in the Black Hills in the use of electric power purchased from a central plant. It was also the first in the district to use Chilean mills for fine grinding and belt elevators for raising pulp ground in cyanide solution. Finally it had the distinction of using the first vacuum leaf-filter which remained in successful operation for any length of time.

The total quantity of ore milled was approximately 285,000 tons, of which about three-fifths came from the firm's mines and the balance from custom ore. Owing to the operation of the mill as a custom plant, no figures on operating costs or extraction have been published.

The total production has approximated \$1,500,000 and the net earnings, after deducting the cost of the property and all improvements, have been several times as great as estimated when the property was bought, although the actual value per ton of ore proved to be less than originally estimated.

Coarse Crushing and Sampling

Company ore was delivered by surface and aerial tramways to a 135-ton bin at the top of the mill. The quantity of ore thus received was estimated by periodically weighing the mine cars and applying average weights to the number of loads delivered. Custom ore was received by railroad and accurately weighed on track scales. It was then dumped into one of two bins of 40 and 60 tons capacity, situated under the tracks about 80 ft. from the mill, and trammed by hand to the crusher.

All ore was passed over grizzlies of 1½-in. space, and the over-size was crushed in a No. 4D Gates gyratory. The Gates product joined the grizzly undersize and was elevated to the crushed-ore bin of 75 tons capacity. The elevator was 37 ft. high, with 12-in. belt and No. 8 Acme buckets, 12 in. by 6 in. by 5 in.

An automatic sampler was placed at the elevator discharge.

This machine was of local construction, designed by Mr. Dorr. It consisted of a narrow spout which was drawn across the discharge chute at regular intervals, cutting out a portion of the ore stream and delivering it to the crusher floor, where it was further crushed and sampled for assay. The sampler was operated by a stream of solution flowing into a counter-balanced box placed in the main mill building.

Accurate sampling was a necessity at this mill, for during the entire period of its operation the company purchased and treated over \$1,000,000 worth of ore. Figures for the period

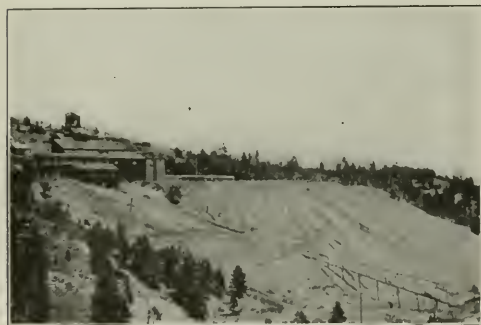


FIG. 1.—LUNDBERG, DORR AND WILSON MILL, TERRY, S. D. VIEW FROM MINE TERMINAL OF TRAMWAY

the property, stopped operations and turned the mines over to leasers. Later the property was sold for a nominal sum, as being entirely stripped of ore that could be profitably handled. The new owner leased it to John Lundberg, and for nearly two years the ore mined was treated at a custom cyanide mill operated by Lundberg & Dorr at Deadwood. About 10 years later the firm of Lundberg, Dorr & Wilson was organized and bought the property, believing that it still contained a large amount of \$4 to \$6 ore that could be made to yield a profit by treatment in a modern mill at the mines. Accordingly construction of the present mill and tramway was begun in 1903 and operations commenced in January, 1904.

The estimated mill capacity of 75 tons per day was gradually increased to 110 tons; and combined mining and milling costs were reduced until ore averaging \$3 per ton could be profitably handled.

1907-1912, inclusive, show that the average value of all ore treated was \$5.11 per ton, and during this time there was a variation of but 1 cent per ton between the value indicated by the automatic sample and another taken from the feed to the Chilean mill.

Grinding in Cyanide Solution

From the crushed-ore bin the ore was fed by a shaking-trough feeder to a set of 14-in. by 30-in. Carterville geared rolls. A spray of cyanide solution was applied from above to keep the rolls clean and the crushed ore was sluiced to the Chilean mill by an additional stream of solution. Operated in

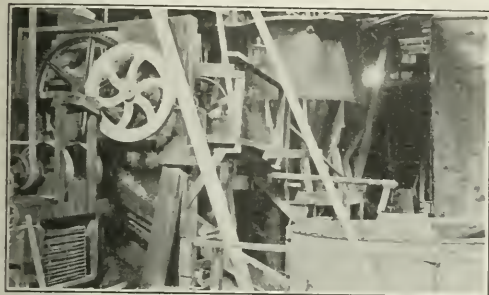


FIG. 2.—GEARED ROLLS, CHILEAN MILL AND ELEVATOR

this way, wet roll-crushing proved entirely satisfactory. Fig. 2 gives a view of the rolls and Chilean mill.

The Chilean mill was a 6-ft. Trent Monadnock, fed through a launder entering the housing about 6 in. above the screens. This method of feeding was adopted several years ago on information received from Mr. Orford, of Delamar, Nevada. It causes no irregular wear of the die and reduces wear on the feed launder supplied with the mill.

This Chilean mill made a remarkable operating record. The bronze spindle bushing supplied with the mill was still in use when the plant was closed. The original gear wheel was changed two years ago after six years' service, when the original spindle was replaced with another of improved type, designed at the Mogul mill, at Pluma.

Additional cyanide solution is fed to the Chilean to make the ratio of solution to ore about 5:1. Tyler ton-cap screens of different mesh have been used with satisfaction. The following tabulation gives screen analyses of the ground pulp for two different screens and ratios of solution to solids:

Mesh.	Ton-cap screen 0.025-in. slot.	Ton-cap screen 0.020-in. slot.
	4.53 tons solution per ton ore.	6.66 tons solution per ton ore.
+ 20	0.60%	0.00%
+ 30	8.22	4.72
+ 40	7.38	5.12
+ 60	16.82	15.95
+ 100	18.24	18.30
+ 200	15.50	21.66
+ 200	32.04	34.25

The metal consumed in the various grinding machines, per ton of ore treated, is shown in the following table. The figures are based on the assumption that the entire metal is used:

Machine.	l.h. Metal per ton ore.
Rolls	.0002
Chilean mill	
Tires	.4046
Dies	.3024
Gyratory	
Concaves	.0012
Head	.0357

The average life of the different crusher parts was: Roll

shells, 10 mo.; Chilean tires, 4 mo.; dies, 3 mo.; gyratory concaves, 16 mo.; heads, 12 mo.

Sand-Slime Classification

The Chilean mill product was elevated by belt and bucket elevator fitted with No. 14 Salem buckets, 12 in. by 6 in. by 5 in. Only two sets of these buckets were used during eight years of operation. Belts lasted about two years. The elevator housing was of canvas, protected with an inner lining of steel at points of excessive wear. The elevator discharged into a Dorr duplex Model B classifier, the sand product of which passed to leaching vats and the slime to cones and a Dorr thickener. The separation produced about 45 per cent sand and 55 per cent slime, the following being the exact figures for the past four years:

	1909	1910	1911	1912
Per cent sand.....	45.0	48.6	52.0	47.2
Per cent slime.....	55.0	51.3	47.9	52.8

Sand Leaching

The sand from the classifiers was sluiced with solution, through launders having a pitch of $1\frac{1}{2}$ in. to the foot, to the leaching vats. A view of this part of the mill is given in Fig. 3, which shows the 8-arm, ball-bearing wooden distributor which served the line of vats. The latter were 18 ft. diameter and 11 ft. deep. Their original capacity was increased to 106 tons by the simple expedient of an inside launder raised 12 in. above the rim, as is shown clearly in the figure. During the early operation of the mill the sand vats were discharged by hand through bottom gates, and the dry sand was trammed to the dump in 4-ton cars of the Mercur type. The cost of this operation was \$6 per tank. Later the system was changed and the vats were adapted to discharge by sluicing.

The total time allowed for sand treatment was five days, of which 32 hours was required for filling a vat, 12 hours for leaching with solution and the balance for washing with barren solution and water.

Vacuum Filtration for Slime

Two settling cones, one 18 ft. and the other 22 ft. diameter, were formerly used for thickening slime for filtration. They were so placed in the mill that the clear, overflowing solution would gravitate to the grinding apparatus. The usual difficulty was experienced with these cones, viz., that solid slime would build up on the sides, making it impossible to secure a constant underflow of uniform consistency. Dissatisfaction

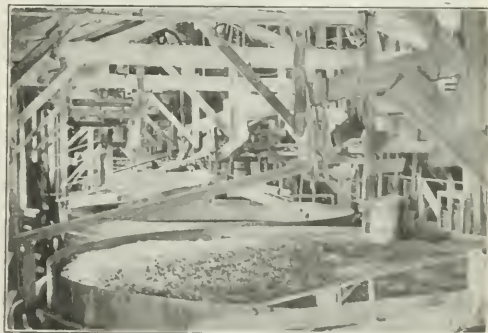


FIG. 3.—SAND LEACHING VATS AND DISTRIBUTOR

with these devices led to the invention of the Dorr continuous thickener at the Mogul mill in Pluma, and when more settling capacity was required at the mill under consideration, one of the sand vats was equipped with the thickening mechanism and proved of great benefit. This thickener is 18 ft. diameter and 8 ft. deep, and has a capacity of from 40 to 50 tons in 24 hours. It was operated at 1 to 1.5 r.p.m., and actual power meas-

urements showed a consumption of only 1/20 hp at the pulley.

The system of slime treatment was very simple. The slime from the classifier was divided between the cones and thickener. The clear overflow from the former gravitated to the roll-supply tank and from the latter to the Chilean mill. Thick-

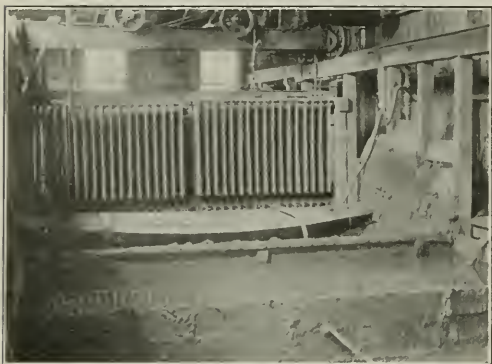


FIG. 4.—MOORE VACUUM FILTER

ened slime from the Dorr machine and from the 18-ft. cone was pumped by air-lift to the 22-ft. cone, and thence to an 8-ft. air-agitation cone which discharged to the loading tank of the Moore filter.

The filter equipment comprised five tanks, one each for loading, barren solution wash, water wash, discharging and acid treating; two sets of 34 filter leaves, 6 ft. 6 in. by 4 ft. 6 in.; and a 10-ton hydraulic crane operated under pressure of 105 lb. The loading and discharge tanks had hopper bottoms. The



FIG. 5.—SAND TAILINGS IN FOREGROUND, STACKED SLIME TAILINGS IN BACKGROUND

cycle of filtration varied with the condition of the leaves. The loading time was 25 to 35 minutes. Barren solution wash was applied for 45 to 60 minutes and wash water for 25 minutes. Ten minutes was required for discharge of cake and

return of filter to the loading tank. The moisture in the feed was usually maintained at from 60 per cent to 70 per cent, and the discharged cake contained 25 per cent to 35 per cent moisture. A view of the filter is given in Fig. 4.

Until two years ago the filtered slime was trammed from the mill and spread over the hillside. About that time the creek was piped for some distance along the foot of the dump and it became possible to sluice tailings to the bottom of the gulch. The early necessity of discharging nearly dry tailings and holding them on the side of the gulch, as well as the anticipated efficiency in slime filtration, led to the adoption of the Moore filter process. By not discharging the cake under water, it has been possible to produce tailings dry enough to be retained on the dump without building walls or dams. The view given in Fig. 5 shows sand tailings in the foreground and slime tailings banked in the background.



FIG. 6.—COKE FURNACE FOR MELTING BULLION

The mechanical loss of cyanide was determined by a very careful series of tests to be 0.3 lb. when washing with a 1-lb. barren solution.

Precipitation and Refining

Gold-bearing solution from the slime filter and the sand-leaching vats was precipitated on zinc shavings. The latter was packed in 30 barrels, arranged in 10 sets of 3 each, having a total capacity of 60 cu. ft. of shavings. Precipitation has been entirely satisfactory, the gold value of the tailing solutions ranging from 2 to 6 cents per ton.

It was the usual practice to make a clean-up twice a month, at which time all the contents of the head barrel and part of the second and third barrels were removed. The coarse zinc was sorted and returned to the second barrel and the fine zinc and gold precipitate refined. The barrels were then moved up one step, the second being placed at the head and the first at the foot of the series. A trolley was used for handling the barrels. The gold precipitate and fine zinc was treated with sulphuric acid, washed, filtered and dried in a shelf furnace, where it was exposed to the gases of combustion. The dried precipitate was then fluxed and melted in the coke furnace shown in Fig. 6.

One reason for the success of the property has been the indefatigable energy and careful attention of Mr. J. M. Pennington, Jr., the manager. Mr. Pennington entered the employ of the firm as assayer at the beginning of operations and gradually assumed more responsibility until he became manager. Many of the men employed in the mines and mill, including the shiftmen, were with the firm during nearly the whole time of operation, and by their faithful and intelligent service contributed materially to the success of the venture.

The Arizona Copper Co. produced only 1300 tons of copper during July, the output being interfered with by the work of enlarging No. 6 concentrator.

An Electrical Method for Comparing Thermal Conductivities

By Wm. Eves, 3rd

Most methods for measuring or comparing thermal conductivities require the determination accurately of the working temperatures of the source of heat and of the sink by which the heat is carried away. Although there are many very accurate ways of determining temperatures by electrical means and otherwise, yet in the particular case of heat flow, conditions arise which cause errors in obtaining the actual temperatures of the points required.

The method for comparing thermal conductivities, which is described in this article, was suggested and devised by Dr. E. F. Northrup, who greatly assisted the writer in carrying it out. It does not give thermal conductivity in absolute measure, but by means of it, one substance may be compared with a standard substance of which the thermal conductivity is known.

The method was first used for comparing the thermal conductivities of commercial impregnating compounds such as are used for the field coils of electrical machines. The operation of this measurement is described in this article. It is now being extended to the comparison of any solid substances, a report of which will be given later.

Briefly, this experiment consists in the heating of two coils of copper wire, identical in every respect, namely, size, shape, electrical resistance and in insulation, except for the compounds with which they are impregnated and of which the conductivities are to be compared. This heating is done by passing such currents through them that will cause them to become each equal in resistance to some set resistance, which is previously calculated and the magnitude of which determines the mean temperature of the coils. Sufficient time must be taken to allow a steady state of heat flow to be reached, in order not to obtain false results.

If this steady state has been reached and if the external temperatures are maintained equal, the total energy input into each coil will be equal to the heat flow and therefore to the energy dissipated by the coil. Since the coils are identical in size and shape, the thermal conductivities of the wire, insulation and impregnating compounds of the coils will be proportional to the amount of energy input, since the material of greater conductivity allows proportionately greater energy to be conducted through it. The watts dissipated from the two coils will be:

$$\text{Coil A: } W_1 = C_1^2 R_t \quad (1)$$

$$\text{Coil B: } W_2 = C_2^2 R_t \quad (2)$$

where R_t equals the resistance of coils A and B at the higher temperature t , and C_1 and C_2 are currents passing through A and B, respectively.

Therefore,

$$\frac{k_A}{k_B} = \frac{W_1}{W_2} = \frac{C_1^2 R_t}{C_2^2 R_t} = \frac{C_1^2}{C_2^2} \quad (3)$$

where k_A = conductivity of coil A and k_B = conductivity of coil B. Hence it follows that for equal mean internal temperatures steadily maintained for both coils, their thermal conductivities will be in direct ratio to the squares of the currents passed through each, which are measured as described below.

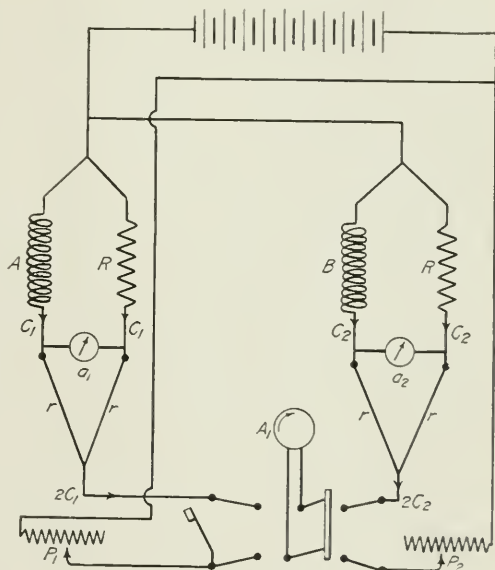
Four identical coils of No. 20 double cotton-covered copper wire were wound on a form so that they were 10 cm in diameter and 10 cm long in order to obtain minimum surface for a given weight. They were bound with tape to keep from falling apart and the forms taken off, leaving a 1-cm hole running through the center. Three of these were then impregnated with three different commercial insulating compounds, which are called A, B, C and the fourth coil D was left unimpregnated. These coils were adjusted so that their resistances were equal to 1/5 of 1 per cent at the same tem-

perature, and the two to be compared were placed in the apparatus as shown by the coils A and B in Fig. 1.

In main, the apparatus consists of two Wheatstone bridges, in which the heating current passes through the bridges. Coils R,R are adjustable resistances of wire of very low and equal, or zero, temperature coefficient, which are set to the calculated resistance which the coils A and B will have at the temperature at which it is desired to take the measurement.

Resistances r, r, r, r are ratio resistances, equal in value. In this case they were made of nichrome strip mounted on a board with binding posts for terminals. They should be low in resistance and with large area in order to prevent heating to excess. They were also adjusted to within 1/5 of 1 per cent and were taken equal approximately to 1/10 of the coils R,R.

a_1 and a_2 are small portable galvanometers, which were found sufficiently sensitive for this method. In fact millivoltmeters could be used. A_1 is an ammeter connected through a double-throw switch and short-circuiting switches, and of suitable range to measure the total currents passing through the



METHOD FOR COMPARING CONDUCTIVITIES

bridges, which are each twice the currents flowing through the coils A and B, or R,R, since the arms of the bridges are equal in resistance. P_1 and P_2 are variable rheostats of any rough type for adjusting the impressed voltage.

Current was passed through the bridges until the detectors showed that the mean internal temperatures of the coils A and B had risen until their resistances were equal to the set resistances R,R. Thus the internal temperatures of A and B are identical provided the copper in each has the same temperature coefficient. But if they should differ in temperature coefficient, coils R and R can be adjusted to the proper resistances by knowing the coefficients, so that they will come to the same temperature when the instruments a_1 and a_2 show balances in the bridges.

The coils A and B have the same external temperature provided the proper precautions have been taken, such as a constant temperature bath well stirred. But in this case the coils were placed very near together in the air, so as to more nearly represent actual conditions on an electrical machine and still be certain of having the same external temperature. The experiment was carried out in a closed room, free from drafts.

so that this steady state was quite easily maintained, as indicated by the detectors.

Coils B and C were each compared with coil A, in order to obtain comparisons of the impregnating compounds, and also coils A and D were compared, in order to compare the ability of an unimpregnated coil to get rid of its heat with the ability of an impregnated coil, and the results showed that the dead air pockets between the wire layers are relatively poor heat conductors. In each case, thermometers were placed in the central holes of the coils, merely as a check. One was also placed about half a meter away from the coil to determine the external or room temperature.

These temperatures were not at all necessary except to have some idea as to the temperature difference between inside and out, since the experiment was continued by making the measurements at various differences of temperatures by raising the internal temperature. It was found that the ratio of the conductivities did not vary within the limits of experimental error for different differences of temperatures, although it has been shown by other experimenters that in wider ranges the conductivity varies.

The numerical results of the measurements are as follows: Resistances of coils A, B, C and D at 23.7 deg. C. = 8.91 ohms. Run No. 1, Feb. 28, 1913. Observer W. E. Comparison of coils A and B.

$$R = 8.91 [1 + 0.0038 (80 - 23.7)] = 10.806 \text{ ohms.}$$

$$r = 1.156 \text{ ohms}$$

Current coil A, $2C_1 = 2.223$ amps. Current coil B, $2C_2 = 2.098$ amps.

Mean internal temperature = 80.0 deg. C.

Observed room temperature = 23.9 deg. C.

Conclusion: $k_A : k_B = 112.3 : 100$.

Run No. 2, March 12, 1913. Observer W. E. Comparison of coils A and D (coil D unimpregnated).

$$R = 10.806 \text{ ohms.}$$

$$r = 1.150 \text{ ohms.}$$

Current coil A, $2C_1 = 2.268$ amp. Current coil D, $2C_2 = 1.880$ amps.

Mean internal temperature = 80.0 deg. C.

Observed room temperature = 23.9 deg. C.

Conclusion: $k_A : k_D = 145.5 : 100$.

Run No. 3, March 15, 1913. Observer W. E. Comparison of coils A and D (temperature of coils higher than No. 2).

$$R = 8.91 [1 + 0.0038 (105.7 - 23.7)] = 11.619 \text{ ohms.}$$

$$r = 1.150 \text{ ohms.}$$

Current coil A, $2C_1 = 2.715$ amps. Current coil D, $2C_2 = 2.228$ amps.

Mean internal temperature = 105.7 deg. C.

Observed room temperature = 23.5 deg. C.

Conclusion: $k_A : k_D = 148.5 : 100$.

N. B.—During this run the resistance of coil D increased to 8.98 ohms at 25.3 C., causing lower current value for coil D, and therefore a greater ratio of conductivities than is correct.

Run No. 4, March 29, 1913. Observer W. E. Comparison of coils A and D (at still higher temperatures than No. 3).

$$R = 8.91 [1 + 0.0038 (134 - 23.7)] = 12.634 \text{ ohms.}$$

$$r = 1.148 \text{ ohms.}$$

Current coil A, $2C_1 = 3.124$ amps. Current coil D, $2C_2 = 2.583$ amps.

Mean internal temperature = 134.0 deg. C.

Observed room temperature = 25.0 deg. C.

Conclusion: $k_A : k_D = 146.3 : 100$.

Run No. 5, April 1, 1913. Observer W. E. Comparison of coils B and C:

$$R = 10.806 \text{ ohms.}$$

$$r = 1.105 \text{ ohms.}$$

Current coil B, $2C_1 = 2.102$ amps. Current coil C, $2C_2 = 2.263$ amps.

Mean internal temperature = 80.0 deg. C.

Observed room temperature = 23.5 deg. C.

Conclusion: $k_C : k_B = 115.9 : 100$.

Summary:

Comparison of coils A and D at different temperatures:

$$k_A : k_D \text{ at } 80.0^\circ - 23.9^\circ = 145.5 : 100.$$

$$k_A : k_D \text{ at } 105.7^\circ - 23.5^\circ = 148.5 : 100 \text{ (see note Run No. 3.)}$$

$$k_A : k_D \text{ at } 134.0^\circ - 23.7^\circ = 146.3 : 100.$$

Comparisons of coils A, B, C, and D at $80.0^\circ - 23.5^\circ$ and 23.9° :

$$k_C : k_A : k_B : k_D = 100 : 96.9 : 86.3 : 66.6.$$

The resistances were always measured before and after each run and in no case, except the run noted was the error greater than 0.4 of 1 per cent. Briefly, the observations show that the ratio of the conductivities changed little if any in the range of temperature used. Two of the compounds tested were far superior to the third and all three had much higher conductivities than an unimpregnated coil, showing that the impregnating compounds tested will not only make a better insulation, but will conduct away the heat faster than trapped air.

The most necessary precaution is to continue the experiment for several hours after an apparently steady state has been reached. In most cases, an observation can be completed in twenty-four hours if readings are taken and the current values changed once an hour or probably more often. A change of one-tenth degree in the difference between internal and external temperatures, caused approximately a change of one scale division of the galvanometer, but such accuracy in temperature is not claimed. Nevertheless, it shows the sensitivity of the method. The accuracy desired at the beginning of the experiment was within 1 per cent., and it is safe to assume that this was obtained, since all resistances checked better than 0.4 of 1 per cent., and the current could be read to better than 0.4 of 1 per cent. The source of e.m.f. used was from 24 to 36 volts direct current, and by means of a flexible system of rheostats the adjustments and measurements are very easily made. The apparatus may be set up in a small space and may be made up entirely of commercial apparatus, the accuracy of which, of course, should be checked, as was done with all instruments in this experiment. The method is commercially practical, in that a measurement may be very quickly made after a short experience and with a little foresight, and also since it is not necessary to make any corrections whatsoever due to theoretical inaccuracies.

PALMER PHYSICAL LABORATORY,
Princeton, N. J.

Electric Furnaces, Their Design, Characteristics and Commercial Application

By Woolsey McA. Johnson and George N. Sieger

In presenting this series of articles to the public, the authors wish to state their three-fold aim as being to give in concise form the scientific, engineering and commercial principles and facts involved in the industrial utilization of electric furnaces. The serial embodying this idea is designed for three classes of readers, to wit:

1. Engineering students who wish to learn the technology of electric furnaces.
 2. Engineers who are desirous of having in convenient form a "desk-book" covering the latest intensive application of electrical energy.
 3. Business men who might need to learn quickly for a specific purpose the general principles of electric furnaces.
- As the business principles of any applied science are in reality the most important and oftenest neglected, the third aspect has been emphasized everywhere, in order that the authors and the readers might have conjointly in mind the relation between the dollar and the kilowatt hour.

A short ton (a long ton being practically equal to a metric ton) of average coal with a heating value of 11,600 b.t.u. per pound contains exactly the same heat as a horsepower year. The ton of coal costs from \$1.00 to \$8.00 with an average value f.o.b. factory for the country of \$2.00. A horsepower year costs from \$10.00 to \$50.00, with an estimated average value at the American electro-metallurgical plants of \$18.00. It is seen clearly the cost per unit of heat, e.g. b.t.u. or caloric, is greatly in favor of coal or other fuel as gas or oil and that

electric heat is only applicable where certain metallurgical peculiar advantages exist or where its application is so convenient and so efficient in other ways that its primal disadvantage of large cost per calorie is overcome by other inherent but superior characteristics.

With this thought held prominently in mind, the scientific facts and practical details become aids and props and without it the commercial advance is greatly hindered. While much that is apparently useless to the practical man will be taken up, sooner or later he will appreciate its value and the scientific and commercial facts which he has mastered will yield him an increased profit.

In order to have a dividing line between electric furnaces and other classes of electric heating apparatus such as electrically heated ovens, etc., an arbitrary line has been drawn and the authors do not consider as an electric furnace any electrical heating apparatus where the working temperature is below the minimum visual red, e.g. 550 deg. C. or 1022 deg. F.

Article I—General Principles of Electric Furnaces Design and Economics

The commercial application and the cost of heat of course govern the industrial progress of metallurgy. It is a fact that electric heat is expensive and to be used at all must be used efficiently.

The essential differences between heating with carbonaceous fuel and electrically where there is no radiation or heat lost (there are never any furnaces where there is no radiation, but for purposes of illustration such ideal furnaces are a convenient assumption) are expressed as follows:

a. Flow of heat from where it is generated to where it is usefully absorbed is a function of the temperature difference and a constant of resistance, and a law analogous to Ohm's law obtains.

$$1. H \text{ (Calories transmitted in unit of time)} = \frac{T_s - T_{ab}}{K_r}$$

Where K_r is the constant of thermal resistance for any given furnace, T_s temperature of source of heat and T_{ab} temperature of absorption or the furnace's working temperature. K_r will vary with size and type of furnace.

b. In "carbon-heated" furnaces T_g has a definite upper limit because not enough heat is generated to do more than raise the products of combustion above a certain maximum temperature, e.g., Richard's Metallurgical Calculations, part I, page 36 *et seq.*

c. Where T_{ab} or working temperature is high, e. g., in a ferrosilicon furnace, $T_g - T_{ab}$ must be small. K_r is a constant, hence heat imparted usefully is small because the temperature difference between source of heat and place of heat absorption is small. Finally when this difference becomes zero no heat is usefully transmitted and the operation becomes uncommercial. But the temperature difference is the driving force.

d. K_r the constant of thermal resistance of any given furnace depends on its size and shape. These are governed by the design, and the design limited by nature of reactions proceeding. For instance, in an induction furnace K_r is extremely small, for the heat is generated directly in the metal bath, whereas in a reverberatory arc or resistance type as in De Laval zinc furnace, the heat must be reflected down and transferred by radiation, convection or conduction. Hence, K_r is in this case large, and to attain a high thermal efficiency T_s must be high, 2500 deg. C. plus, and only limited by danger of melting down arch.

Let us now consider, in rough approximation, the limits of carbon-heated furnaces and electric furnaces when there is no radiation. No mathematical analysis is needed because the principles stand out in bold simplicity. These principles are:

(1.) The heat generated and imparted per cubic foot of furnace laboratory (working cubical contents of furnace) is several times as great in the electric furnace as in a "carbon-heated" furnace.

(2.) The percentage of heat wasted in radiation and conduction through the furnace walls therefore must be much less in an electric furnace than in a carbon-heated furnace and hence entirely apart from consideration of equation 1 electric furnaces are usually much more efficient thermally considered than their rivals. Moreover, the design of the shape of walls and materials used often allows better heat insulation.

When we consider the fact that radiation is always much less and sometimes several times smaller, the thermal efficiency of the electric furnace is many times higher.

Returning to mathematical expression, equation 1 can be transformed thus:

$$2. H \text{ (Cal. transmitted per unit of time)} = \frac{T_s - T_{ab}}{K_r} - R$$

where R equals heat lost by radiation to the surroundings of the furnace.

$$\frac{T_s - T_{ab}}{K_r} - R$$

$$3. \text{ Thermal Efficiency} = \frac{H}{\frac{T_s - T_{ab}}{K_r} - R}$$

A consideration of equations 1, 2 and 3 shows that in certain cases thermal efficiency becomes quickly zero and it would be unprofitable to attempt to perform any thermal or thermochemical work above the normal limits of T_g , which are fixed strictly by the laws of chemistry and physics at 1600 deg. C. to 1900 deg. C. in the cases of most furnaces heated by fuel or, to use a new term which we will adopt, "carbon-heated" furnaces.

For instance, in the manufacture of ferrosilicon the iron blast furnace is now more economical, making a product of 80 per cent Fe, 20 per cent Si, but the economical thermal limit on which is superimposed the metallurgical limits of the need of an intensely reducing atmosphere (much easier attainable in an electric furnace where no air is blown than in a blast furnace where air must be blown in to burn up the coke and thus make heat). But above this percentage the blast furnace does not compete with the electric furnace and all ferrosilicon about 20 per cent Si is made today in electric furnaces. The electric furnace competing with itself makes ferrosilicon in most places most economically of 60 per cent Si and 40 per cent Fe.

A study of the theoretical temperatures necessary in the manufacture of ferrosilicon and the theoretical temperatures possible with a coke furnace supplied with hot blast indicates that as T_{ab} increases with an increase of percentage of silicon in product, the quantity $T_s - T_{ab}$ becomes smaller or the direct thermal "drive" is less, for T_s is fixed by laws of thermochemistry and physics. It is thus costing much more to transfer heat to the charge, we find that as the percentage of silicon increases electricity is a better commercial agent than its rival, because the temperature required for reaction is higher.

When the electric furnace begins to compete with itself, we find that as the requisite working temperature increases, the radiation from the furnace is so increased that a large amount of money is lost in heat dissipated to the atmosphere. So, it is better business to make a ferrosilicon high enough in silicon to stand a shipment but not so high that kw-hours per lb. of silicon is a large number. With cheap power it would be good business to make a richer ferrosilicon, other things always being equal.

Let us reiterate that as the cost of an electrical calorie is three to ten times greater than the cost of a "carbon" calorie, electric heat is only commercially applicable because of its efficiency in other lines.

With this in mind, in the comparison between the electric furnaces and their rivals and taking into consideration the design of electric furnaces in light of this comparison, it becomes self-evident that there are certain inherent chemical or, better expressed, metallurgical and commercial points of superiority of the former over the latter.

In any case where strongly reducing conditions above 800 deg. C. are needed the electric furnace begins to become the better to use, and as the temperature requisite becomes higher their inherent chemical superiority is increased. The commercial inherent superiority is also increased because such heating under reducing conditions must be done indirectly through the walls of a muffle, retort, or crucible, and as the temperature is increased the size of the unit is decreased. Hence the labor charge is greatly increased and labor-saving devices so necessary to-day cannot be used. Such conditions exist in the zinc retorting plant, the brass foundry, the crucible steel plant, and the gradual commercial elimination of these three is inevitable.

As we go higher in the scale of reductivity and temperature we see the calcium carbide, the carborundum, the alundum, and the graphite furnace where no competition by coke-fired furnaces can exist, for the reactions both chemical and physical are scientifically impossible as well as commercially impossible save in an electric furnace. By a natural industrial evolution these operations have arrived as commercial successes quicker and in the main easier than in the other cases above mentioned.

The following facts can therefore be laid down as true:

(1) The relative thermal efficiency of carbon and electric furnaces are such that the higher the temperature and the more difficult the metallurgical work to be done, the more advantageous is the electric furnace.*

(2) The relative commercial efficiencies of the carbon-fired and electric furnaces are such that the harder the metallurgical work to be done the greater the inherent points of superiority of the electric furnace.

As the design and characteristics of electric furnaces become better understood, other advantages, it can be prophesied, will be more prominent.

These are:

- (1) Control of temperature.
- (2) Control over quality of product.
- (3) Decreased cost of repairs and renewals.
- (4) Adoption of machinery for handling raw materials and products.
- (5) Decreased capital cost because of intensity of heat transfer.

(6) A more efficient use of high-priced intelligent labor.

These six sub-advantages must always be considered in the general design of electric furnaces, even though their existence is not so apparent.

In general, it can be stated as a fact that the closer the source of heat is to the place of the absorption of the heat by the charge, provided there is a regular flow of electrical energy and charge, the closer will be the control over temperature and over quality of product. Thus, we are also making thermal efficiency higher by making K_r smaller.

For this reason a number of smaller electrodes is better than a few large ones. Large thick walls and a big bath of slag or metal act as thermal balance wheels and keep conditions more uniform. Increase of size of unit, provided the characteristics of the furnace and process are really understood, also make for all of the above sub-advantages.

Some Practical Types of Electric Furnaces

Unquestionably there is no form of electric furnace simpler than the "DIRECT RESISTANCE" type invented by A. H. & E. A. Cowles in 1879. This as used by Acheson in his carborundum and graphite furnace is simple, practical and efficient. The disadvantage of building a new furnace after each run, the way the Chinese ancestrally have done in iron smelting, is of course apparent, but the fact that there is a monopolistic market for the product renders it unnecessary to make metallurgical and mechanical refinements. The commercial aim is to turn out the goods and sell them. Salesmanship is the essential thing after the goods are made.

A much more finished metallurgical instrument is the "ARC FURNACE OF THE SIEMENS TYPE." This as developed by Heroult

and engineers of the United States Steel Corporation, by Girod and others for making a high quality of steel on a tonnage basis with automatic feed of electrodes in a tilting furnace, is pretty near a finished invention.

The "REVERBERATORY" ARC FURNACE where the arc passes from one upper electrode to another and as developed by Stassano for steel, de Laval and Weeks for zinc dross and zinc ores, has a much narrower field than was supposed recently. The fact that the heat is generated some distance from the charge and reflected down from the arch makes the transfer of heat slow and costly. Moreover, the danger of melting down the arch, common in open-hearth steel furnaces and copper reverberatories, is multiplied several times in this type of a furnace.

The "BURIED ARC" type as used for the manufacture of calcium carbide or the slag resistance type, for the two blend into each other, such as used by Irvine for the production of phosphorus is a furnace capable of wide metallurgical application. Both the two main advantages and the six sub-advantages are present in natural and balanced proportions. Rightly applied and designed for continuous work so that each part of the furnace is functioning at a high efficiency and delivering its charge to parts below it with charge progressively operated upon, these types of furnaces have great commercial possibilities.

Electrical Heat Must Be Applied Efficiently

Considered in another aspect, we have the subject of pre-heating by heat generated by carbonaceous fuel. As has been shown, the imparting of thermal energy by electrical means to a charge becomes commercially much more efficient as the temperature increases. Thus it is good common sense and sound business to do where possible low-temperature heating by the heat of combustion of carbonaceous fuel until its efficiency becomes low and the high-temperature work by electrically generated heat. Col. Robert M. Thompson, of the International Nickel Company, former employer of one of the authors, enunciated this principle of economic electrometallurgy: "*Electricity is an expensive agent and should only be used for the finishing metallurgical operation.*" Applying this principle directly, it can be stated axiomatically that in any metallurgical process, especially that of a commercial nature, the first work to be done should be done by fuel-generated heat and the last work to be done should be done by electrically-generated heat.

Broadly speaking, the electric furnace process should be so designed that each co-ordinate part does its duty as cheaply as possible and likewise should be considered both as it attains the end and also in relation to the other parts.

Mr. J. E. Johnson, Jr., has analyzed the different functioning parts of the iron blast furnace, the chemical work to be done and the chemical force applied from the tuyeres to the charging floor. He has found that the design of the modern American blast-furnace as worked out by an evolutionary process conforms to the law that there should be a progressive application of metallurgical agents each doing its work at a maximum efficiency.

This principle, which is correspondent with the "Thompson principle," should be applied in the design of an electric furnace and in electric furnace processes.

Thus we find that too large electrodes and the resultant generation of a large amount of heat will often produce "hot spots" in an electric furnace and that this localized and intensified electric heating will produce untoward effects such as burning out the lining of the furnace in proximity of the electrodes or the reduction of oxides to their elemental forms, with the subsequent vaporizing of, for instance, silicon and aluminium.

While for obvious practical reasons it is well to increase the kilowatts generated per cubic foot of working space to a maximum, yet this maximum should be within safe workable limits and no overheating either of the entire furnace or in spots is permissible. The kilowatts generated per cubic foot of

*Note it can be estimated that in general to-day the dividing line is about 1100° C. (2000° F.) with a margin of 200 or 300 C. each way.

working space must then be placed at a safe maximum as allowed by other metallurgical and superimposed particular conditions.

High Voltage a Good Commercial Point

As will be shown in a later article, the power factor of an electric furnace becomes lower as the size of the unit increases and above 1000 kw the inductive drop becomes considerable. Since the amount of copper tied up in the form of bus-bars, switches, transformers and generating plant is so increased without financial return, the furnace should be designed to be operated at as high a voltage as possible, for a high-voltage design will increase power factor and commercial efficiency in other obvious ways.

So too with the "load factor" (the actual output of a station in kw-hours during a given period divided by the output obtainable by operating station at its full rated capacity for that period), for the best power plant operates somewhat irregularly. The electric furnace should if possible be so designed as to absorb the fluctuations of the output of power plant, usefully. By shape of unit, contents and thermal insulation thereof, it is possible to have a number of electric furnace units that will give to a power plant a "load factor" that would be, practically speaking, nearly 100 per cent.

The question of heat insulation is not nearly so important as the layman deems it to be. By increasing to 5 or 10 kw per cubic foot of laboratory it is possible to design even as small a unit as 200 kw with water-cooled walls emitting 0.5 kw per square foot of free surface with a thermal efficiency of 60 to 70 per cent.

Where a corrosive slag is used these walls will have a long life, whereas walls well insulated have a short, uncertain and a merry life.

Far more important is the geographical location of the plant, the kw-hours per pound of product, and the cost of power per kw-hour. The cost of power per kw-hour and the cost of power per pound of product are positive determining factors in the success of any electric furnace process. With the immense amounts of secondary hydroelectric power available at less than 2.5 mills per kw-hour for nine or ten months in the year, it is of prime commercial importance that electric furnaces and companion processes be developed that absorb in a manner useful to the nation this waste power. Undoubtedly this application of the electric furnace will be a larger one in the future.

Summary

This chapter can be summarized as follows:

1. Electric heat is expensive and can only be used where its efficiency is great.
2. Electric heat can be employed in certain work where it is needed badly.
3. If intelligently studied and applied, electric furnacing will become of large industrial importance and the existing limits pushed to extremes not now thought possible.

If an electric furnace be thought of as a short-circuit under control, we must strive to put it under good and intelligent control.

Notes on Chemistry and Metallurgy in Great Britain

(By our Special Correspondent)

Institution of Mechanical Engineers

The summer conference of the Institution of Mechanical Engineers was held at Cambridge. Sir H. F. Donaldson presided and there was a good attendance of well-known members. The two papers taken on the first day were "A New Method of Cooling Gas Engines," by Professor Bertram Hopkinson, and "Modern Methods of Measuring Temperature," by Mr. Robert S. Whipple.

The Cooling of Gas Engines

Professor Hopkinson, in his paper, dealt with the difficulty of the external cooling of gas engines and advocated the injection of water inside the cylinder. He contended that diffi-

culties arising from corrosion and associated with lubrication could be overcome by regulating the quantity of water injected so that the temperature of the engine was maintained above 100 degs. C. Under these conditions the whole of the water injected was raised to boiling point on contact with the walls of the cylinder, and thus no accumulation of liquid was possible, and the drops of water would dissolve very little gas, such as sulphur dioxide, on their way to the walls because of their small surface and the short time—a mere fraction of a second—for which they were in contact with it, and what little they did absorb was expelled by evaporation. Experiments carried out by Dr. Dugald Clerk, the author, and others, had established the fact that the rate of heat-flow from the gas into the metal was more rapid at and soon after the moment of ignition than subsequently, and this led to the supposition that the heat flow into the bore of the cylinder during the last three-fourths of the stroke might be so small compared with that of the initial period that direct cooling of the last three-fourths of the cylinder could be omitted. And this supposition had been found to be correct, and the injection of water on to the surface of the combustion chamber and the piston effected the cooling of the whole of the barrel by conduction into the piston which was kept cool by projecting water onto the head when near the inlet-centre. The author then described experiments with a Crossley engine with a cylinder 11½ in. in diameter, having a 21-in. stroke, rated at 40 b.h.p. at 180 r.p.m., to which a new cylinder without any water-jacket was fitted. The method had also been applied to larger engines, one of which was 18½-in. bore, rated at 105 b.h.p., and another of 36-in. bore, rated at 1000 h.p. All these trials were quite satisfactory and generally resulted in the attainment of a higher efficiency.

The discussion was opened by Engineer Rear Admiral E. Little, who was of opinion that if the author's method proved successful there was hardly any limit to the size of cylinder to which it could be applied. However, all the cylinders experimented with were of the horizontal type, and he wished to know if the principle could be applied to vertical engines.

Mr. M. Longridge said that the author's research was based on a scientific foundation, and although his method of cooling applied mostly to cylinders of comparatively large size it should be of value in smaller engines and make them cheaper and more economical.

Another speaker raised the question of erosion in the cylinder, consequent on deposits resulting from the evaporation of water.

Professor Hopkinson replied that he had not yet tried his method on vertical engines, but in the horizontal types with which he had experimented the solids deposited passed away by the exhaust valve. He wished to point out, too, that this method of cooling prevented pre-ignition.

Measurements of Temperature

Mr. R. S. Whipple's paper reviewed the various thermometric methods and instruments used mainly for technical purposes. He said that for steam plant mercury thermometers might generally be employed; but the thermometer in front of the smoke-stack and those on either side of the economizer were troublesome to read and were often neglected, and resistance thermometers which could be read on a galvanometer in the engine room would be preferable. Similarly, the resistance thermometer would be advantageous for cold storage work in which the opening of doors of the rooms for the purpose of reading mercury thermometers involved difficulty of maintaining a uniform temperature and entailed considerable cost for labor. With regard to pyrometers, one of their most important applications to the treatment of metals was the measurement of the temperature of the air in the hot-air mains of blast-furnaces; and for this purpose resistance thermometers with Callender recorders were best. To ensure an equal degree of accuracy with thermo-couples the scale control-board should be employed and due precaution adopted with regard to the cold junctions.

After dealing with the great difficulty of ascertaining the temperature of molten metal for casting, the author said that the heat treatment of metals afforded the greatest field for the use of pyrometers, and success depended on the way in which the pyrometer was mounted in the furnace. Experience had proved that with large furnaces it was advisable to place the pyrometer in the floor of the furnace, as advocated by Mr. F. Culpan.

With regard to the use of Seger cones in porcelain works, the author remarked that while they were of great utility in showing the maximum temperature reached, yet they afforded no index of the rate of heating, which was a most important matter.

For the lower temperature ranges thermo-couples were best, and for the higher ranges radiation pyrometers. The paper concluded by dealing at some length with the standardization of thermometers and pyrometers.

In the discussion Mr. C. Wicksteed remarked on the satisfactory results given by the apparatus described for determining temperatures in annealing furnaces. Dr. Glazebrook said that the greatest possible care was taken in making these instruments, and a very high degree of accuracy had been attained; but there were still difficulties at very high temperatures, and the problem was to find the necessary material for the production of an instrument for measurement up to 2000 degs. C. Sir Robert Hadfield said that at some Sheffield steel works about 5000 pyrometric readings were taken every week. The importance of temperature reading in steel works' practice was demonstrated by the great effects of comparatively small variations of temperature. It was necessary that an efficient method of ascertaining the temperature of molten metal should be available.

The British Association

The autumn meeting at Birmingham will open on the 10th September and continue until the 16th under the presidency of Sir Oliver Lodge. In the chemical section, Professor Soddy will discuss radio-active elements and the periodic law, while Dr. Desch, Dr. Holt, Professor Cohen, Professor Turner and Dr. Rosenhain, among others, will present papers of metallurgical interest.

In the physical science section Professor Barkla will contribute a paper on the nature of the X-rays.

The Institute of Metals

The summer meeting of the Institute of Metals will be held at Ghent on the 28th and 29th August. Professor A. K. Huntington will preside, and the second report to the corrosion committee by Dr. G. D. Bengough and Mr. R. Jones will be presented.

The International Electrotechnical Commission

The Berlin meeting of the Commission will extend from the 1st to the 5th of September. The first two days are to be devoted to special committees which will deal with the subjects of nomenclature, symbols, rating and prime movers. The official opening of the meeting will be on the 3rd, when the president, Dr. Budde, will deliver his address.

The Iron and Steel Institute

The autumn meeting will be held at Brussels, where the Grand Hall of the Palais des Académies has been placed at the disposal of the Institute for reading and discussion of papers on the 1st and 2nd of September. On the 3rd there will be a special excursion to the Ghent Exhibition; for the 4th, visits to the works of the Société John Cockerill, at Seraing; other works near Liège, to Charleroi and to Mons have been arranged.

Engineering Imports and Exports

The returns issued by the Board of Trade for the six months ended on the 30th of June show very satisfactory increases in both imports and exports of all the chief classes of engineering materials and products as compared with the corresponding period of 1912. Imports of iron and steel, including manufac-

tures, amounted to £7,662,575, an increase of £1,948,711; and exports touched £227,932,692, an increase of £6,382,041. Imports of other metals, including manufactures, totalled £16,484,060, a rise of £1,967,937; and exports are valued at £6,898,155, an improvement of £1,355,117. In electrical goods the imports are put at £746,129, an increase of £24,308; and exports went up to £2,682,811, an increase of £616,478. Imports of machinery amounted to £3,757,738, a rise of £330,363, while exports to the value of £18,424,543 show an improvement of £2,735,357. New ships were imported to the amount of £14,345, an increase of £43; and were exported to the value of £4,323,731, an increase of £1,627,141.

The figures for the month of June alone are, excepting a small decline in electrical exports, of the same satisfactory character. Iron and steel values increased £291,164 in imports and £1,117,869 in exports; imports and exports of other metals rose £112,669 and £193,441, respectively; imports of electrical goods advanced £30,399, but exports declined £20,125; machinery showed increases of £51,393 for imports and £674,386 for exports; imports of new ships fell £632, but exports improved by £575,540.

Market Prices, July, 1913

	£	s.	d.
Aluminium, ton	88.	0.	0.
Alum, lump, loose, ton	6.	0.	0.
Antimony, black sulphide powder, ton	20.	0.	0.
Borax, British refined, crystal, ton	17.	10.	0.
Copper sulphate, ton	21.	10.	0.
Carbolic acid, liquid 97-99%, gal.	1.	3	
Caustic soda, ash 48%, ordinary, ton	5.	10.	0.
Ebonite, rod, per lb.	4.	6	
Hydrochloric acid, cwt.	5.	0.	
India rubber, Para fine, lb.	3.	8½	
Mica, in orig. cases, medium, lb.	3/6	to	6/-
Petroleum, Russian spot, gal.	9½		
Quicksilver, bottle (Spanish)	7.	5.	0.
Sal ammoniac, lump, tsts, del. U. K., ton	44.	0.	0.
Sulphate of ammonia, f.o.b. Liverpool, ton	12.	16.	3.
Sulphur, recovered, ton	5.	3.	0.
Shellac, cwt.	4.	13.	0.
Platinum, oz., nominal	9.	5.	0.
Tin ore, ton	£115	to	£117
Zinc, Vieille Montagne, f.o.b. Antwerp	24.	15.	0.

Tin in the early part of the month dropped off considerably. Opening at 193, it had fallen by the 10th to £177; recovered till the 11th, £183; dropped again to £179.10.0 on the 15th and reached £185 on the 24th. Closes £182.

Copper remained about £63 till the 10th, when it dropped to £61.5.0 and remained flat till the 15th, when a steady rise set in. It reached 65½ by the 24th and continued very active, closing at £67.2.6.

Hematite remained flat at 72¼ for three days, then dropped to 71 6, and remained (with a slight dip on the 14th) at or about this level until the end of the month, closing 71/9.

Scotch Pig opened 65/- and closes lower at 64/-.

Cleveland opened 55¼, showed slight falling off from the 17th to 21st, but recovered 22nd. It closes flat at 54/10½.

Lead has risen steadily, opening at £19.15.0, reaching £20 by the 15th and closing £21.5.0.

DIFFERENCES

	Higher.		Lower.
	£ s. d.		£ s. d.
Ebonite rod	6	Aluminium	2. 0. 0
Quicksilver	5. 0	Borax, ton	1. 0. 0
Shellac	4. 0	Zinc	1. 2. 6
Copper	2. 17. 6	Tin	12. 0. 0
Lead	1 10. 0	Hematite	1. 3
		Scotch Pig	1. 0
		Cleveland	7

The Great Falls Electrolytic Copper Refinery*

By Willis T. Burns

These notes are submitted, not as a discussion of the modern practice of electrolytic-copper refining, but as the record of a refinery that was among the pioneers in the field and that is to-day, and has been for 17 years, operating under conditions not to be found in any similar plant in the country. I refer to the current density, which is at least 60 per cent higher than that regularly employed elsewhere, and to the fact that the anodes treated are the direct product of the converter rather than of the reverberatory furnace.

This refinery, now one of the oldest plants of its kind operating in this country, was built in 1892 by the Boston & Montana Consolidated Copper & Silver Mining Company to refine the product of its Great Falls reduction works. The estimated capacity of the plant as designed was 65,000 lb. of cathodes per day at a current density of 16 amp per square foot.

The plant produced its first cathodes in February, 1893, and has been in almost continuous operation since that date. The output of the plant for the year 1893 was 3179 tons. As the smelter output increased, demands for a larger refinery output were made, and as no appropriation for enlarging the plant was forthcoming increased production could be obtained only by increasing the current density. As an abundance of water power was available and preliminary experiments had demonstrated that cathodes of satisfactory quality could be economically produced at more than double the density the plant was then operating at, new and larger generators were installed in 1896 and the current density brought up to 40 amp per square foot. Since that date the current density has been reduced by adding 76 refining tanks and by increasing the number of electrodes per tank.

In 1893, 3179 tons of copper were refined at an average current density of 10 amp per square foot; the production for the year 1912 was 31,596 tons and the average current density 34.1 amp per square foot.

General Arrangement of Refinery

As originally designed and constructed the refinery consisted of the tank house, a brick building 171 ft. 6 in. x 108 ft. (Fig. 1) containing 288 electrolytic-refining tanks of the following inside dimensions: 9 ft. 7 in. long, 2 ft. 4 in. wide and 3 ft. 9 in. deep, built of Western pine. The sides and bottoms were of 3-in. lumber and the ends of 2-in. material. The tanks were tied at the ends with 3-in. x 1/2-in. iron straps held together by 1-in. rods as shown in Fig. 2. The tanks were lined with 3/8-in. chemical sheet lead weighing about 8 lb. per square foot.

The tanks, which were supported in place by two 8-in. x 8-in. timbers which rested on stone piers 18 in. high, were insulated from the timbers by 4-in. x 4-in. x 1-in. glass insulators, eight under each tank. Had 5 ft. or 6 ft. of head room, instead of 18 in., been provided under the tanks, subsequent operations would have been greatly facilitated.

The tanks were arranged in 18 double rows, eight tanks deep (Fig. 1) with an aisle between each double row. The tanks as originally built were all on the same level, making it necessary to establish an individual system of circulation of the electrolyte for each tank. This was accomplished by providing two circulating tanks for each section of 66 tanks; these tanks were 10 ft. square by 8 ft. deep, lined with 12-lb. chemical sheet lead. One of these tanks was placed above the level of the refining tanks and one below, as shown in sketch. From the overhead tank the electrolyte was delivered, through an overhead system of lead pipes, to each separate refining tank. The solution was discharged into the refining tank at one end, near the bottom of the tank, and was allowed to overflow through a 1-in. lead pipe entering the tank 3 in. from the top near the other end. This pipe discharged into an open launder leading to the lower circulating tank, from where it was raised to the overhead tanks by various devices to be described later. This system, which permitted only a periodical circulation of the solution, soon gave way to a more efficient system to be described further on. Heating the electrolyte was effected by passing live steam through 125 ft. of 1-in. 8-lb. lead pipe in each of the overhead circulating tanks.

A 3/4-in. x 4-in. steel trolley track was suspended from the roof timbers over each row of eight tanks. The anodes and cathodes were removed in and out of the tanks by means of

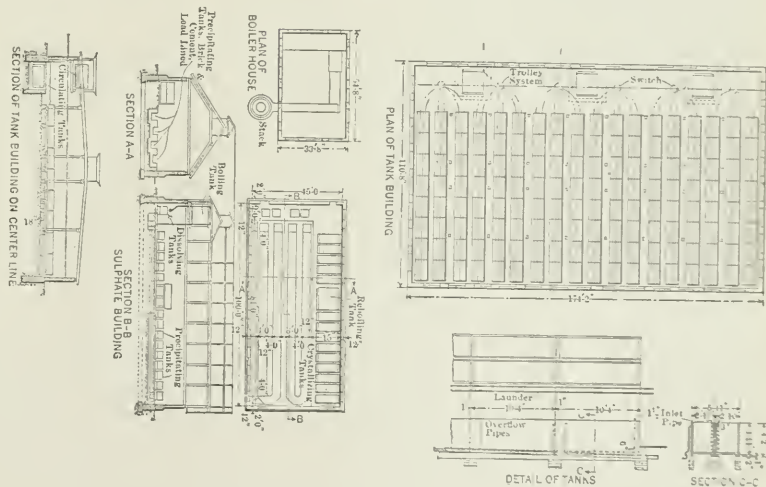


FIG. 1—GREAT FALLS ELECTROLYTIC PLANT AS ORIGINALLY BUILT

chain blocks suspended from two-wheel trolleys on this track.

The current was distributed by means of 3/4-in. x 4-in. rolled-copper busbars attached to the sides of the tanks with wooden brackets. The tanks were all connected in series and the electrodes in multiple.

The original installation included a sulphate plant for the manufacture of bluestone and a boiler house containing one 100-hp locomotive-type boiler to furnish steam for heating and boiling solutions.

The sulphate plant, shown in Fig. 1, was well appointed for the manufacture of bluestone as described later. The capacity of this plant was 2,500,000 lb. of bluestone per year.

The above is a brief description of the plant as it originally existed. Fig. 3 is a sketch of the plant as it exists to-day. One division of 32 refining tanks has been added to the main tank room, 44 new electrolytic refining tanks for the production of starting sheets have been housed in additions as have the 10 new insoluble-anode tanks shown in sketch. An addition has been built south of the tank room in which is installed the equipment for pumping and heating the electrolyte. An addition has been built on the east containing the scale house

*Extracts from a paper read before the American Institute of Mining Engineers in Great Falls, Mont., on August 16th. The complete paper is published in the August issue of the American Institute of Mining Engineers, pages 201 to 2049. As given above, the paper is practically complete in all main points, with exceptions of the chapters on the transmission line and on physical testing.

and a 7-ton traveling crane for handling the cathodes and scrap after weighing. The addition on the north side of the building contains store room, shops and office. Fig. 4 is a flow sheet of the electrolytic plant.

In the new building south of the bluestone building are the carpenter and lead-burning shops and the machinery used in trimming sheets. The four large round tanks situated further south are for the storage of solutions.

The original boiler house has been replaced by a brick building containing two 150-hp Heine and one 100-hp Scotch marine

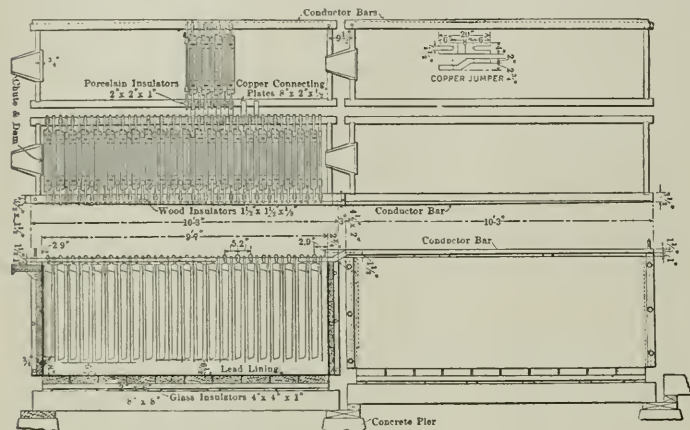


FIG. 2—TANK DETAILS

boiler from which steam for heating and boiling solutions is obtained.

As previously stated the refining tanks, as originally installed, were all on the same level and to obtain a circulation of the electrolyte it was necessary to deliver the electrolyte to each tank through a separate pipe line. This system was found unsatisfactory and the refining tanks were rearranged in a cascade of eight tanks with a drop of $2\frac{3}{4}$ in. between tanks as shown in Fig. 3. The electrolyte is now delivered to the upper tank of the cascade at the surface and is drawn off from a point 8 in. from the bottom at the lower end of the tank by means of the dam shown and is delivered to the next tank through the cast-lead antimony chute shown in Fig. 2. From the last or lower tank of the cascade the electrolyte is returned to the pump room through a 2-in. 7-lb. lead pipe attached to the sides of the tanks.

The original refining tanks were lined with $\frac{1}{8}$ -in., 8-lb. chemical sheet lead and the overflow consisted of a 1-in. lead pipe burned to the lining. Tanks constructed in this manner were found to last about 10 years in service. As they gave out they were replaced by tanks of the same dimensions lined with $\frac{1}{8}$ -in., 7-lb. lead containing 6 per cent antimony. The cast chutes of the same material were built into the ends of the tanks. Tanks thus constructed have now been in service for 11 years and do not show much deterioration. The antimonial lead is harder and has a less tendency to creep and buckle under extreme changes of temperature than is the case with the chemical lead. Practically all of our tanks are now lined with antimonial lead.

When the change from 4000 amp to 9000 amp was made in 1896 it became necessary to increase the size of the tank busbars. The $\frac{3}{4}$ -in. x 4-in. rolled-copper conductors were removed from the sides of the tanks and a taper bar $4\frac{1}{2}$ in. x 2 in. in the center and 1.5 in. x 3 in. at the ends was laid on the top of the tanks, as shown in Fig. 2. The bars were cast at our works from wire-bar copper. These bars are insulated from the tanks by means of $\frac{1}{2}$ -in. x 4-in. pine strips previously boiled in pine tar. These strips are renewed once in six months.

Into the ends of the busbars $\frac{3}{8}$ -in. x 6-in. iron studs are screwed to receive the copper shunt used to cut four tanks out of circuit for removal of slime.

The anode and cathode rods and the connection plates, used between the tanks, are also cast at our furnace refinery in copper molds. Details of tank connections are shown in Fig. 2.

Anodes

As previously mentioned this refinery is the only plant of its kind treating converter-copper anodes. The general practice elsewhere is to transfer the molten copper from the converter to the reverberatory furnace. Here the necessary oxidizing and poling of the copper to bring it to the proper pitch for casting takes place. The copper is then cast into anodes by means of a suitable casting machine.

This refining process has the effect of increasing the copper contents of the metal by from 0.3 per cent to 0.4 per cent, sulphur dioxide being the principal impurity eliminated. The resulting anode is much denser than the converter anode and the casting is free from the uneven surfaces which appear on the converter anode. Analyses of anodes are shown in Fig. 5.

Table I shows one of a number of comparisons, made at this plant, between converter and refined anodes. In conducting this test three divisions, of 32 refining tanks each, were employed. One-half of the tanks in each division were treating refined anodes while the other half contained converter anodes. By this arrangement the effect of the personal equation of the men in charge of the tanks was reduced to a minimum.

It will be noted that while the ampere efficiency of deposit was 3.6 per cent higher in the refined-anode tanks than in the

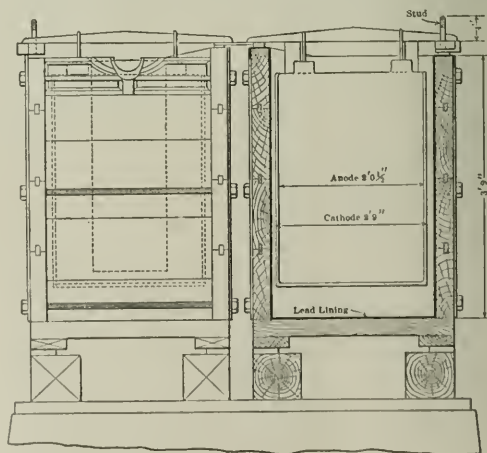


FIG. 2A—TANK DETAILS

tanks containing converter anodes the production per kilowatt-hour was slightly higher in the converter-anode tanks. Subsequent experiments have shown that the refined anodes admit of closer spacing than do converter anodes, without the same reduction in ampere efficiency, the effect of which is to decrease the resistance and increase the production per kilowatt. The difference, however, was not found to be great.

The advantages in favor of the refined anodes, as they appear

in Table I, are: First, higher grade of slime resulting in a saving in cost of slime refining; second, lower silver contents of cathodes, all of the silver in the cathode representing a dead loss; third, the lower percentage of anode scrap.

When reduced to dollars per ton of copper refined the sum of the savings that would result from the use of refined anodes in a plant of this type was found to be less than one-half the cost per ton of the reverberatory-furnace treatment of the copper. The cathodes produced from the two types of anodes differed but little in physical appearance.

TABLE I.—COMPARISON OF CONVERTER AND REFINED ANODES CAST IN THE SAME MOLDS

	Converter Anodes	Refined Anodes
Number of days covered by test	50	50
Number of refining tanks employed	48	48
Average analyses of anodes:		
Per cent. Cu	98.91	99.27
Per cent. As + Sb	0.072	0.071
Oz. Ag per ton	59.09	61.14
Oz. Au per ton	0.200	0.219
Average analyses of electrolyte:		
Specific gravity	1.20	1.20
Grams per liter Cu	43.5	43.5
Grams per liter free acid	160	160
Grams per liter As	11.97	11.97
Grams per liter Sb	0.49	0.49
Grams per liter Fe	10.09	10.09
Grams per liter Cl	0.045	0.045
Average temperature of electrolyte:		
Inlet of 8-tank cascade, °C	58	58
Outlet of 8-tank cascade, °C	54	54
Rate of circulation of electrolyte, gal. per min.	6	6
Number of anodes per tank	20	20
Number of cathodes per tank	20	20
Average weight per new anode, lb.	525	632
Average thickness per new anode, in.	3	3
Distance, center of anode to center of cathode, in.	2.87	2.87
Active cathode surface per tank, sq. ft.	252	252
Average amperes per tank	8387	8387
Average volts for 48 tanks	27.21	28.53
Average volts per tank	0.567	0.594
Average kilowatts for 48 tanks	228.2	239.3
Total copper deposited, lb.	1,103,749	1,148,749
Average age of cathodes drawn	2½	2½
Average ampere efficiency of deposit, per cent.	88.3	91.9
Average amperes per sq. ft. cathode surface	33.3	33.3
Average lb. copper deposited per kilowatt-hour	4.03	4.00
Average oz. per ton silver in cathodes	1.25	.93
Average per cent. As + Sb in cathodes	0.0043	0.0043
Average per cent. anode scrap	8.00	5.30
Analyses of silver slime:		
Per cent. Cu	40.3	18.80
Oz. Ag per ton	6755.00	14079
Oz. Au per ton	18.34	38.45

In a crane-operated electrolytic refinery, where the anodes and cathodes are handled in tank units and not individually as here, difficulties would result if converter anodes were used because of the fact that converter anodes corrode less uniformly than refined anodes. Many of the converter anodes would become scrap before the date set for drawing the scrap from a tank and an equal number of pieces would not be scrap at the time for drawing.

The above comparison, together with other tests made at this plant, has demonstrated that refined anodes, while very desirable, would for this plant be an expensive luxury.

Fig. 6 shows the earlier form of anode used in this plant and the Morrow clip type¹ now used. The advantage of the clip type of anode is that the amount of inactive copper above the solution line is much less than is the case with the lug type. The average percentage of anode scrap for the year 1912 was 5.0; the average weight of the converter anode is 500 lb. The

scrap resulting from each anode weighs about 30 lb., this is remelted in the converters. The clip type of anode also has the advantage, as compared with the lug type, of being suspended in such a manner as always to hang plumb in the tank. The standard converter anode measures 24.5 in. by 35.75 in.

The clip or hanger is made from a ¾-in. round copper rod 60 ft. long rolled in a rod mill from a bar cast at the refining furnace. This rod is cut and bent into the loop form, in an automatic machine, and placed in the anode mold. Both the anodes and cathodes are supported in the tanks by cast-copper rods as shown in sketch. The anodes are spaced 5.2 in. center to center. It will be noted in Fig. 6 that the anode is 3 in. thick at the top and 2.5 in. thick at the bottom. It is found that the anode corrodes more rapidly at the upper end where the current enters than at the lower end, that the wedge shape yields a lower percentage of scrap and that the anode retains its original shape for a longer time than is the case with an anode of uniform thickness.

Cathodes

The starting sheets are produced in 44 standard refining tanks divided into two circuits connected in parallel in the main circuit and are thus operating at one-half the current density of the commercial refining tanks. Each tank contains 21 cathode blanks and 22 anodes. The blanks are made of ½-in. rolled copper 27.5 in. x 39.5 in. below the solution line, as shown in Fig. 6.

The anodes used in these tanks are of the same type as the standard anode but larger, being 20.5 in. x 38½ in.

The 12-hour sheets are stripped at an average weight of 4 lb. each, measuring 26 in. x 36.5 in. after trimming. The men employed in stripping the plates do their work directly over the tanks, using a traveling bench suspended from the trolley track. This bench, on which the sheets are piled after stripping, is moved from tank to tank as the men advance. The blanks are covered with a thin coating of low-grade mineral oil to which 0.5 lb. of artificial graphite per gallon of oil is added. Grooved wooden strips are used on the edges of the plates to prevent deposition of copper. The strippers work independently and each man strips and delivers to the trimmer 360 sheets per eight-hour shift. The electrolyte used in these tanks differs slightly from that of the main tank room. An average analysis is given in Table II.

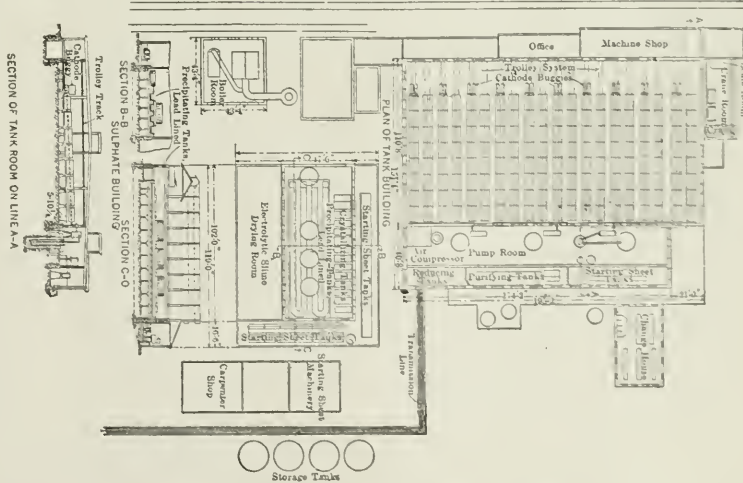


FIG. 3.—GREAT FALLS ELECTROLYTIC PLANT AS REMODELED

TABLE II.—STARTING-SHEET TANK ELECTROLYTE

Specific gravity	1.175
Free H ₂ SO ₄ , grams per liter	120.0
Cu, grams per liter	50.0
As, grams per liter	5.0
Sb, grams per liter	0.4
Fe, grams per liter	4.5
Cl, grams per liter	0.04

¹ U. S. patents No. 621,121 and 631,471.

Table III shows the effect of sulphuric acid in the electrolyte on the efficiency of deposition, the production per kilowatt-hour increasing as the acid increases. The sheets produced when the acid is about 120 g. per liter are found to be smoother and tougher than when acid is higher.

The starting sheets are trimmed on the four edges to prevent short circuits. Two clips or loops, cut from similar sheets, are then attached to each sheet by means of the Morrow clip machine² (see Fig. 6).

TABLE III.—EFFECT OF FREE SULPHURIC ACID ON STARTING-SHEET PRODUCTION

Converter anodes, 39.5 in. by 27.5 in. by 0.25 in. blanks, 3 in. anode; 2.5 in. center of anode to center of blank
Circulating electrolyte at 4 gal. per minute

Amperes	Amps. per Sq. Ft.	Efficiency	E.M.F. per Tank Volts	Lb. Cu. per Tank-hour	Average Weight per Sheet	ELECTROLYTE		
						Gal. per Liter Free H ₂ SO ₄	Cu	Temperature Co
4,580	16.2	86.5	0.475	4.75	5.86	74	42.6	52.5
4,490	15.9	86.3	0.462	4.84	5.78	74	42.6	52
4,453	15.7	85.2	0.462	4.80	5.61	70	40.1	55
4,450	15.7	83.6	0.475	4.55	5.50	69	39.8	49
4,390	15.3	90.3	0.482	4.84	5.87	69	40.1	50
4,403	15.6	90.2	0.496	4.71	5.87	73	38.4	47.5
4,290	15.2	88.2	0.462	4.94	5.60	80	38.9	51
4,397	15.5	91.6	0.470	5.04	5.93	82	38.8	52
4,538	16.0	90.3	0.476	4.90	6.06	88	38.8	51
4,461	15.8	91.9	0.450	5.28	6.06	89	39.9	52
4,408	15.6	92.5	0.426	5.63	6.03	110	40.4	52
4,506	16.0	92.5	0.452	5.29	6.17	109	38.9	53
4,496	15.9	91.7	0.424	5.53	6.10	116	40.6	53
4,512	16.0	90.3	0.426	5.50	6.02	126	40.5	50
4,550	16.1	89.7	0.420	5.53	6.04	128	40.4	48
4,430	15.7	90.7	0.400	5.86	5.94	128	42.0	48
4,435	15.7	84.7	0.385	6.09	5.56	126	42.4	50
4,420	15.6	87.3	0.365	6.09	5.62	126	42.4	50
4,445	15.7	91.8	0.357	6.23	6.03	124	43.6	48
4,453	15.7	85.1	0.360	6.13	5.60	132	42.1	46
4,404	15.6	87.0	0.342	6.55	5.66	148	40.8	49
4,450	15.7	88.6	0.350	6.62	5.82	149	42.3	46

Note.—The sheets produced with between 116 and 126 g. per liter of acid were the toughest and best.

The starting sheets are now ready for the tanks. It will be noted that the cathodes hang $\frac{3}{4}$ in. lower in the tank than the anode and are 1.5 in. wider than the anode. When using anodes and cathodes of the same size a cathode very rough on the edges resulted; increasing the size of the cathode has resulted in a cathode with comparatively smooth edges and an increase of 5 per cent in ampere efficiency.

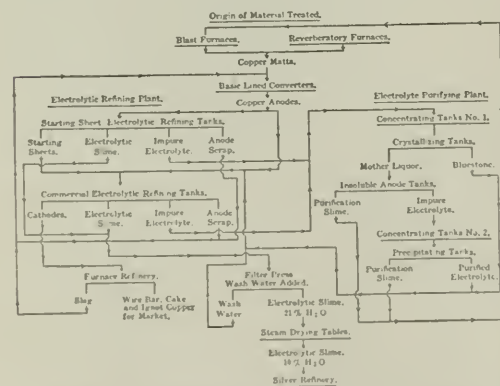


FIG. 4.—FLOW SHEET OF GREAT FALLS ELECTROLYTIC PLANT

When the output of the generators is 2000 kw or more the best results are obtained while drawing two and three-day cathodes. With a lower generator output older cathodes are drawn. The generator output is governed by the head on the water wheels.

¹ U. S. patent 600,498, Mar. 8, 1898.

To determine the most economical age of cathodes and number of electrodes per tank at which to operate, the labor, the production per kw-hour, the silver lost in cathodes and the impurities in the anodes and cathodes must be considered. Assuming the quality of the product as regards impurities to

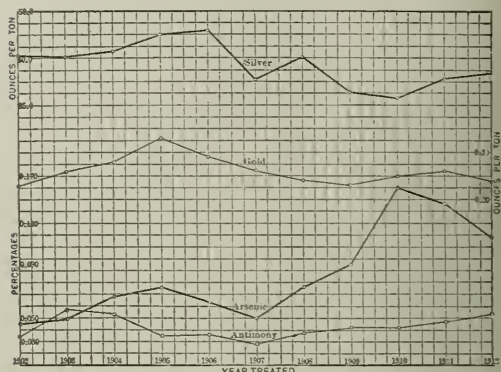


FIG. 5.—CONVERTER ANODES

remain satisfactory under the different conditions the cost per ton remains as the only consideration. Table IV is a sum-

TABLE IV.—EFFECT OF VARYING THE AGE OF CATHODES AND THE NUMBER OF ELECTRODES PER TANK

Age of Cathodes, Days	ELECTRODES PER TANK		Average Amperes	Average Amperes per Sq. Ft.	Ampere Efficiency, Per Cent.	Cu per Kilowatt-hour, Lb.	Cathode Cu, per Ton	Analyses As, Sb, Fe Per Cent.
	Anodes ^a	Cathodes						
4	20	20	9,300	36.9	88.0	3.93	1.32	0.0036
2	20	20	8,808	35.0	90.85	3.72	0.83	0.0030
3	20	20	8,877	35.2	89.00	3.75	0.83	0.0032
2	21	21	9,035	34.1	90.90	3.84	0.89	0.0032
2	21	21	9,223	34.8	89.40	3.87	1.02	0.0029
2	22	22	9,071	32.6	90.50	4.02	0.89	0.0033
3	22	22	9,167	33.0	88.80	4.07	0.95	0.0030

^a Converter anodes. Average analysis of anodes: Cu, 99.13; As, 0.127; Sb, 0.055 per cent.; Ag, 33.97 oz. per ton; Au, 0.22 oz.

mary of various comparisons made between different ages of cathodes with varying numbers of electrodes per tank.

Table V is the comparison, in detail, between 21 and 23 pairs of electrodes per tank while drawing two and three-day cathodes.

The Electrolyte

The 320 commercial refining tanks are divided into three sections, two sections containing 96 tanks each and the third section 128 tanks.

Each section is provided with a separate electrolyte and circulating system. Copper and acid determinations are made daily on samples of each electrolyte. As, Sb, Fe and Cl determinations are made weekly.

The two groups of starting-sheet tanks each have a separate electrolyte which is sampled and assayed in the same manner as the commercial tanks electrolyte.

Some analyses of the electrolyte in the commercial refining tanks are shown in Fig. 5. The analysis of solutions is ordinarily reported in terms of grams per liter. With a current density of 34 amp per square foot and a circulation of 6 gal. of electrolyte per minute the best results, while treating converter anodes, are obtained with an electrolyte carrying 40 g. per liter of copper and 160 g. per liter of free sulphuric acid.

The circulation of the electrolyte is through a cascade of eight tanks as already described. In the pump room the electrolyte is delivered to the receiving tank, a round lead-lined tank 11 ft. in diameter and 4.5 ft. deep (see Fig. 3). In this

TABLE V.—EFFECT OF VARYING THE AGE OF CATHODES AND NUMBER OF ELECTRODES PER TANK

Details of 21 and 22 pairs of electrodes per tank in Table V.
Standard Converter Anodes

	21 Anodes and 21 Cathodes per Tank	22 Anodes and 22 Cathodes per Tank
Number of refining tanks used	160	160
Copper deposited, 2-day cathodes, lb.	1,778,352	1,704,743
Copper deposited, 3-day cathodes, lb.	1,742,678	1,889,401
Total copper deposited, lb.	3,521,030	3,594,144
Spacing of electrodes, anode centers, in.	5.45	5.20
Average amperes for 2-day cathodes	9,035	9,071
Average amperes for 3-day cathodes	9,223	9,167
Average amperes per sq. ft., 2-day cathodes	34.1	32.6
Average amperes per sq. ft., 3-day cathodes	34.8	33.0
Average ampere efficiency, 2-day cathodes, per cent	90.9	90.5
Average ampere efficiency, 3-day cathodes, per cent	89.4	88.8
Average drop per tank in volts, 2-day cathodes	0.612	0.581
Average drop per tank in volts, 3-day cathodes	0.600	0.562
Average copper deposited per kilowatt-hour, 2-day cathodes, lb.	3.837	4.021
Average copper deposited per kilowatt-hour, 3-day cathodes, lb.	3.871	4.075
Average copper deposited per kilowatt-hour, 2- and 3-day cathodes, lb.	3.854	4.048
Analyses of cathodes:		
2-day cathodes, As + Sb, per cent	0.0032	0.0033
3-day cathodes, As + Sb, per cent	0.0029	0.0030
2-day cathodes, Ag, oz. per ton	0.89	0.89
3-day cathodes, Ag, oz. per ton	1.02	0.95
Same both set of Tanks		
Analyses of Electrolyte:		
Specific gravity	1.219	
Cu, grams per liter	41.1	
Free acid, grams per liter	133	
As, grams per liter	6.49	
Sb, grams per liter	0.51	
Fe, grams per liter	5.11	
Cl, grams per liter	0.44	
Average temperature of electrolyte (8 tanks in cascade):		
Inlet tank, C°	57	
Outlet tank, C°	53	
Speed of circulation of electrolyte, gal. per min.	6	
Analyses of anodes (converter):		
Cu, per cent	99.13	
As, per cent	0.127	
Sb, per cent	0.055	
Ag, oz. per ton	33.97	
Au, oz. per ton	0.22	

tank are 150 ft. of 1-in. 8-lb. antimonial lead pipe; live steam is supplied to these coils at a pressure of 65 lb. per square inch, by means of which the electrolyte is heated. From the receiving tank the electrolyte is raised to the discharge tank from which it is again conducted to the head tank of the cascade.

Various means of elevating the electrolyte have been employed in this plant. The first method used was the steam injector constructed of lead and antimony. These pumps were of limited capacity and the dilution of the solution was excessive. They were followed by a system of lead-lined cast-iron eggs in which the electrolyte was collected and blown to the overhead tank with compressed air. This system was very inefficient and gave way to a type of boiler-fed pump, the water end of which was of bronze. These soon became leaky from corrosion and were abandoned. A type of plunger pump with a lead-lined barrel and rubber valve and plunger was next tried. This pump was continued in service for several years with satisfactory results but was replaced 10 years ago by the air lift, which has continued to hold the field. The air lift as used here is constructed of cast lead-antimony pipe 6 in. inside diameter, 34-in. walls. The pipe is cast in 4-ft. lengths with flanged joints. A brick wall 20 ft. deep and 4 ft. in diameter, made waterproof, with a cement lining, contains that portion of the lift that is below the floor level. Working against a head of 14 ft. 8 in., 160 gal. of electrolyte (1.22 specific gravity) is raised per minute with a consumption of 80 cu. ft. of free air compressed to 16 lb. per square inch (see Fig. 7).

The lift, while it is of low mechanical efficiency and sensitive to variations in the air pressure, is very satisfactory in its operation, having no moving parts and requiring but little attention. The loss in temperature of the electrolyte during its passage through the air lift from the receiving tank to the discharge tank averages 0.5° C.

TABLE VI—EFFECT OF SPEED OF CIRCULATION OF ELECTROLYTE ON THE CONCENTRATION OF THE SOLUTION

Current density 36.0 amperes per square foot, 4-day cathodes, converter anodes. Circulating Electrolyte at Rate of 6 gal. per minute

Zone	Specific Gravity	GRAMS PER LITER						Copper per Kilowatt-hour Lb.
		Acid	Cu	As	Sb	Fe	Cl	
1	1.213	174	38.4	7.5	0.56	2.95	0.044	
2	1.216	172	39.2	7.2	0.59	2.95	0.045	
3	1.216	171	40.2	7.1	0.59	2.95	0.043	
4	1.228	153	49.9	7.2	0.59	2.90	0.044	
Average	1.218	168	41.9	7.3	0.58	2.94	0.044	3.92

Circulating Electrolyte at the Rate of 4 gal. per minute								
1	1.211	163	40.0	7.3	0.52	2.90	0.042	
2	1.214	161	41.1	7.3	0.55	2.92	0.040	
3	1.216	159	42.0	7.4	0.58	2.95	0.040	
4	1.263	139	67.8	7.4	0.55	2.90	0.040	
Average	1.226	156	47.7	7.4	0.55	2.92	0.040	3.85

Circulating Electrolyte at rate of 2 gal. per minute								
1	1.210	167	37.8	7.6	0.52	2.90	0.042	
2	1.215	165	39.0	7.1	0.49	2.95	0.042	
3	1.217	162	41.0	7.6	0.51	2.95	0.042	
4	1.265	146	66.4	7.9	0.55	2.95	0.041	
Average	1.226	160	46.1	7.6	0.52	2.94	0.042	

Sampled after circulation had been entirely shut off for 7 hr. with current passing at the rate of 36 amperes per square foot

1	1.185	179	23.4					
2	1.210	164	38.0					
3	1.230	151	51.8					
4	1.255	138	65.5					
Average	1.220	158	44.7					

The compressed air for the operation of the lift is obtained from the converter-plant blowing engines which are operated by water power. A motor-driven compressor at the tank house supplies air in cases of emergency.

Circulation of the electrolyte is maintained at a speed of 6 gal. per minute, a lower circulation reducing the production per kw-hour while a higher rate has the effect of disturbing too much the settlement of the slime and increasing the silver in cathodes. Table VI shows the effect on the concentration of the electrolyte of varying the speed of circulation. The samples which were taken in the regular refining tank and marked zones 1, 2, 3 and 4, were obtained in the following manner: A glass tube, closed at the top, was lowered to a point 6 in. below the surface of the electrolyte and a sample collected at this point and called zone 1. Zone 2 sample was taken in the same manner at a point 17 in. below the surface; No. 3, 28 in. below the surface and No. 4, 30 in. below the surface. The column of electrolyte in the tanks was 40 in. high, there being 3 in. of slime in the tanks. These figures represent the averages of 104 samples taken during seven consecutive days from 48 refining tanks at 36 amp per square foot. The samples were all taken between the center pair of electrodes.

These figures show a rather remarkable concentration of the electrolyte as regards copper and acid with negative results regarding impurities in solution and demonstrated that, while operating at 36 amp per square foot, circulating the electrolyte at the rate of 6 gal. per minute delivering the solution at the surface of the tank and discharging from a point near the bottom, fails to maintain the electrolyte in a homogeneous state but is found to be the most economical speed at which to operate.

The arsenic and antimony are the impurities with which we are chiefly concerned in our refining operations, and as their effects on the conductivity of the copper are much alike our practice is to report them combined as arsenic plus antimony in the cathodes, making a separation on the average monthly sample to note the relations. These impurities in the electrolyte and in the anodes are determined separately.

There is no silver in solution in the electrolyte and all of the

silver present in the cathodes is deposited mechanically from the slime in suspension; the slime is entrapped on the rough surfaces of the cathodes, the silver contents increasing with the age and roughness of the cathode. The arsenic and antimony are deposited both mechanically from the slime and electrolytically from the electrolyte.

When refining converter anodes carrying 0.12 per cent As, at 34 amp per square foot the arsenic in the electrolyte is not allowed to exceed 7 g. per liter.

The results of an experiment with covered cathodes and converter anodes may be of interest. Three regular cathode starting sheets were placed in separate frames covered with 8-oz. duck in such a manner as to entirely inclose the sheets. These were substituted for three regular cathodes in a commercial refining tank and after remaining 2.75 days they were removed, sampled and assayed as shown in Table VII.

TABLE VII.—ANALYSIS OF ELECTROLYTE

Specific Gravity	GRAMS PER LITER					
	Acid	Cu	As	Sb	Fe	Cl
1.200	152	41.2	7.0	0.44	1.80	0.04

ANALYSIS OF ANODES

Per Cent. Cu	Oz. Ag per Ton	Oz. Au per Ton	Per Cent. As	Per Cent. Sb
99.15	39.0	0.25	0.211	0.04

CATHODES PRODUCED

	Inclosed Cathode	Regular Cathode
Current density, amperes per sq. ft.	31.6	35.7
Age of cathodes, days	2.75	4.0
Ag, oz. per ton	0.2	1.2
As+Sb, per cent	0.0036	0.0042

The lower circuit density of the covered cathodes is due to the increased resistance offered by the covering.

The canvas-covered cathodes were drawn at 2.75 days rather than four days as the canvas was showing signs of failure. The results show that the canvas covering was effective in preventing but little of the silver-bearing slime in suspension from coming in contact with the cathodes but that it had a lesser effect on the arsenic and antimony contents of the deposited copper.

The original method employed at these works of purifying the electrolyte was the manufacture of bluestone. An amount of electrolyte sufficient to maintain the working solutions at the required degree of purity was daily run off from the tank house to the sulphate plant. Here it was heated to the boiling point by means of steam coils and passed through open tanks containing shot copper into which compressed air was blown until all of the free acid was neutralized. After the necessary concentration by boiling the solution was allowed to cool in an open chute where most of the copper crystallized out. The crystals from the chute were redissolved in water, concentrated, and run into the final crystallizing tanks. The crystals from these tanks were washed, dried and packed in barrels for shipment. The mother liquor, after it had become too foul for further use, was run over scrap iron for the recovery of the small amount of copper remaining in solution. After passing over the iron the solution, which contained only the impurities, was wasted.

This was an efficient method of freeing the electrolyte of impurities and would probably still be employed had not the market for the bluestone failed during 1899. In 1897 there were 2,300,000 lb. of bluestone produced and shipped from our sulphate plant.

The electrolytic method was next employed for purifying the electrolyte. This process consisted in removing from the working solutions each day such an amount of electrolyte as was necessary to maintain the purity of the electrolyte. This solution was passed through four standard refining tanks containing lead anodes and copper or lead cathodes. These tanks were arranged in a cascade the electrolyte flowing from one tank to another. The Cu, As and Sb contents of the solution, as it left the last tank, depended on the speed of flow. With a circulation of 2 liters per minute but a trace of Cu, As and Sb remained. The ampere efficiency of deposit, however, was extremely low at this rate of flow.

Table VIII shows the results obtained while circulating at 4 liters per minute. The percentages of elimination as shown

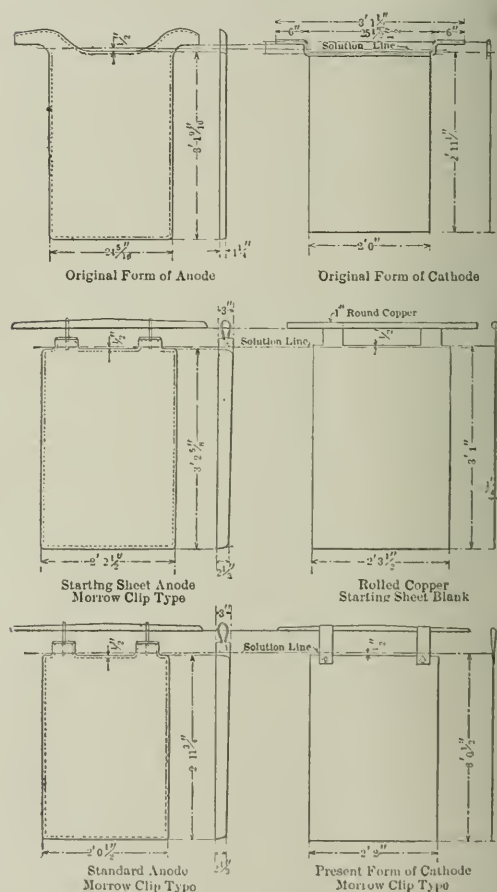


FIG. 6.—ANODES, CATHODES, AND STARTING SHEET BLANK

in this table are high while the ampere efficiency is very low, the two lower tanks in the cascade doing but little work. In subsequent experiments the most economical speed of flow was found to be 7 liters per minute with an elimination of 99 per cent of the copper, 78 per cent of the arsenic and 91.1 per cent of the antimony with a total ampere efficiency of about 50 per cent.

The increase in the iron contents of the electrolyte between the inlet of the first and outlet of the last tank, as shown in Table VIII, is due to the concentration of the electrolyte by reason of the heat evolved by the electrolysis. These analyses are corrected on the basis of an unchanged volume of electrolyte in Table IX.

TABLE VIII.—REMOVAL OF COPPER, ARSENIC AND ANTIMONY FROM ELECTROLYTE IN INSOLUBLE ANODE TANKS

Circulation, four liters per minute. Lead anodes, copper cathodes, 9,000 amperes, 31.8 amperes per square foot

	GRAMS PER LITER					Volts per Tank	Temperature, C°
	Acid	Cu	Fe	As	Sb		
Inlet tank No. 1.	144	37.060	6.242	3.200	0.463		17
Outlet tank No. 1.	184	7.376	6.813	2.240	0.260	2.22	42
Outlet tank No. 2.	194	0.504	7.364	0.400	0.061	2.25	57
Outlet tank No. 3.	208	0.088	7.701	0.056	0.038	2.25	64
Outlet tank No. 4.	216	0.048	7.915	0.028	0.028	2.25	65

TABLE IX.—CORRECTED ANALYSES

	GRAMS PER LITER					PERCENTAGE ELIMINATION OF ORIGINAL AMOUNTS			Ampere Efficiency Per Cent.
	Acid	Cu	Fe	As	Sb	Cu	As	Sb	
Inlet tank No. 1.	144	37.060	6.242	3.200	0.4630		35.9	48.7	71.70
Outlet tank No. 1.	165	6.760	6.242	2.350	0.2380	81.8	53.3	40.2	19.50
Outlet tank No. 2.	165	0.427	6.242	0.339	0.0517	17.1	53.3	40.2	19.50
Outlet tank No. 3.	169	0.071	6.242	0.045	0.0308	0.9	9.2	4.7	1.68
Outlet tank No. 4.	170	0.038	6.242	0.022	0.0220	0.1	0.7	1.7	0.15
Totals and averages.....						99.9	99.3	95.3	23.26

The process thus far described has removed only the Cu, As and Sb from the solution, leaving the Fe, Ni, Bi and Zn. To remove these the electrolyte, as it leaves the insoluble-anode tanks, is run into a lead-lined tank 13 ft. in diameter and 4.5 ft. deep lined with 12-lb. chemical sheet lead and containing 600 ft. of 1-in., 8-lb. chemical-lead pipe. In this tank it is concentrated to 55° B_e, decanted to an open tank 10 ft. long, 4 ft.

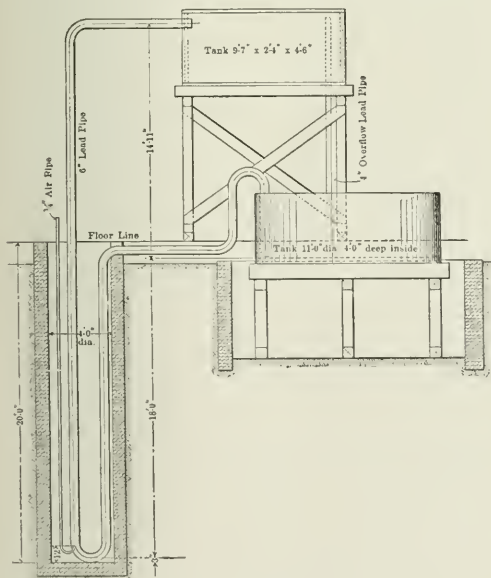


FIG. 7.—ELEVATION OF SOLUTION AIR-LIFT PUMP

wide and 3 ft. deep, where it is allowed to stand for four days, during which time the salts of Fe, Ni, Bi and Zn crystallize out, leaving a solution of approximately the following composition to be returned to the tank room: H₂SO₄, 1100; As, 1; Sb, 0.2; Fe, 1; Ni, 5.3, and Zn, 1.5 g. per liter.

The Cu, As and Sb deposited in the insoluble-anode tanks are in the form of a black slime, some of the slime adhering to the cathode but the greater portion settling to the bottom of the tanks. An average assay of this slime is given in Table X.

TABLE X.—SLIME FROM INSOLUBLE-ANODE TANKS

(Treating electrolyte direct from tank room)

Moisture, per cent.		9.66
Cu, per cent.		46.30
SiO ₂ , per cent.		0.38
FeO, per cent.		1.00
Al ₂ O ₃ , per cent.		0.4
CaO, per cent.		1.08
S, per cent.		5.02
As, per cent.		21.48
Sb, per cent.		2.28
Ni, per cent.		0.35
Zn, per cent.		0.32
Ag, oz. per ton.		3.01
Au, oz. per ton.		0.03

This material was sent to the blast-furnace plant for treatment.

The method of purifying the electrolyte now in use is a modification of the one just described and consists in running off daily from the tank house about 25,000 liters of solution, concentrating this, by boiling, to 48° B_e. From the boiling tank the concentrated solution is run to crystallizing tanks and allowed to stand for four days. At the end of this period 82

TABLE XI.—ANALYSIS OF INSOLUBLE-ANODE TANK SLIME (Treating mother liquor from crystallizing tanks)

Moisture, per cent.		9.66
Cu, per cent.		46.30
SiO ₂ , per cent.		0.38
FeO, per cent.		1.00
Al ₂ O ₃ , per cent.		0.4
CaO, per cent.		1.08
S, per cent.		5.02
As, per cent.		21.48
Sb, per cent.		2.28
Ni, per cent.		0.35
Zn, per cent.		0.32
Ag, oz. per ton.		3.01
Au, oz. per ton.		0.03

per cent of the copper will have crystallized out. The mother liquor has the following analysis: Acid, 475; Cu, 17.4; As, 20.2; Sb, 1.1, and Fe, 15.2 g. per liter. This is run to the insoluble-anode tanks where it is treated for the removal of Cu, As and Sb as described above.

Table XI gives an analysis of the slime obtained when using this method.

After treatment in the insoluble-anode tanks the solution is either returned to the tank room or treated, by further concentration and crystallization as described above, for the removal of the Fe, Ni and Zn.

The analysis of slime from the second concentration and precipitation for the removal of Fe, Ni and Zn is as follows: Moisture, 22.1; S, 16.6; Ni, 5.6; Cu, 1.3; Fe, 10.3; Zn, 2.7 per cent.

The advantage of crystallizing the copper from the solution before treatment in the insoluble-anode tanks is that the ampere efficiency of these tanks is greatly increased because of the concentrated solution treated and that the amount of slime produced is reduced by nearly one-half, as shown in Table XII

TABLE XII.—PRODUCTION OF INSOLUBLE ANODE TANK SLIME

	Wet Slime per Month Lb	As Per Cent
Treating electrolyte direct from tank room	38,333	10.50
Removing 82 per cent of copper before treating	21,600	21.48

The slime from the insoluble-anode tanks is sent to the blast furnaces for treatment, the iron slime is washed for the removal of free acid and sent to the blast furnace or wasted if free of copper.

The arsenic liberated from the anodes in the refining tanks finds its way into the following products in the proportions shown: In cathodes, 1.22; in silver slimes, 17.0, and in purification slimes and cathodes, 81.59 per cent. It is therefore necessary to remove from the electrolyte 81.59 per cent of the arsenic liberated at the anode.

Electrolytic Slime

From a converter anode carrying 40 oz. silver and 0.24 oz. gold per ton, a slime carrying 43.3 per cent Cu, 5000 oz. silver and 34 oz. gold per ton is obtained. See Table I for complete analysis.

This slime is removed from the refining tanks once in 60 days in the following manner: A copper jumper, held by the iron studs provided, connects the two ends of tank bars on adjacent tanks and short circuits four tanks, as shown in Fig. 2. The anodes and cathodes are removed and the slime, 5.5 in. having accumulated, is diluted with water and pumped by means of a bronze steam injector to a round lead-lined tank, 12 ft. in diameter, 4 ft. deep, situated 200 ft. distant in the slime-sampling room. Before entering this tank the slime passes through a lead screen with $\frac{1}{8}$ -in. holes, and $\frac{1}{2}$ -in. centers. In this tank the slime is washed with hot water to remove the free acid and soluble copper.

From this tank the slime is dropped into a cast-copper, lead-lined egg from which it is forced by compressed air at 100 lb. pressure into a modified Bushnell filter press containing hard-copper plates and rings, 8-oz. duck being used for filter cloth. Thirteen cakes 26 in. in diameter and 1.25 in. thick are made at each charge; the wet weight is 865 lb. per charge at 21 per cent moisture. From the filter press the slime is placed on steam-jacketed copper-drying tables where the moisture is re-

carriage holding 66 sheets. The solution dripping from the cathodes is held in these carriages to be later drawn off through an outlet in the bottom. When the carriage is loaded it is moved by hand to the scales, situated as shown in Fig. 3. After weighing the cathodes are lifted from the carriage by the traveling crane. The ends of the angle irons on which the cathode rods rest in the carriage are punched to receive the crane chain hooks by means of which the crane raises the load. After the cathodes are removed the carriage is weighed to obtain the tare. The crane then raises the cathodes and loads them on to a 4-wheel truck for delivery to the furnaces.

After the cathodes are drawn the tank men inspect the anodes, marking such as are ready to be removed as scrap. The same carriages as were used for the cathodes are now spotted under the trolley track at the end of the tanks and the anode scrap is deposited in them. When all of the scrap has been removed in this manner the carriages are moved to the scales and weighed. The crane then lifts the scrap from the carriages, by means of the angle irons, and conveys it to a tank where the slime is washed from the surfaces with a stream of water. The scrap is then placed on trucks, by the crane, from which it is loaded into box cars for shipment to the converters.

After the scrap has been all drawn from the tanks and the carriages removed the tank men proceed to fill the vacancies, thus made in the tanks, with new anodes, one anode at a time being handled.

As but four cathodes are removed from one tank at a time it is not necessary to cut the tanks out of circuit while drawing copper.

As compared with the traveling-crane system of handling the anodes and cathodes in tank units, the hand chain-block system of handling, as employed here, admittedly required more labor but is not without its advantages, which are as follows:

First, the chain-block system permits the use of converter anodes.

Second, the chain-block system, with aisles between the refining tanks, permits of starting sheets being straightened and placed in the tanks in such a manner that subsequent removal and straightening is not necessary.

Third, respacing of anodes and cathodes, from time to time, as the anodes corrode, to reduce the resistance of the tank, is unnecessary as the tank men properly space each separate anode and starting sheet as placed in the tank.

Fourth, there is a lower percentage of scrap and no scrap to be returned to the tanks.

Cash prizes are awarded to the tank men on the divisions making the best and second best records for the month. The division producing the highest tonnage of cathodes, after deducting the weight of the anode scrap, is considered as having the best record. This plan has been found very effective in stimulating rivalry between the different divisions and thereby improving the efficiency of operations.

Four insoluble-anode tanks to take care of the increase of copper in solution are provided. These tanks, as shown in Fig. 3, are situated in a separate room south of the pump house and, by means of a system of circulating pipes, can be run on any of the three electrolytes. It is not usually found necessary to operate more than two of these tanks at a time.

Four-day cathodes at 34 amp per square foot are produced in these tanks. The product of these insoluble-anode tanks is of the same purity as the product of the soluble-anode tanks provided the rate of flow of the electrolyte is sufficiently high to insure ample copper in solution in the lower tanks. When the rate of flow is greatly reduced or altogether stopped the electrolyte soon becomes impoverished in copper and arsenic is deposited out of the solution with the copper. The circulation is maintained in these tanks at the rate of 10 gal. per minute.

Table XIII is a summary of the average amount of copper, silver and gold locked up in the process while drawing two-day

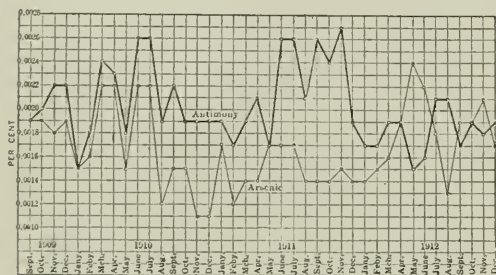


FIG. 8.—RELATION OF ARSENIC TO ANTIMONY IN WIRE BAR

duced to 10 per cent. The slime is then crushed with a roller on a cement floor and sampled in 22,000-lb. lots for shipment.

The slime is sampled in the following manner. The 22,000-lb. lot is shoveled into a conical pile, each shovelful being delivered upon the apex of the cone. To insure thorough mixing of the material this operation is repeated twice. The material is then split-shoveled four times; these operations reduce the sample to about 300 lb. From this point the material is quartered down with repeated crushing to about 130 oz., the laboratory sample.

The slime is then packed in 10-oz. canvas sacks holding 140 lb. each. Four of these sacks are packed in paper-lined wooden boxes in which shape the slime is shipped to an Eastern refinery for treatment. About 560,000 lb. of slime are produced and shipped per year.

Operating Details

The main tank room is divided into 10 divisions of 32 tanks each. Each division is taken care of by two tank men who are held responsible for these tanks. One inspector, locating and removing short circuits between anodes and cathodes, is employed for each two divisions. The inspector's work is done on night shift.

Sixteen tanks of two-day cathodes, per division, or one-half of the tanks, are drawn daily, each tank containing 22 anodes and 22 cathodes. The day's starting sheets having been delivered to the different divisions by the night shift, the tank men begin drawing cathodes in the morning. Four cathodes are drawn at a time with a 1-ton Yale & Towne triplex chain block and multiple hook. The cathodes after being replaced with starting sheets are deposited in lead-lined wheel carriages, each

cathodes and refining at the rate of 174,000 lb. copper per day.

To arrive at the weight of copper in the refining tanks, for inventory purposes on the first of the year, a spring balance is attached to the chain-block hook and the anodes and cathodes are raised clear of their supports and weighed in solution. A factor, previously determined, representing the difference in

TABLE XIII.—METALS LOCKED UP IN PROCESS

	Copper, Lb.	Silver, Oz.	Gold, Oz.
In anodes.....	2,300,000	44,000	316
In slime.....	22,300	140,000	850
In cathodes.....	180,000		
In solutions.....	95,000		
Totals.....	2,597,300	184,000	1,166

weight of the average electrode, weighed in solution in this manner, and its actual weight, is applied to the weights thus obtained. Weights obtained in this manner are found to check actual weights very closely.

The amount of slime in the bottom of the refining tanks at any time is calculated from the tank cleaning record, the number of days the slime has been accumulating and the amount of slime that is liberated at the anode per tank per day per ampere being known.

Current readings are taken and recorded hourly from a 10,000-amp Weston station ammeter the shunt of which is situated in the center of the tank house. At the power house on each generator-switchboard panel is a 5000-amp instrument of the same type. The averages of the hourly readings from these instruments are compared daily with the readings of the tank-house instruments to detect possible current leakage.

A system of pressure wires connects the refining tanks and the generators with a voltmeter switchboard situated in the office at the electrolytic plant. Hourly readings of the voltages at generators, at the tank house and at the different divisions and subdivisions of refining tanks are taken and recorded. In this manner abnormal conditions in any group of refining tanks or any change in resistance of the transmission line are promptly made known.

The daily report contains, on one sheet, the cathode production for the day and to date for the month, the number and weight of starting sheets used, the pounds of copper deposited per tank per day per ampere in the different sections, the percentage ampere efficiency of the total production, the amount and percentage of anode scrap for the day and to date for the month, the average amperes, volts and kilowatts for the day and for the age of the cathodes drawn, the average weight per cathode drawn, the pounds of anodes, cathodes and anode scrap on hand, the pay roll for the day and a digest of the electrolyte men's reports showing amount of solution treated in the purifying plant, character and speed of circulation of electrolyte and other minor data.

The present practice is to draw three-day cathodes on Mondays and Tuesdays, two-day cathodes on the four days following and none on Sundays, the generators running continuously during the seven days.

Under the head of 22 anodes and 22 cathodes per tank in Table V, the results obtained are shown in detail.

The average daily production of refined copper is 174,000 lb.

Furnace Refining

The refining-furnace plant, to be operated in connection with the electrolytic refinery, began producing anodes from converter copper in October, 1892.

The original installation consisted of two coal-fired reverberatory furnaces. These furnaces were identical in design, having a hearth 15 ft. 6 in. x 10 ft. 6 in. with a firebox 4 ft. x 4 ft. Each furnace was provided with a separate brick stack 65 ft. high, 28 in. x 28 in. inside. The capacity of these furnaces was 30,000 lb. of copper each.

The furnace building was 80 ft. x 128 ft. steel frame covered with corrugated iron.

During 1893 and 1894 two additional furnaces of the same type and capacity were added. Two of the furnaces were employed in producing anodes from converter pig and anode scrap from the electrolytic plant and two were used to make wire bar, cake and ingot from the cathodes.

All of the furnaces were dipped by hand, 9-in. ladles being used. The copper was cast in iron molds, the electrolytic copper being dumped in boshes containing water. The anodes were removed from the molds, by means of hooks, and allowed to cool in the air.

Later these four furnaces were replaced by two larger furnaces of an estimated capacity of 50,000 lb. of copper each, subsequently increased to 100,000 lb. each. The hearths of these furnaces are 14 ft. x 24 ft. and the fire boxes are 7 ft. long by 8 ft. wide. One of these furnaces is connected to one of the original brick stacks, described above, and the other has a new brick stack 74 ft. high with a 34-in. x 34-in. flue.

These two furnaces with a combined capacity of 200,000 lb. per day now handle the cathode output of the electrolytic refinery, the anodes coming direct from the converters.

The product of the refining furnaces is dipped into copper molds, made at the furnaces. These molds are hung in trunnions over cast-iron boshes through which cold water is circulated. The mold is capsized, as soon as the metal is set, and the copper drops into the cooling water. Cooling in this manner prevents the formation of cupric oxide on the surface of the castings.

The copper is conveyed from the furnaces to the molds by means of large ladles suspended by chains from a trolley running on a 34-in. x 4-in. track suspended over the boshes. Hand-forged Welsh ladles of the following sizes and capacities are used: 14-in., capacity, 200 lb. of copper; 16-in., 250 lb. of copper, and 19-in., 350 lb. of copper. The size of the ladle used is governed by the weight of the casting to be made.

The principal forms into which the copper is cast are as follows: Ingot, ingot bar, cake, wire bar and a special form of round billet used in the manufacture of seamless copper tubing. The copper is shipped direct to the consumer. A 100,000-lb. charge is taken out of each furnace once in 24 hours.

The usual practice of rabbling and poling the copper, for the oxidation of impurities and bringing of the metal to the proper pitch for casting, is carried on. The rabbling is effected by introducing compressed air at a pressure of 16 lb. per square inch into the molten bath by means of two 34-in. iron pipes, the end of the pipes being kept below the surface of the metal. About two hours is required for this operation. The reducing action is obtained by forcing green pine poles into the bath and covering the surface of the metal with charcoal. About 28 poles, 8 in. at the butt and 4 in. at the top, and 35 bushels of charcoal are required per charge of 50 tons of copper.

The condition of the bath, as regards oxygen contents, is noted from time to time by the refiner as he examines the successive button samples taken from the bath in a small ladle. When the physical appearance of the button indicates that the "tough-pitch" stage has been reached the poles are removed and the dipping operation is begun.

As the dipping proceeds the refiner observes the set of the castings made and adds charcoal or logs of wood to maintain the oxygen at the proper point.

The skimming of the furnace takes place just before the rabbling or oxidizing is started. The slag obtained is equivalent in weight to about 3.5 per cent of the upper charged, varying with the amount of lime used. The following is the average analysis of slag produced at furnace No. 2 producing wire bar: Cu, 62.89; SiO₂, 19.1; FeO, 2.0; Al₂O₃, 1.8; CaO, 7.2; MgO, 1.4; S, 0.17; As, 0.02; Sb, 0.025 per cent; Ag, 0.350 oz per ton.

The fuel used in these furnaces is known as Lochray lump coal, a bituminous coal mined at Tracy, Mont., 15 miles from these works. The analysis is as follows: Moisture, total, 7.01; moisture, combined, 1.7; volatile matter, 24.7; fixed carbon, 48.0; ash, 18.5; sulphur, 3.5 per cent; B.t.u. per pound of dry coal, 10,803.

It will be noted that this coal is high in sulphur. To protect the copper as it melts from the sulphur dioxide gases, resulting from the combustion of the fuel, about 30 per cent of the cathodes going to make up wire bar charges are dipped in milk of lime before charging. As the copper melts the lime forms a protective covering for the metal, hindering in a large measure the absorption of sulphur by the molten copper. This whitewashing process accounts for the lime and magnesia in the furnace slag.

The average analysis of wire bar produced at these furnaces is shown in Fig. 5, and the relation of arsenic to antimony in wire bar is shown in Fig. 8. The analysis of cake, ingot and other furnace products, in which high electrical conductivity is not essential, will contain 0.02 to 0.05 per cent less copper, due to the increase in oxygen contents.

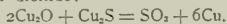
Copper Deoxidation by Means of Magnesium¹

By Dr. Friedrich Hüser

Copper has a great affinity to oxygen, and a still greater one to sulphur. The affinity increases with increase of temperature. Copper absorbs chiefly oxygen from the furnace gases, but also some sulphur dioxide² and hydrogen, while nitrogen, carbon dioxide and carbon monoxide³ are not absorbed. The three first-named gases dissolve in liquid copper during the oxidizing smelting process.

When the liquid copper solidifies, part of the hydrogen forms a solid solution, while oxygen and sulphur separate as cuprous oxide and cuprous sulphide.⁴

Since at a high temperature the affinity between sulphur and copper is smaller than between sulphur and oxygen, there will be a reaction between cuprous oxide and cuprous sulphide, liberating sulphur dioxide:



Refined copper is supposed to contain only traces of sulphur. Nevertheless the evolution of sulphur dioxide may become troublesome when copper solidifies, since cuprous oxide and cuprous sulphide form eutectics with copper and finally solidify as such. The solidification of the pure copper begins from the wall of the mould so that the eutectics are forced to the center, forming there an enriched portion. A more vigorous reaction sets in, which is shown by the rising of the metal. The copper becomes porous. Naturally particles of these eutectics remain stuck between the growing copper crystals, as shown in Figs. 1 to 3 and 5.

When poling copper the sulphur dioxide dissolved in it is mechanically removed and the cuprous oxide is reduced, with the exception of a small amount.⁵ Schnabel⁶ says: "The porosity of copper may also be avoided in other ways than by means of poling, by reducing the cuprous oxide by means of substances which do not develop gases during reduction. Phosphorus, phosphor-copper and manganese-copper are such substances."

It may be said in general: If to melted copper additions are made of other more easily oxidizable elements like Sn, Zn, Ni, Fe, Mn, P, Si, Al, Mg, the whole of the cuprous oxide present is theoretically reduced with formation of the oxide of the other metal. However, on account of the great dilution in the solvent (copper in this case) an excess of the reducing agent is necessary. This excess must be the greater the smaller the tendency of the reducing agent to form oxides. If this excess remains in the copper it may prove deleterious.

In the series of metals mentioned above the affinity to oxygen increases from the left to the right.

Magnesium is the metal furthest to the right. It has the strongest affinity for oxygen. Small additions of it suffice to reduce any cuprous oxide remaining in molten copper. The

addition of $\frac{1}{8}$ per cent., which has been mentioned (Electrician, vol. 59, 1907, p. 463), is arbitrary; the amount of magnesium required depends, of course, in each individual case on the content of cuprous oxide which is to be reduced.

I have made a number of melts in a crucible, which showed that copper containing oxygen can always be refined by means of magnesium. The starting product used in these experiments was electrolytic copper of great purity. It was free from tin, iron, lead, arsenic, sulphur, and contained 0.01 per cent. zinc, a trace of nickel and 0.0237 per cent. of oxygen. Six to ten kilograms of this copper were melted for each experiment in a graphite crucible, in a coal-fired furnace working with forced draught.

The required amount of magnesium varied with the speed of melting and also with the length of time for which the melted copper remained in the furnace after the melt was over.

If the melting took place very rapidly under a charcoal cover, a few hundredths of a per cent. of magnesium were sufficient. An alloy of magnesium and copper, containing 20 per cent. of magnesium, is preferable to more magnesium for these experiments. Its use prevents a too violent reaction and losses in manipulation. This was of some importance, on account of the small quantities that could be melted in the crucible. This alloy is brittle and may be weighed very conveniently. For refining on a large scale the introduction, without loss, of pure magnesium into the copper bath will prove less difficult.

The addition of an acid slag, finely powdered and easily fusible, after the deoxidation had taken place, proved very advantageous in these melts. This slag melted very rapidly and by vigorous stirring of the contents of the crucible it was possible to absorb all the magnesium oxide formed. In this way pure copper was always obtained.

In the beginning I used an iron calcium silicate slag (of the composition $\text{FeO} \cdot \text{CaO} \cdot 2\text{SiO}_2$); in later experiments a cupola slag of similar composition served the same purpose. An addition of quartz in the shape of building sand made the slag more liquid, when it had taken up much oxide.

The illustrations show two copper samples. Figs. 1 to 3 show a copper that had been melted in a crucible and was cast into a cylindrical mould of 60 mm. diameter. The electrolytic copper mentioned previously was used as starting material. The melt was made without any addition for refining. As the copper had a chance to absorb some of the furnace gases, it was very turbulent during casting and rose in the mould. The bar shown in the illustrations was cut lengthways, polished and etched with cuprous ammonium chloride. It was 600 mm. in length. The illustrations represent the bottom end, the top end and a cut through the center.

The whole bar is porous. When crystallization has started it has been disturbed by the development of gas. A strong tendency toward dendritic formation (pine-tree crystals) is noticeable all over the surface. The oxygen content of the copper after melting was

$$0.298\% \text{ O}_2 = 2.67\% \text{ Cu}_2\text{O}.$$

The microphotograph of Fig. 7 ($\times 75$ enlargement) corresponds to a highly poled copper. The darker portions represent cuprous oxide, which, mixed with copper, has the eutectic composition. The dendritic architecture of the copper crystals can also be seen.

The same quantity of electrolytic copper as used in the previous experiment was refined after melting by the addition of 0.1 per cent. magnesium. The bar cast from this melt underwent the same treatment as the one represented in Figs. 1 to 3. The results obtained with the magnesium treatment are shown in Figs. 4 to 6.

At the top (Fig. 4) a deep depression is shown, thus proving the absence of gas. The etching shows well-formed crystals. The varying luster of the crystals is due to their varying orientation toward the surface.

The corresponding Fig. 8 is a microphotograph ($75 \times$ en-

¹Translated from *Metall und Erz.*, May 22, 1913.

²Schenck and Henkelmann, *Metall. und Erz.*, vol. 1, 1913, p. 283.

³Siewerts and Krumpholtz, *Zeit. f. Phys. Chem.*, vol. 74, 1910, p. 3.

⁴Hein and Bauer, *Zeit. f. Anorgan. Chemie*, vol. 39, 1904, p. 1, and *Metallurgie*, vol. 3, 1906, p. 73.

⁵See the photograph in *Metallurgie*, vol. 19, 1909, p. 609.

⁶Schnabel, *Handbuch d. Metallh.*, 1901, vol. I, p. 270.

largement) and shows only very small amounts of inclusion in the copper.

The copper refined in this way and cast has a breaking strength of 18.5 to 21 kilograms per square millimeter, with an elongation of 30 to 37 per cent. It is very ductile and can be rolled into very thin sheets without tearing at the edges.

Table I shows the comparative results obtained when various substances were used as refining (reducing) agents in the melt.



FIG. 1.—TOP SECTION (WITHOUT MAGNESIUM TREATMENT)

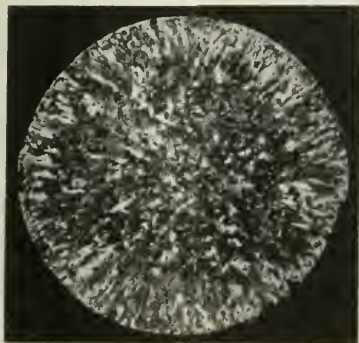


FIG. 2.—CENTRE SECTION (WITHOUT MAGNESIUM TREATMENT)



FIG. 3.—BOTTOM SECTION (WITHOUT MAGNESIUM TREATMENT)

Most of these belong to the series of elements given before.

For the melts I used those amounts of reducing materials, which, as determined by experiment, were required to produce a sinking of the copper when casting it. The residue remaining in the copper was determined later by analysis. This residue appears to be large in some cases in comparison with the amount added. This is due to the fact that a part of the

oxide formed is retained by the copper and shows in analysis. The starting material was in all cases the above-mentioned electrolytic copper.

Each of the bars produced was rolled to a wire of 8 mm. diameter and then drawn to 5 mm. diameter. The wire was

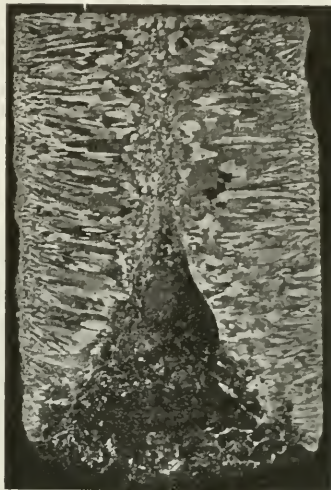


FIG. 4.—TOP SECTION (WITH MAGNESIUM TREATMENT)

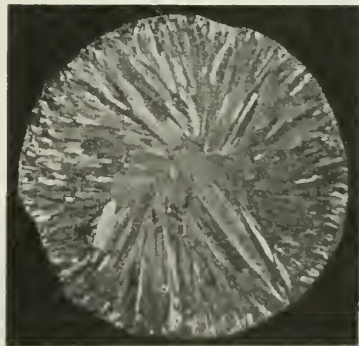


FIG. 5.—CENTRE SECTION (WITH MAGNESIUM TREATMENT)



FIG. 6.—BOTTOM SECTION (WITH MAGNESIUM TREATMENT)

not annealed; it, therefore, shows high breaking strength and small elongation. Generally speaking, the breaking strength increases with an increase in the addition of the reducing agent, as was to be expected. If the wire is annealed, the breaking strength decreases by 15 kilograms and more, while the elongation increases correspondingly.

The most important figures in the table relate to electrical

TABLE I

Addition of refining material in per cent.....	0.4 Sn	0.4 Zn	0.2 Ni	0.2 Fe	0.2 Mn	0.2 Si	0.15 Cr	0.1 B	0.1 P	0.1 Al	0.1 Mg	0.1 Mg	0.1 Mg	Starting Material
Content of the material in copper in per cent.....	0.37 Sn	0.35 Zn	0.18 Ni	0.16 Fe	0.13 Mn	0.17 Si	0.04 Cr		0.07 P	0.09 Al	0.013 Mg	0.024 Mg	0.039 Mg	
Electrical conductivity in per cent. of that of purest copper (100—60).....	73.0	73.0	79.1	58.8	70.5	55.3	91.0	83.8	62.1	83.8	98.5	94.0	89.1	97.1
Breaking strength, kg. per sq. mm.....	50.1	44.5	40.8	45.5	43.3	42.6	39.3	42.6	44.4	41.7	42.9	41.4	41.9	42.8
Elongation in per cent.....	3.5	3.0	2.0	3.5	3.0	3.0	3.0	3.0	2.5	3.0	2.75	2.0	3.0	2.0
Number of twists, radius of 100 mm.....	7	8	8.5	6.5	7	7	7	8	8.5	7	8	7.5	6	5.5

conductivity. This is always diminished by foreign substances in the copper. Special attention is, therefore, necessary to see that nothing of these foreign substances remains in the copper after they have been used for deoxidation. Very small amounts depress the conductivity considerably, and their use

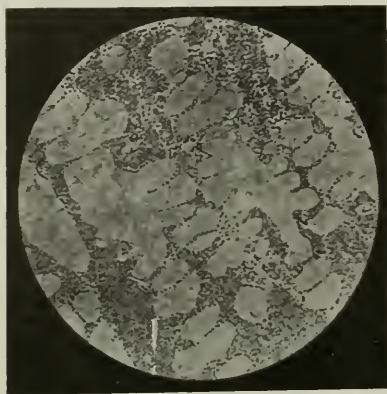


FIG. 7.—MICROPHOTOGRAPH (X 75 DIAM.) WITHOUT MAGNESIUM TREATMENT

for refining purposes is to be discouraged, if a high conductivity is required and a residue is unavoidable.

The figures in the table are self-explanatory. The three samples refined with magnesium show the highest electric conductivity. In the first of these three samples the conductivity

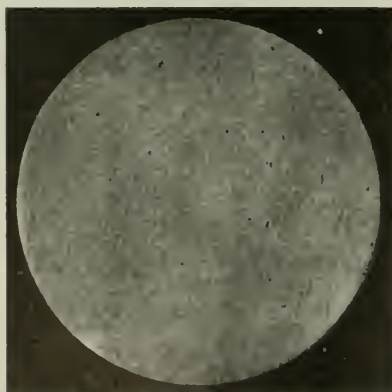


FIG. 8.—MICROPHOTOGRAPH (X 75 DIAM.) WITH MAGNESIUM TREATMENT

is higher than that of the starting material, although the latter was very pure and was considered to be of the highest conductivity.

It is to be noted that, though the amount of magnesium

added was the same in all three cases, the degree of purity differs for the three melts. This must be due to varying amounts of Cu_2O at the time of refining. It is difficult to determine the instantaneous degree of purity in small melts of a few kilograms in a crucible. In the regular operation on a large commercial scale this is easily possible.

As a result of these experiments it may be said that magnesium is well suited for the refining of copper. If the object is to remove the last traces of oxygen in desulphurized poled copper or in melted electrolytic copper a few hundredths of one per cent. of magnesium are sufficient. It may be assumed that when the refining by means of magnesium is carried out on a large scale with the use of a slag and simultaneous poling, the magnesium oxide will rise very quickly to the surface of the bath.

To sum up, the comparative experiments on the deoxidation of copper with different reducing agents show that the best quality of copper, especially the highest electrical conductivity, was obtained when using magnesium. The use of a bi-silicate slag for absorbing the resulting oxide proved very advantageous.

The Canadian bounty on lead ores has been extended until 1918, or until the remaining available sum of \$600,000 is expended. The bounty was originally established in 1903, at which time an appropriation of \$2,500,000 was made to stimulate lead production in the Dominion. The rate of the bounty is based on the price of lead, decreasing as the market price rises.

The Buckhorn cyanide mill in Eureka county, Nevada, is nearing completion as rapidly as possible. All the concrete foundations for machinery are in, and the crushing plant is practically finished. This property belongs to George Wingfield, and will be a thoroughly modern plant.

Registration and protection of patents and trade-marks in the Philippines has been provided for by an act of the Philippine legislature. Patents registered in the United States Patent Office, and then filed in the Executive Bureau of the Philippines, will receive the same protection as is accorded in the United States.

Cyanide will be placed on the free list if the Senate committee has its way in the revision of the tariff. The House proposed a tariff of $1\frac{1}{2}$ cents per pound, which is $2\frac{1}{2}$ cents lower than at present. It is uncertain what the outcome will be when the matter is considered in conference.

The University of Utah School of Mines is a beneficiary of the late James McGregor to the extent of \$50,000. Mr. McGregor was a mine owner in Utah, and interested in mining education. The fund is to be known as the James McGregor University of Utah Endowment Fund, and the income is to be used as the interests of the school may require. The Rose Polytechnic Institute, Terre Haute, Ind., was residuary legatee, and will receive about \$200,000.

Ferronickel and ferrochrome are to be produced electrically by a French company which is constructing hydroelectric power plants and electric smelting works in New Caledonia. The project is to cost over \$1,000,000, and will not be completed for two years.

The value of Montana metal production in 1912, according to the United States Geological Survey, was \$64,754,613, against \$46,955,287 in 1911. The increase was due largely to the greater production and higher price of copper.

Montana Meeting of the American Inst. of Mining Engineers

An Account of the Proceedings of the Eminently Successful Meeting Held at Great Falls, Anaconda and Butte, with Extracts of Papers

The 105th meeting of the American Institute of Mining Engineers was opened on Saturday, August 16, at Great Falls, Mont.

Saturday—Great Falls

One hundred members and guests registered on Saturday morning in Great Falls. The falls and hydroelectric power plants were visited in the morning, the Boston and Montana smelter in the afternoon, while in the evening a professional session for the presentation of papers was held in the Court House.

C. W. Goodale, for the local committees, made an address of welcome to the visitors, whereupon papers were presented by M. Hebgren, W. T. Burns, C. W. Goodale and J. H. Klepinger, E. E. Brownson, A. E. Wheeler and M. W. Krejci, D. A. Lyon and R. M. Keeney. Very great interest was manifest in the big hydroelectric developments and the possible uses of electric power for metallurgical purposes. Abstracts of the papers presented are herewith given.

Hydroelectric Development in Montana

The paper by **Max Hebgren**, Butte, Montana, on this subject first outlined briefly the natural features of the state of Montana affecting power development and then reviewed early water-power developments at the Big Hole plant, Canyon Ferry plant, Madison plant No. 1, and Black Eagle Plant, and the later developments of the Madison river system, the Missouri system, the Great Falls system, the Missoula river development, and the Big Fork development.

The total present generating capacity of the fourteen plants now operating on the combined systems of the Montana Power Company and the Great Falls Power Company is 72,100 kw (of which the capacity of the Rainbow plant is 25,000 kw and that of the Hauser Lake plant 14,000 kw). The principal transmission lines of the combined system operate at 46,200, 66,000 and 100,000 volts.

The paper concluded with notes on the uses of this power for lighting, traction, mining, and irrigation, and data on new power developments and undeveloped powers. The larger possibilities for developing new powers exist in the northwestern part of the state of Montana where developments like the Flat-head lake, Kootenai falls and other large streams present possibilities for the development of at least 250,000 hp.

Great Falls Electrolytic Plant

The paper by **Willis T. Burns**, Great Falls, Mont., on this subject, is printed elsewhere in this issue, practically in full.

Great Falls Flue System and Chimney

A long paper by **C. W. Goodale**, Butte, Mont., and **J. H. Klepinger**, Great Falls, Mont., describes in detail the new flue system, completed in 1909 at the Boston and Montana Smelter at a cost of \$1,100,000. It includes a chimney 506 ft. high and 50 ft. inside diameter.

A comparison of the losses of copper, silver and gold, from the old flue system and from the new flue system shows that the new flue system has effected a saving of \$372.18 per day, or \$130,263 per year.

In addition to this saving it is expected that a substantial revenue will be derived from the treatment of the more arsenical dust, in the recently completed arsenic plant.

In considering the large expenditure for the installation of

this elaborate flue system, it should also be borne in mind that a reduction in losses of flue dust was not the only object in view. The low elevation at which the gases were discharged from the old chimney was a source of annoyance from smoke complaints. The overtaxed condition of the old flues and chimney prevented the operation of certain furnaces at full capacity, and for the same reason the smoke and fume around the furnaces materially interfered with the efficiency of labor. Furthermore, dust chambers were inadequate and their poor arrangement made the handling of flue dust expensive and troublesome.

All these defects have been eliminated, and the anticipated efficiency of the new flue system has been fully realized.

Determination of Arsenic and Antimony in Copper

A paper by **E. E. Brownson**, Butte, Mont., dealt with the determination of arsenic and antimony in converter and electrolytic copper. It described the methods used in the laboratory of the Boston and Montana Smelter. The determination is finished in 24 hours.

Great Falls Converter Practice

A long paper by **Archer E. Wheeler** and **Milo W. Krejci**, Great Falls, Mont., gave many interesting details of Great Falls converter practice.

After a general chronological review of the principal events and changes of practice at the Boston and Montana Smelter, the authors gave more detailed accounts of mechanical manipulation, casting of finished copper, tamping of linings, early acid linings, acid linings with the use of ore, introduction of flux through the tuyeres, bottom blast, disposal of converter slag, development of different classes of converters (five classes being distinguished, successively increasing in size, the largest class V being now considered the standard for the plant), basic lining, number and size of tuyeres, cast copper tuyeres, efficiency of air, size of converter mouth, height of converter.

The chief conclusions of the authors are as follows:

The basic lining has shown its superiority over the acid lining and therefore the acid lining can be dismissed as not being the best practice.

A large converter is better than a small one. There is no practical limit to the size of a converter which will work metallurgically. The limits are mechanical, and depend on the size of unit a plant will stand from the point of view of quantity of production, but even in this latter instance there can be a rather wide range of choice. The authors have found that the Class V (latest and largest size) or 20-ft converter can be used as a storage for matte or for finished copper for long periods if the mouth is properly covered. Therefore in a plant of comparatively small production, a combination of ample matte storage capacity in blast-furnace settlers and reverberatory furnaces, together with a large converter, can be made so as to allow all converting to be done on one shift, and the casting of the copper perhaps to be done on the next shift.

With the success of the large units even the largest plants should have only a few large units and not a large number of small units.

From the experiments and observations which have been made, the authors feel that the tuyeres for any converter from 12 to 20 ft. diameter should be about 2.25 in. internal diameter. Tuyeres 2.75 in. internal diameter have not proved successful.

Tuyeres smaller than 2.25 in. diameter introduce undesirable construction features, particularly weakening the lining at the tuyeres, introduce unnecessary resistance to the entrance of the air, and add to the work of punching.

The mouth of the converter should be of ample size. Openings of 6 ft. 6 in. diameter for a 12-ft. converter and 8-ft. for a 20-ft. converter have not been found too large.

The bottom lining must not be too close to the tuyeres on account of a tendency of the bottom to build up and interfere with the punching of the tuyeres and also often causing a concentration of air in a few places with consequent throwing out of the charge. At least 5 in. should be allowed from the lowest point of the tuyeres to the bottom lining. There is no reason why this dimension cannot be increased, the determining factor being the size of charge it is desired to finish. There must always be copper enough at the finish to cover the tuyeres and a very deep bottom would require a large charge.

Air can be used efficiently in large volumes, for instance in the 20-ft. converter up to 22,000, or even more, cubic feet per minute. However, the best results seem to be for this converter about 18,000 cu. ft. while slagging.

As to the height of the converter, the Class IV, 13 ft. 8.5 in. high, and the Class V, 17 ft. 7.75 in. high, have proved very satisfactory with proper attention to the details of operation, and trials of a taller converter were not satisfactory. Probably the converter should not be made appreciably lower. The proper dimension to quote in this particular is from the center of the tuyeres to the lowest point of the mouth. These dimensions for the two classes are: Class IV, large mouth, 7 ft. 11 in.; Class V, 8 ft. 9.25 in. Class V is now considered the standard for the plant.

These the authors believe to be a few of the basic principles which have been demonstrated in practice.

Smelting of Copper Ores in the Electric Furnace

A paper by Dorsey A. Lyon and Robert M. Keeney, of the U. S. Bureau of Mines, Pittsburgh, Pa., discussed the smelting of copper ores in the electric furnace.

The authors first emphasize that the electric furnace was not developed as a competitor of the combustion furnace, but:

(1) For the purpose of doing high-temperature work which it is not possible to do in the combustion furnace; and (2) for the treatment of ores from deposits which are located in regions where fuel is scarce and costly, and where hydroelectric power is comparatively cheap, as is the case in Chili, in Canada, in certain parts of this country and Mexico.

"This paper is not presented with the idea of trying to prove that the electric furnace should replace the reverberatory or the blast furnace as used at present in the smelting of copper ores, but that it may be substituted for them in those localities which are not favorable to them, and where either of the two following conditions may exist:

(a) "A district remote from railway transportation facilities, and where there is a deposit, or deposits, containing sufficient values to warrant their being worked, but which, due to the cost of getting coke in for operating a smelter and of getting ore out to a smelter, have to remain unworked. And where, on the other hand, there is plenty of water power which can be developed at reasonable cost for the production of electric power, by the use of which for smelting the ores in an electric furnace, with subsequent Bessemerization if necessary, a concentrated product may be obtained, which can easily stand the necessary transportation charges.

(b) "Or, comparatively speaking, the transportation facilities may be all right, but the cost of fuel be too great to permit of the ordinary methods of smelting, whereas, on the other hand, hydroelectric power can be developed in the district at a low cost, and the ores could be smelted by the electric furnace at a cost that would make their treatment possible."

The authors then take up the principles of the chemistry of copper smelting and concisely deal with the treatment of native copper ores, oxide and carbonate ores, and sulphide ores.

As to *reverberatory smelting*, the reverberatory furnace may be said to be used for the smelting of ores and concentrates which cannot be successfully treated in a blast furnace. If, as Peters states, the proper function of the reverberatory furnace, in smelting copper ores, is to generate heat as rapidly as possible, there is no apparent reason why copper ores cannot be smelted equally as well in an electric furnace, and perhaps even more advantageously, for the thermal efficiency of the average reverberatory furnace is only from 5 to 8 per cent., although about 20 per cent is now obtained in the large Anaconda reverberatories, whereas, that of an electric furnace which would be used for smelting copper ores suitable to refractory smelting, would probably have an efficiency of as much as 70 per cent. Chemically the only ill effects that might possibly result from the use of such a furnace would be the effect of the carbon of the electrodes upon the iron oxides which might be contained in the charge and which would result in:

1. The loss of electrodes.

2. The production of metallic iron, which would contaminate the metallic copper and increase the cost of refining; or, if a matte was the final product, might necessitate re-concentration.

3. The loss of electric energy, due to the additional heat consumed by the reduction of iron.

Since the reverberatory furnace is essentially a means of melting, whether used for sulphide, oxidized or native ores, the results obtained in the electric smelting of native copper ores will serve for the sulphide and oxide ores also, for comparison with the reverberatory in order to clear up the three points referred to above. The slags would be somewhat similar in each case. The atmosphere of the electric smelting furnace charged with sulphide ore would not be as neutral as in the furnace charged with native copper ore, because of the tendency of the elemental sulphur and any sulphur dioxide formed to dilute the air so as to make the furnace practically reducing as far as the consumption of electrode by oxygen from the air was concerned. If the electric furnace was not kept closed, considerable sulphur dioxide might be formed, which would possibly be reduced in part by the carbon electrode. Even with such reduction of sulphur dioxide, the electrode consumption would probably be less when working on sulphide ores than when working on native copper ores. This is shown by Bureau of Mines experiments, when, with a charge of native copper concentrate, the maximum electrode consumption in fourteen experiments was 480 lb. of graphitized carbon per ton of charge, the minimum, 22.1 lb. and the average 34.5 lb. In the same furnace, with charges of sulphide, oxide and siliceous ores mixed, a matte being the product, the maximum electrode consumption for 20 runs was 60.0 lb., the minimum, 10.8 lb. and the average, 23.3 lb.

SMELTING OF FINE MICHIGAN NATIVE-COPPER CONCENTRATES IN THE ELECTRIC FURNACE

For the purpose of investigating the feasibility of smelting fine Michigan native-copper concentrates in the electric furnace, instead of the reverberatory furnace, experiments were conducted by the authors. The objects of these experiments were:

(1) To note the effect of variation of the three main slag constituents, silica, ferrous oxide and lime, upon the power consumption, the operation of the furnace and the purity of the copper produced.

(2) To ascertain the extent of iron oxide reduction caused by the graphite electrode.

(3) To compare intermittent smelting with continuous smelting.

Two grades of concentrates were smelted: No. 3 and No. 4, of which No. 3 was much the coarser; of the former, 69.73 per cent passed through a 60-mesh sieve, as compared with 88.12 per cent of No. 4. The chemical analyses of the ore were as follows:

No. 3. 37.35 per cent Cu, 33.33 SiO₂, 15.40 FeO, 8.78 Al₂O₃, 5.18 CaO, 1.09 MgO, 0.056 S and 0.30 Fe (metallic)

No. 4. 25.35 per cent Cu, 25.92 SiO₂, 21.60 FeO, 12.40 Al₂O₃, 5.98 CaO, 1.09 MgO, 0.096 S and 0.50 Fe (metallic).

Fifteen experiments were performed in a Siemens type of electric furnace, having a capacity of 50 kw. The furnace was operated as a resistance furnace. The charge and products of a typical experiment were the following:

Charge	Black Copper	Slag
	Lb.	Per Cent.
Concentrate No. 3.	22.0 Cu.	98.59 (by difference)
Hematite.	1.1 Fe.	1.10
	S.	0.29
	As.	0.02
		Per Cent.
	Cu.	0.15
	SiO ₂	44.88
	CaO	6.92
	MgO	2.32
	FeO	29.80
	Al ₂ O ₃	15.34

The author's conclusions from these experiments are as follows:

(1) Fine Michigan native-copper concentrates can be smelted in the electric furnace with the production of a good grade of black copper and without excessive losses of copper in the slag or by other means.

(2) The percentage of copper in the slag need not exceed 0.25 per cent., with a slag of proper composition. The loss in the slag should not exceed 0.50 per cent of total copper charged. Other losses should not exceed 1.0 per cent; giving a total loss of 1.5 per cent of all the copper charged.

(3) The loss of copper by volatilization is high, if the slag is much more acid than a mono-silicate, and if it contains much alumina. The lowest losses were obtained by addition of limestone to form a slag containing 35 per cent SiO₂, 22 per cent CaO and MgO, and 25 per cent FeO.

(4) The difference between limestone and hematite as a flux is not marked, one being about as efficient for this purpose as the other, if the slag has not too high a specific gravity. No more iron was found in the black copper product when hematite was used than when limestone was used.

(5) With a furnace operating at a low temperature on a mono-silicate slag, the reduction of iron by the carbon electrode should not be excessive or sufficient to result in a product containing less than 95 per cent. metallic copper. With care a product containing 98 per cent. copper could probably be produced regularly.

(6) The total electrode consumption greatly exceeds the amount of carbon necessary for reduction of the iron found in the metal. An average of 0.35 per cent. of the total was used in iron reduction. The electrode consumption should not exceed 10 lb. per ton of charge.

(7) The electric furnace operates satisfactorily with the high siliceous alumina slags produced by the natural gangue without flux, but the losses by volatilization are high because of the high melting point of the slag. There is no copper oxide in the product, as was shown by microscopic examination.

(8) The power consumption for smelting fine Michigan native copper concentrates of between 25 and 40 per cent. copper in an electric furnace of 750 kw. or greater capacity should not exceed 640 kw. hr. per ton of ore smelted.

PROPOSED PROCESS FOR ELECTRIC SMELTING OF FINE NATIVE COPPER CONCENTRATES.

The commercial smelting of fine native copper concentrates in the electric furnace to attain the greatest economic success would be carried out in a continuous operation, consisting of charging the mixture of ore and flux at intervals into the top of an electric furnace having a short stack above the smelting crucible for preheating the charge before it reached the smelting zone, and tapping off the metal at intervals, the slag being permitted to run continuously from the furnace through a slag notch above the metal hole. The possibility of operation in this manner was shown in an experiment where there was an average of 1.10 per cent. iron in the product and 1.37 per cent. copper in the slag. The total loss of copper was as low as in the intermittent runs, and was largely due to the small scale operation. The slag loss was high because of imperfect settling, and because of the tapping of partially fluid material with the slag; a condition that would not occur in a large

furnace operated properly. The power consumption was lower than in the other experiments.

The essential conditions for the successful performance of this process are a crucible, where the electrical energy is introduced and where the metal settles from the slag, and a short shaft above the crucible for preheating the charge before it reaches the smelting zone. As there is no reduction to be

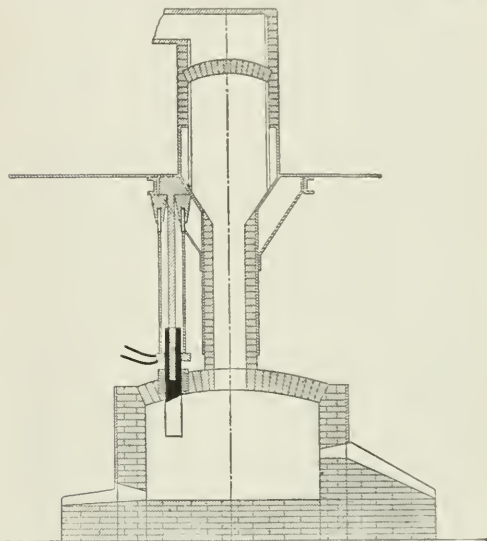


FIG. 1.—ELEVATION OF ELECTRIC FURNACE FOR SMELTING NATIVE COPPER CONCENTRATE

performed, this shaft need not be higher than to give plenty of head room between the top of the crucible and the charging floor for the moving of the electrode.

A furnace of the type shown in Figs. 1 and 2 satisfies these conditions. It consists of a smelting chamber lined with fire brick or silica brick, having three carbon electrodes suspended vertically through the roof with a feeding shaft lined with fire brick over the crucible. A 750-kw. furnace would treat, per 24 hr., about 23 tons of native copper concentrate, containing 25 to 40 per cent. copper. This furnace would be operated with three-phase current at from 50 to 100 volts. The general internal dimensions of the

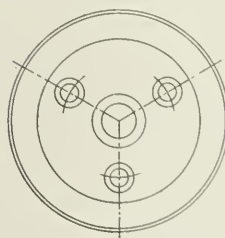


FIG. 2.—PLAN OF ELECTRIC FURNACE FOR SMELTING NATIVE COPPER CONCENTRATE

crucible are 14 ft. 6 in. in diameter, by 6 ft. high, with a shaft 1 ft. 6 in. in diameter by 18 ft. 6 in. high. With a furnace of this type regulation of electrodes would not be necessary after regular operation was established, so that the electrical connection is made close to the roof of the furnace and regulation is by hand. There is less difficulty in handling the electrodes if they are suspended vertically; and, as the shaft is low, it is made vertical, thus eliminating the necessity of suspending them at about 55 degrees.

The slag from such a furnace should not contain over 0.25 per cent. copper, and could be discarded at once without further treatment. It could be granulated with water as discharged from the furnace. The metal would require further refining for elimination of iron and other impurities, which could be done by charging the hot metal directly into a re-

reverberatory furnace, where it would be refined as previously described.

Briefly, the present process for treating these concentrates consists of the following steps: first melting and refining the coarse concentrate in either one or two reverberatory furnaces, from which a slag is produced containing from 10 to 30 per cent. copper; second, agglomerating the fine concentrate in a reverberatory furnace, or briquetting it with lime; third, smelting the slags of the reverberatory furnaces and the agglomerated or briquetted fine material, with coke in a blast furnace, the slags from which contain 0.6 to 0.8 per cent copper and the metal, 90 to 95 per cent copper; and fourth, retreatment of this metal in a reverberatory furnace.

The use of the electric furnace for treatment of these concentrates gives the following advantages: first, no agglomeration or briquetting would be necessary; second, no slags produced by this furnace would require retreatment; third, a large proportion of the higher-grade concentrates containing 75 per cent copper could be mixed with the fine concentrates and treated directly in the electric furnace, thus reducing the amount of slag necessary for re-smelting; and fourth, if cheap power was available the electric furnace could be used for reducing the copper from the reverberatory slags, and the resulting metal would probably be lower in iron and the slag lower in copper, because of the greater ease of regulating the reducing atmosphere, as only sufficient coke is present for the reduction of combined copper.

The limit to which the higher-grade concentrates could be mixed with the fine material would be determined by the electric conductivity of the mixture in the furnace. In the experiments performed with a charge containing 37 per cent metallic copper, no difficulty from short circuits or other causes was experienced. The point to which the amount of copper in the charge could be increased would have to be ascertained by experiment.

Technically the electric furnace does away with the necessity of briquetting or nodulizing, and, in general, facilitates handling the materials. The commercial factors influencing the cost would largely depend upon local conditions as to cost of power and fuel. Opposed to the cost of power would be the cost of fuel and briquetting or nodulizing, with possibly a higher labor cost for the present practice. If the cost of power was equal to these three factors, and no less, the electric furnace might still have the advantage of facility of operation. Also, in the case of a mine remote from smelting plants, it might be cheaper to concentrate the low-grade product of wet concentration by smelting in a small electric furnace, and thus reduce the cost of freight. Finally, to save losses of wet concentration, the ore might not be concentrated to so high a degree in this manner, but a low-grade concentrate made, which would be smelted directly in the electric furnace and the high-grade black copper shipped to the custom smelter.

We have obtained figures from a reliable source showing that the cost of treatment of these fine concentrates by agglomeration in a reverberatory furnace, refining of this copper produced, retreatment of slags in the blast furnace, and remelting and casting the cupola copper into marketable shapes, is \$8.64 per ton of 35 per cent native copper concentrate. This includes overhead charges, all repairs and maintenance of plant, but not amortization.

The treatment of the same material by the proposed electric furnace process to produce marketable shapes is estimated as follows:

COST OF ELECTRIC POWER		Cost of Treatment Per Ton of Concentrate
Per Kilowatt Hour	Per Kilowatt Year	
0.5	\$1.80	\$7.18
0.625	1.80	8.08
0.75	65.80	8.87

A rate of one-half cent per kilowatt hour with a high load

factor is possible in many districts with the use of steam turbine plant, if cheap hydroelectric power is not available. The load factor in this case would be above 90 per cent, and the power factor not below 0.85. In practice the cost of refining "cupola" block of the electric furnace would probably be lower than in the case of blast furnace cupola blocks, because of the higher percentage of copper in it. In the reverberatory process 81.3 per cent of the material charged must be retreated as slag in the blast furnace. In the electric furnace process, the only material to be retreated would be about 5 per cent as slag from the refining of the cupola blocks in the reverberatory. This slag could be retreated in the electric furnace. The great saving in retreatment charges accounts for the possibility of use of the electric furnace in this connection, even with expensive electric energy.

With respect to ordinary copper-blast-furnace smelting with carbonaceous fuel the author's first deal with the prerequisites of the process and the different losses and discuss whether any of the losses in a blast furnace could be avoided in an electric furnace. The dust losses, due to blast, would be less, although more or less fine particles would be carried away as dust by the gases escaping from the electric furnace. Volatilization losses would be less if the electric furnace would be operated more in the manner of a resistance furnace than as an arc furnace.

As to the losses in the slag due to particles of matte or metal, which have failed to separate properly from the slag, such losses would occur just the same if an electric furnace were substituted for the blast furnace, but there is no reason to believe that they would be any greater and on the other hand they should not be as large, for, due to the fact that it is possible to so control an electric furnace as to obtain the temperature desired, a sufficiently high temperature could be given to the matte and slag before it left the furnace to insure its being fluid enough to cause perfect separation in the settler, which would thus obviate this loss. As to the oxidation of valuable metals, this loss should be reduced to a minimum in a furnace operation without a blast, for the loss by oxidation is doubtless due to the action of the blast upon the products of the furnace as they pass the region of the tuyeres.

The authors summarize the experimental work which has been done before by others on the electric smelting of raw sulphide, oxidized and carbonate ores. This comprises Vattier's experiment (Report of Canadian Commission, 1904), Wolkoff's experiments (*Metallurgie*, vol. 7, p. 99) and Stephan's experiments (*Metal and Erz*, October, 1913; *METALLURGICAL AND CHEMICAL ENGINEERING*, January, 1913, page 22).

ELECTRIC SMELTING OF SULPHIDE ORES

The authors then describe a series of experiments which they have carried out themselves at the Bureau of Mines Laboratory. The objects in view were: (1) to determine if there are any conditions arising which make electric smelting without air anything but a simple melting operation; (2) to note the percentage of concentration and the sulphur removal; (3) to study the possibility of condensation of the elemental sulphur as such; (4) to get general figures on power consumption with varying charges; (5) to determine gold, silver and copper losses, and (6), to study the use of a low-copper matte as a collecting agent for gold and silver.

1. *Ores Treated:* The ores used in the experiments were: A

	Pyrite, Per Cent	Nodule, Per Cent	Siliceous Ore, Per Cent
SiO ₂	2.50	4.00	74.53
Fe	44.07	65.60	10.71
S	48.20	...	8.36
Al ₂ O ₃	0.24	1.76	0.40
CaO	0.08	0.45	...
MgO	0.73	0.46	0.26
P	0.02	...	...
As	0.20	...	...
Cu	1.30	0.07	...
Au	0.01 oz. per ton	0.03 oz. per ton	1.28 oz. per ton
Ag	0.14 oz. per ton	0.05 oz. per ton	3.20 oz. per ton

heavy sulphide low-grade copper ore; a gold; a silver, siliceous

ore, and some roasted ore of the analyses shown below. The limestone contained 63.2 per cent CuO and 5.7 per cent MgO .

The ores were treated in the electric furnace described under the native copper experiments. The top was roofed and kept closed tightly to prevent escape of sulphur and admission of air. A condenser, consisting simply of three rectangular, horizontal chambers with several baffles in it, was attached to the furnace to condense the sulphur and catch any escaping dust.

Twenty experiments were made with these ores. A typical charge was as follows:

Charge		Matte		Slag	
Pounds		Per Cent.		Per Cent.	
Pyrite	8.8	Cu	1.22	Cu	0.05
Nodule	13.2	Fe	64.18	SiO_2	35.25
Siliceous Ore	5.76	S	22.38	FeO	41.30
Limestone	2.73		Oz. per Ton	Al_2O_3	4.40
		Au	0.72	CaO	10.00
		Ag	0.96	MgO	1.64
				S	3.66
				Oz. per Ton	
				Au	0.03
				Ag	0.32

2. *Conclusions:* (1) Smelting in an electric furnace treating sulphide ores consists simply of melting the charge, volatilizing about 60 per cent of the sulphur as elemental sulphur and separating the slag and matte.

(2) The ratio of concentration is simply that possible from a separation of matte and slag, together with a reaction of oxides and sulphates upon sulphides, and the volatilization of 10 per cent more than one atom of sulphur, but with no formation of iron oxide to enter the slag, because there is nothing present to oxidize the iron sulphide.

(3) Qualitatively, it is possible to condense some of the elemental sulphur driven off.

(4) The loss of copper, by volatilization and in the slag, is low.

(5) A matte containing about 1 per cent copper, from a charge containing 0.30 per cent copper, makes a good collecting agent for gold and silver, if a clean separation of slag and matte is obtained.

(6) There is very little loss by volatilization of gold in the electric furnace, but there is some loss of silver in this way. This would not probably occur in a larger furnace with closer temperature regulation.

(7) The electrode consumption in the smelting of a sulphide ore is low, and in practice need not exceed 5 lb. per ton of charge.

(8) In a large commercial furnace the power consumption for most ores would be about 480 kilowatt-hours per ton of ore, or 0.055 kilowatt-years.

ELECTRIC FURNACE VS. THE REVERBERATORY FURNACE AND BLAST FURNACE AS A MELTING AGENT

The authors believe to be justified in making the following statements as to the possibility of using the electric furnace for this purpose:

(1) The melting of native copper oxide or sulphide ore of copper can be done just as efficiently in the electric furnace, and perhaps more so, than in either the reverberatory furnace or the blast furnace.

(2) The reactions desired in reverberatory smelting, or ordinary blast furnace smelting, can be obtained in the electric furnace as well, and perhaps better, than in either of those furnaces.

(3) The loss of electrodes is small, varying from 5 lb. to 10 lb. per ton of ore melted, and the presence of the carbon electrode does not cause enough reduction of iron to be troublesome, or increase the consumption of electrical energy appreciably.

(4) The losses of copper, gold and silver by volatilization and in the slag would be no greater, as shown in all the experimental work cited, than they are in reverberatory smelting or ordinary blast furnace smelting.

(5) A matte containing as low as 1 per cent copper can be used as collecting agent for gold and silver in the electric furnace, as well as in combustion furnaces.

(6) The comparison of costs would depend entirely upon the nature of the ore treated, and the relative cost of coal or coke, and electrical energy. In general, from the work done in the experiments, we may say that from 500 to 700 kilowatt-hours per ton charged would be required for smelting copper ore, depending upon the nature of the ore.

SMELTING SULPHIDE ORES WITHOUT ROASTING, OR PARTIAL PYRITIC SMELTING

The authors finally take up semi-pyritic and pyritic smelting of sulphide ores, the object being to bring about the required degree of oxidation and the melting, at practically the same time, and to obtain either a whole or a part of the heat required for the process from the combustion of the iron and sulphur contained in the ore. As is well known, if practically all the heat necessary for the process is obtained in this manner, we have what is known as pyrite smelting. If only a part, semi-pyrite smelting.

The question, therefore, that presents itself in connection with the smelting of copper ores in the electric furnace is to determine whether electric heat may be used to replace the heat which is derived from the combustion of coke in pyrite and semi-pyrite processes. In other words, whether it would be possible, and if so, whether it would be commercially feasible, to carry out these processes in an electric furnace so constructed as to use a blast, thus obtaining as much heat as possible from the oxidation of the sulphur and iron, and supplying by electric energy whatever additional heat might be needed to carry out the process successfully.

The authors assume an electric furnace similar in construction to a modern copper blast furnace, and that part of the furnace, including the tuyeres, is practically identical with the same. Below the tuyeres, the furnace could be constructed as shown in Fig. 3. By referring to this it will be noted that there are electrodes extending down into the crucible, the arrangement of which, along the sides of the crucible, can be noted by referring to the plan of the furnace in Fig. 4.

The authors discuss at some length the possibilities of semi-pyritic smelting in an electric furnace and conclude that there does not seem to be any metallurgical reason why the chief objects of the pyrite smelter cannot be carried out in an electric copper blast furnace as well as in a coke copper blast furnace; namely, "to melt the great mass of SiO_2 and inert earths, to oxidize enough of the sulphides in the charge to insure a suitable matte—incidentally obtaining the heat evolved by the oxidation." On the other hand, it would seem that the difficulties ordinarily encountered in operating a pyrite furnace might be avoided when using an electric blast furnace, namely, the difficulties which arise when too much or too little coke is

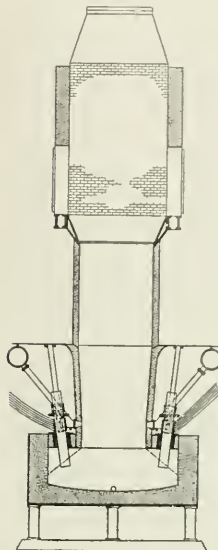


FIG. 3.—ELEVATION OF ELECTRIC BLAST FURNACE

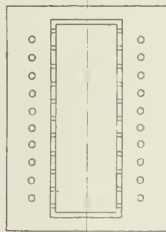


FIG. 4.—PLAN OF ELECTRIC BLAST FURNACE

added to the charge, and moreover that it not only permits "introducing heat into the furnace, without at the same time robbing the combustion zone of oxygen," of which Dr. Peters states, "nothing would be so welcome to the furnace man as to do this," but would also permit of the heat being entirely under control and easily regulated, and thus avoid freeze-ups with their consequent vexations and costly delays. By this the authors do not mean that any sort of a charge could be put through the furnace and not freeze it up, an idea which seems to be quite prevalent in regard to electric furnaces, but that, if the charge be properly calculated, a very much wider variation would be permissible in the composition of the slag from that calculated than would be the case in blast furnace smelting.

By referring to Figs. 3 and 4, it will be noted that the chief difference in the construction of the furnace would be in that part of it below the tuyeres—in other words, in the crucible. Because electric furnace construction has received the attention of some of the very best mechanical and electrical engineers, and because a crucible based upon the principle shown in the design in Fig. 3 is now extensively used in the iron electric reduction furnaces of Norway and Sweden, the authors believe that no difficulty would be experienced in this respect. Various methods and arrangements can be used for connecting up electrodes. If this be true, only the matter of costs has to be considered.

In figuring the cost, the authors assume that the furnace is to smelt 384 tons of charge per day that the charge contains 5.21 per cent Cu, 26 SiO₂, 19 FeO, 11 S, 4 Al₂O₃, 17 CuO. Such a charge has been smelted at the Washoe smelter with 8.2 per cent coke, or with 164 lb. of coke per ton of charge. If its calorific value was replaced by electric heat, the authors reason that taking into account the relative efficiencies of the furnaces, 400 kw. hours would be required to smelt one ton of charge. Since 384 tons are to be smelted per day or 16 tons per hour, 6400 kw. would be the constant load of the furnace.

"We may say that the costs of coke and electrical energy are about on a par when coke costs \$7 a ton, and electrical energy 0.15 cents per kilowatt hour, or \$13 per kilowatt year,—or say in the ratio of 1:1.8. Although there may be advantages in the use of an electrical blast furnace instead of an ordinary blast furnace, these will not be taken into consideration in this connection, and we will assume that the cost of electrodes and of the electricity on the one hand, and its equivalent in heating value of coke on the other, would be the same; in other words, that the cost of electrodes and electricity would balance the cost of the amount of coke that would be necessary to do the same amount of work. We will not take into consideration the saving in blowing by reason of not having to furnish air for burning the coke.

"The average consumption of electrodes per ton of iron produced at Trollhättan, Sweden, is about 10 lb. Assuming 1 ton of iron is equivalent to 2 tons of charge, the consumption of electrodes per ton of charge smelted in a furnace, such as the one we have been discussing, would not be over 5 lb. per ton. If we assume that the electrodes do not cost over 6 cents per pound, which allows a fair margin for transportation over their cost at the factory, the cost of electrodes would be about 30 cents per ton of charge smelted. The consumption would be probably considerably less than 5 lb. per ton, as was shown in an experiment performed by the writers in which air was blown into the furnace, and in which the electric consumption was only 2 lb. per ton of ore treated. Such being the case, if our combined electrical energy and electrode cost is not to exceed our equivalent coke cost, with coke at \$9 a ton, the cost of electrical energy would be $9 \times 1.8 = \$16.20$ a kilowatt year, minus the cost of electrodes, or about \$16.00. Compared on the basis of calories, if the coke contains 80.24 per cent fixed carbon, it would contain 1604.8 lb. of carbon per ton of 2000 lb., and this carbon would, if completely burned to CO₂, give a calorific value of about 13,000,000 calories. As one kilowatt year of energy is equal to 16,616,000 cal., it would require something

like 1.2 tons of coke, containing 80.24 per cent fixed carbon, to equal in calorific value the kilowatt year, but it is to be remembered that this assumes that all the carbon of the coke is completely burned to carbon dioxide, which rarely happens in practice, and so, as has been determined from actual practice, it requires from 1.5 to 1.8 tons of coke to yield the same number of calories as may be obtained from 1 kilowatt year of electrical energy."

THE USE OF THE ELECTRIC BLAST FURNACE IN THE SMELTING OF PYRITIC ORES

"As previously stated, in all true pyritic smelting, fuel (usually coke) in varying amounts is added to the charge. We will assume for the sake of argument, that the ore being treated is entirely suited to pyritic smelting, and that it is not necessary to use more than 0.5 per cent of the weight of the charge in coke. Could such an ore be treated pure as well in an electric blast furnace as in an ordinary blast furnace, assuming that the costs would be the same in each case. As what we have had to say in regard to the construction of the furnace and the costs of applying the process is as applicable to true pyritic as it is to semi-pyritic smelting, we will only take up the subject from the metallurgical and chemical standpoint.

"In the first place let us consider the nature of the atmosphere in a pyrite furnace. As the air enters the tuyeres, practically all of it enters into immediate combination, forming SO₂ and FeO. Although some oxygen escapes combination, it is not sufficient to support combustion within the furnace, as has been shown by Sticht and others. Now let us briefly consider the changes which take place in the charge as it descends from the charging door to the fusion zone.

"They are few, and may be briefly stated as follows:

- (1) "The iron sulphide loses one-half of its sulphur as elemental sulphur, and becomes FeS.
- (2) "Chalcopyrite. It loses about one-fourth of its sulphur and becomes practically a mixture of Cu₂S and FeS.
- (3) "The silica is unchanged.
- (4) "The coke is for the most part consumed in the upper zone of the furnace, but not by oxygen, as no free oxygen exists in that part of the shaft, but by the O from SO₂, that is, the carbon of the coke reduces the SO₂ giving S and CO, and this CO reacts with more SO₂, forming S and CO₂.

Aside from these reactions, which are of no great importance in the chemistry of the process, the chief office of the coke as pointed out by Sticht, 'Is apparently to heat up the sulphides and the quartz in preparation for their active oxidation deeper in the furnace.' If, therefore, coke was omitted from the charge we would not have this preparation of the sulphides and the quartz as stated by Sticht. Such being the case, how detrimental to the process would this lack of preparation be?

"In order to answer this question, it must be borne in mind that in true pyritic smelting, there is 'no heat to spare,' and that although, as pointed out by Sticht, the heat evolved by the combustion of the coke by the oxygen of sulphur dioxide is only one-third as much as it would be if the oxygen were furnished direct by the blast, and that it is given off at a point considerably higher than the focus of the furnace, it nevertheless 'assists in preparing the charge for the reactions which take place in the focus of the furnace, for with 1 per cent of coke in the charge enough heat is generated to supply one-third of the heat required to melt the entire pyrrhotite contents of the charge.' This extra heat 'doubtless furnishes just the necessary aid to bridge the operation over some critical point.'

"From this we see that the office of the coke then is solely for the purpose of furnishing heat. Inasmuch as the nature of the charge in true pyritic smelting is such as to demand the oxidation and removal of a large excess of iron of silicate, and as it is necessary to cause the silica present to unite with as much FeO as possible (due to the fact that

generally silica-bearing materials have to be added for slag-forming purposes, and often these are barren and hard to obtain), and as the saturation point of silica with iron oxide is greater the higher the temperature attainable in the pyritic furnace, it is quite necessary that the reactions in the zone of the furnace proceed as energetically as possible, in order to meet the requirements stated above, and to provide enough heat to keep the process going.

"If, therefore, pyritic smelting be attempted in an electric blast furnace, it would be necessary to supply the heat from the electric current in such a manner as to insure there being enough heat at the focus to carry on the reactions at that point in the same manner as if coke had been added to the charge. The writers are of the opinion that it would be possible and perfectly feasible to do this in the furnace of the type shown in Fig. 3. Moreover, that the use of the electric furnace would permit of a more uniform operation of the furnace than is now possible, for, with the electric current, it would be an easy matter to quickly increase or decrease the heat required for the successful operation of the furnace. In this way we could maintain the 'degree of concentration at a constant,' which, as pointed out by Sticht, is very important, but which is extremely difficult to do. Also, we could avoid the troubles arising from the 'constant fluctuations' which are bound to occur, and which at present demand unbroken attention and frequent change in the charge, especially in its proportion of silica 'due to the fact that in true pyritic smelting there is no heat to spare,' whereas, with the electric blast furnace, it would be possible, as just stated, to increase or decrease the heat as necessary in order to meet these fluctuations.

SUMMARY

"Summarized then we may say that the electric smelting of copper ores is nothing more than the substitution of electric heat for the heat derived from the combustion of carbon. Inasmuch as the carbon which is used either in the reverberatory furnace or in the blast furnace plays no important part in the necessary reactions which take place in these furnaces, there is no reason, metallurgically, why electric heat may not be substituted for the heat derived from the combustion of carbon. In fact, as we have tried to point out, in some cases the reactions would take place to better advantage in the neutral atmosphere of the electric furnace than in the reducing or partially reducing atmosphere of the combustion furnace. Therefore, as to whether the electric furnace would be used for the smelting of copper ores would largely depend, so far as we are able to make out at the present time, upon the relative cost of coke and electric power.

"As the use of the electric furnace is not advocated as a competitor of the combustion furnace, but as a substitute for it in those localities where it is not advisable because of the high cost of fuel, we see no reason why the electric furnace may not be developed as a substitute for the combustion furnace, where the conditions are such as to warrant its use, especially in the treatment of copper-bearing ores. In this connection it is to be remembered that the reason why the electric furnace was developed in the iron industry, for the reduction of iron from its ores was due to necessity. As a matter of fact the field for the electric furnace in the reduction of iron from its ores is a limited one. Perhaps the name is true as regards the possible application of the electric furnace to the treatment of copper ores, but, judging from the comparative costs as shown in the preceding pages, it would seem that the chances in favor of the electric furnace for the treatment of copper ores are greater than those for the treatment of iron ores, because there is not so great a difference in the cost of coke and electric power in copper mining districts as in iron smelting centers. Also the cost of electric power is constantly becoming less, due to improvements in gas engines and steam turbines, so that, in districts where water power is not plentiful but cheap fuels unsuited to coking purposes are available, it may be found more advan-

tageous to use electric heat than the heat derived from the combustion of coke.

"It is sincerely hoped by the writers, that although it is possible at this time to present but few facts, that the comparative study herewith presented may serve to stimulate interest in the subject of the electric smelting of copper ores, and to cause others to attempt a further investigation of it. As a result of their investigations, the writers are convinced that experimental work on a larger scale should lead to the development of an electric blast furnace, which in some cases could be used to better advantage for smelting sulphide ores of copper than the combustion blast furnace."

Sunday—Helena

On Sunday, August 17th, luncheon was taken at noon in Helena and in the afternoon the East Helena lead-silver smelter of the American Smelting & Refining Company was visited.

Monday—Butte

On Monday in Butte the attendance had increased to 150. The morning session in the Auditorium was opened by an address of welcome by C. F. Kelly, Vice-President of the Anaconda Copper Mining Company.

Papers were then presented by R. H. Sales, D. C. Bard and M. H. Gidel, B. H. Dunshee, N. P. Brady, J. Gillie.

Geological and Mining Papers

The very elaborate paper of 100 pages, by **Reno H. Sales, Butte, Mont.**, discussed the geology of the "ore deposits at Butte, Montana." It is impossible to do justice to the paper in an abstract which would be necessarily short. In the discussion Grattan did not agree with the main conclusion as to the origin of the chalcocite Butte ore.

The paper by **D. C. Bard and M. H. Gidel, Butte, Mont.**, discussed "mineral associations at Butte, Mont."

The paper by **B. H. Dunshee, Butte, Mont.**, dealt with "timbering in the Butte mines."

The paper by **Norman T. Brady, Butte, Mont.**, discussed "shaft-sinking methods of Butte."

The paper by **John Gillie** dealt with the "use of electricity in the Butte district."

Excursions—Leaching Low-Grade Copper Ores

In the afternoon of Monday eight different parties were formed, which visited the electrically-driven air-compressor plants and air hoists at the High Ore Mine, the underground workings and pumps at the Original, Leonard, North Butte, and Tramway Mines, the underground workings and the zinc mill of the Butte & Superior Company, the mine and the copper-leaching plants of the Bullwhacker Copper Company and of the Butte and Duluth Mining Company.

Most interesting was probably the inspection of the Butte & Duluth Mining Company's leaching process for low-grade ores with subsequent electrolysis with insoluble anodes. The present output is over one ton of copper daily from a two per cent ore. This is to be increased to ten tons daily.

Those who saw the process were greatly impressed by its potentialities.

It seems not unlikely that it may revolutionize the treatment of carbonate and silicate low-grade ores.

Basic-Lined Converter

The first paper presented in the technical session on the evening of Monday was the paper by **Mr. E. P. Mathewson, of Anaconda, Mont.**, on the "Development of the Basic-Lined Converter for Copper Mattes."

The author gave a historical sketch of early unsuccessful experiments and outlined the later successful development of the basic-lined copper converter. It summarizes the main points to be observed for success as follows: "Not to exceed a temperature of 2100 deg. F.; not to have tuyeres smaller than 1.25 in. (1.5 in. is the preferred size); to drive in punch rods the full size of the tuyere opening, immediately after pouring copper; to maintain in the converter as large a mass

of matte and slag as possible to prevent sudden changes in temperature and overheating of the lining; to employ slag containing preferably about 25 per cent. of silica."

The following test was made at Anaconda with a view to finding out whether the cutting action of converter slag on a magnesia brick lining bore any relation to the silica content of the slag.

Thirteen slags were selected from the daily samples sent to the laboratory. These varied in silica from 22.4 per cent. to 37.8 per cent. and were carefully analyzed for MgO. The results given in a table in the paper show no relation between silica and magnesia content.

Mr. Mathewson concludes: "To Smith and Pierce belongs the credit of taking a long-discarded idea and developing it into a successful process. The great advantages of the process are: decreased cost of lining; the ability to use large units in converting, with consequent economies in labor, power and repairs; neatness and cleanliness of plant, abolishing the danger, from dust, to the health of the lining crew."

The paper was discussed by Dr. Crossdale and Dr. Richards.

Round Tables

A paper by Mr. Theodore Simons, of Butte, Mont., discussed "the evolution of the round table for the treatment of metalliferous slimes."

The author emphasizes the great importance which has been attached in recent years to the treatment of very fine sands and slimes. "For the profitable treatment of the very finest slimes, round tables have as a last resort proven more satisfactory and economical than other machines."

After discussing the general principles of "film sizing" the author outlines the evolution of the round table, beginning with the primitive dressing plant described in Agricola's *de Re Metallica*. From rectangular buddles he passes over to round tables and distinguishes round tables of the intermittent type (convex and concave types), and round tables of the continuous type.

Among those of the latter type the following are described in some detail: 1. Revolving tables with stationary fixtures (Evans). 2. Stationary tables with revolving fixtures (Linkenbach). 3. Percussion round tables (Bartsch). 4. Revolving round tables with adjustable inclination of the deck (Demuth).

While in all these tables the generatrix of the deck surface is a straight line, tests are being made at present at Great Falls with round tables, having a deck the generatrix of which is the arc of a circle, whose chord has an inclination from the horizon varying with the fineness of the pulp to be treated.

Tuesday (Anaconda)

In the morning of Tuesday the members and guests of the Institute were carried by a special train to Anaconda, where the forenoon was spent in an inspection of the Washoe smelter.

In the afternoon a technical session was held at the Margaret Theater, opened by an address of welcome by Mr. E. P. Mathewson, the reply being made by Dr. J. W. Richards. Papers were then presented by Ralph Hayden, Albert E. Wiggin, Robert Ammon, E. S. Bardwell, Edgar M. Dunn, James O. Elton, Robert H. Richards, and E. G. Woodruff.

Slime Concentration at Anaconda

The paper by Ralph Hayden, Anaconda, Mont., first discussed briefly what "Anaconda slime" is, its source and its constitution. "The problem of wet concentration of the Anaconda slimes may be divided into two parts, viz.: First, to prepare the slime for use on some concentrating machine by thickening it to the proper density, and at the same time recover water clean enough for reuse in the mill; and second, to recover the mineral values from the thickened pulp in a concentrate which can be smelted."

The author then passed over the experimental slime tank at Anaconda and described at some length the slime thickeners in use (slime ponds; tanks; comparative efficiency of open and baffle tanks; fallow tanks; Kuchs-Laist centrifugal separator, Garred filter; Dorr continuous thickener) and concentrators (round table; Peck centrifugal concentrator, competi-

tive tests; twenty-deck round table; slime feed distributor). Figures are given of the results of tests, often of several days' duration.

The final conclusions are as follows:

"It is evident that large settling areas are required to thicken the slime where free settling is to be used. If filters or centrifugal machines are resorted to, then space can be economized, but increased maintenance costs and operation costs, together with complex machines, enter into the problem.

"The Dorr tank has nearly the same capacity per square foot of settling area as the smaller units; namely, 0.20 to 0.25 gal. per square foot per minute; but it has the advantage of large units, say 50-ft. tanks, which can be built 3 ft. deep and three in a stand; and consequently requires few discharge plugs, and these of good size. The filter is a very compact and efficient machine, but has the disadvantage of operating in cycles as well as having high maintenance costs.

"The Kuchs-Laist separator is very compact and is continuous in operation, but requires power, and also a satisfactory discharge plug to give constant density. Its product is freer of colloids than that of the other thickeners, which allows concentrators to make a greater net recovery from it. If a uniform spigot density could be maintained this machine would be very satisfactory.

"The round table has proved very efficient for handling slime, it being simple in construction and operation, and a number of decks may be placed one above the other, thus economizing on floor space, where the mill height is available.

"The Peck machine has a good principle, namely, the utilization of the effect of centrifugal force, which allows it to handle a good tonnage in a small space. It makes a good recovery on the slime settled in the ponds.

"The best net recovery of copper in the original slime is from 50.0 to 52.0 per cent. except where the Kuchs-Laist machine is used. The slime contains about 20 per cent. of colloidal copper, which is not recoverable by wet concentration; and there are middling grains to be considered as well. Thus the net recovery of recoverable copper is probably 65 per cent., which is very good for wet concentration on this class of material."

Other Important Papers On Concentration and On Smelter Fumes

More papers followed on concentration practice in Anaconda and Great Falls. In view of the importance of these papers and on account of the limitations of space in this issue, an analysis of these papers is reserved for the next issue, and only the titles of the papers presented will be given here:

"The Great Falls System of Concentration." By **Albert E. Wiggin** (with chapters on the early history of the Boston & Montana mill at Great Falls from 1801 to 1905; the replacement of the Evans' jig by the Hancock jig; the Richards' pulsator classifier system; the Woodbury classifier and jig system, the Great Falls' System, and a comparison of the Great Falls' and Evans' systems at Anaconda).

"The Anaconda Classifier." By **Robert Ammon** (giving a sketch of this hindered-settling classifier and an account of the actual results obtained in practice with the classifier working on the Butte copper ores at Great Falls and Anaconda).

"Application of Hindered-Settling to Hydraulic Classifiers." By **Earl S. Bardwell** (giving data from practice on the question of proper ratio between the area of sorting column and the area of constriction).

"Ore Dressing Improvements." By **Robert H. Richards** (read by Mr. Bradley Stoughton).

The following two papers dealt with the problem of smelter fumes, and abstracts of them must also be reserved for our next issue:

"Determination of Gases in Smelter Flues, and Notes on the Determination of Dust Losses at the Washoe Reduction Works, Anaconda, Mont." By **Edgar M. Dunn**.

"Arsenic Trioxide from Flue Dust." By **James O. Elton**.

A paper by **G. E. Woodruff** on "Typographic Maps for

Mining Engineers" was also read at this afternoon session.

In the evening the members and guests of the Institute were taken by train to Butte, where in the Auditorium moving pictures showing the Montana power plants, etc., were greatly enjoyed. Not less enjoyable was the dance "indulged in by those who wished" at the Columbia Gardens later on.

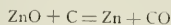
Wednesday (Butte)

In the Wednesday morning session, during which Dr. J. W. Richards was in the chair, papers were presented by Messrs. Johnson, Saunders, Roberts, Billingsley, Guinniss, Herewith, and van Barnevelt.

Reducibility of Metallic Oxides As Affected By Treatment

The paper on this subject, by Woolsey McA. Johnson, of Hartford, Conn., emphasizes the fact that the ease of reduction of metallic oxides depends largely on the way they have been prepared.

The temperature at which zinc oxide evolves zinc according to the equation



depends on the nature of the two reacting bodies.

In general, all equations reacting in the liquid or the gaseous phase have well-defined constant physical conditions which govern the equilibrium point and reaction velocities. But in a reaction where we have one or more solid phases, the temperature conditions requisite for the unbalancing of the stable system are found to vary. "These variations are due to the fact that in a solid there is a very complex molecule, and it is not a question solely of the tearing away of two interlocked atoms, but also of the destruction of a complex molecule containing hundreds, or possibly thousands, of smaller molecules.

"In other words, there are two forces to be overcome: A certain 'chemical' force, F_c and a certain 'physical' force, F_p . Theory indicates that F_c is constant and F_p is variable. F_p depends on the physical configuration of the small molecules of the metallic oxide."

The author described his "reaction tester" devised for such work, and gave some of the results obtained with it.

C.P. zinc oxide made by the wet method and soft 600 deg. C. charcoal gave a reaction temperature of 1022 deg. C.; 48-hour Indian territory coke gave with the same substance a temperature of 1029.5 deg. C.; with Acheson graphite turnings the reduction temperature was 1085 deg. C.

The same substance, C.P. zinc oxide, was treated by heating at definite temperatures for 12 hours with soft coke, and its reaction temperature was determined. The three values of C.P. wet-method zinc oxide are as follows:

Reduction Temperature

No previous treatment	1029.5° C.
1100° C. treatment	1048° C.
1300° C. treatment	1061° C.

From these and other results given in the paper the author deduced the general law: *The higher either of the reacting solid bodies is previously heated the higher is the reduction temperature.*

The author discussed briefly the rationale of this law, and then continued:

"Taking up the question of the changes in the nature of the zinc oxide as affecting reduction temperature, it might be well to consider zinc ferrite, zinc silicate, and zinc aluminate. All these have been made by the writer by heating Fe_2O_3 , SiO_2 , and Al_2O_3 mixed with molecular equivalents of zinc oxide. They are insoluble in ammonium chloride and dilute acids, and if strongly heated, insoluble in concentrated acids. They are also inert to reduction by carbon and have high reduction temperatures.

"We can, however, derive a useful lesson from these compounds. For, if zinc oxide can combine with metallic oxides as aluminum oxide and iron oxides, why can it not combine with another metallic oxide: to wit, zinc oxide? Accordingly, we can say that, when zinc ores are heated very strongly, 'zinc zincate' is formed, and that the energy of this compound must

be supplied to break it up before the reaction can proceed.

"The writer has found that these experimental facts agreed with the practical experience of himself and many others in the zinc business; for instance, it is a fact that 'spiegel oxide,' which is formed in the downcomers and in the hot-blast stoves of the New Jersey Zinc Co.'s spiegelbleien furnaces working on residues from the 'oxide' furnaces, and which is thus made at a low temperature, requires little heat in 'retorting,' whereas 'refuse oxide,' made in the Wetherill furnaces of the same company at a temperature vastly higher, requires a much higher temperature to 'retort.' 'Dead' coal, or coal from the strip pits of western Missouri, has been known to reduce zinc oxide much more easily than Arkansas semi-anthracite, as the tests show. It is natural to expect that the weathering action makes the carbon more active."

After discussing the initial temperature of reduction, the author deals with the rate of reduction. His concluding remarks are:

"Every item of experimental evidence and theoretical speculation indicates that the reducibility of zinc oxide, or any other metallic oxide, depends on the way in which the reducing agent and the material to be reduced are heat treated. Practically we find this to be the case in retorting and also in smelting in the writer's electric furnace. In the latter the reactions proceed in an entirely different manner, for the alternating current is a catalyzer and the slag of the smelting zone introduces new features besides, and only enough carbon is added to reduce the zinc oxide.

"Since researches prove that such practical changes as roasting at as low a temperature as will give a 'dead' roast increase the ease of retorting to such an extent that it is directly and distinctly sensible to the workman, it is but logical to reason conversely and to attribute to the practical judgment of the workman a substantial degree of accuracy. Now, any practical zinc man knows that ores from one mine will 'retort' easily and ores from another will 'retort' slowly. So it is hardly to be doubted that an investigation of the reduction temperature of zinc blende roasted under standard conditions, and with a standard reducing agent, in a Johnson 'reaction tester' would determine the geological history of such mines. The writer knows little about geology, but he is sure that blende deposited primarily would give when roasted a reduction temperature 10 deg. C., and possibly 20 deg. C., higher than a blende deposited in the same mine secondarily and tested under same conditions."

Blast-Furnace Jackets

A paper by Robert P. Roberts, Great Falls, Mont., on the "Thermal Effect of Blast-Furnace Jackets," gives an account of quite an elaborate test of the water consumption in these jackets and the heat thereby carried away. An analysis of these results must be reserved for a future issue.

The following figures were also presented at this session: Rock Drill Economies. By William L. Saunders. The Southern Cross Mine, Georgetown, Mont. By Paul Billingsley. Occurrence and Manufacture of Refractories in Montana. By Guinniss. Mining and Metallurgy at the Panama Exposition. By C. E. van Barnevelt.

The afternoon of Wednesday was again spent in various excursions in the neighborhood of Butte.

Concluding Technical Session

The concluding technical session was held on the evening of Wednesday, when papers by Febles, Burns, Corwin and Rogers, Krejci, Bardwell, Nordberg were presented, abstracts of which are herewith given.

Precipitation of Copper from Mine Waters

A paper by John C. Febles on "the precipitation of copper from the mine waters of the Butte district" gives first, a historical sketch.

The principal reaction which occurs is the precipitation of metallic copper at the expense of metallic iron, is



The resulting ferrous sulphate becomes oxidized by constant

aeration in the flumes and towers to ferric sulphate, which, being subjected to still further oxidation, is thrown out of the solution, both as basic ferric sulphate and as ferric hydrate, with the liberation of free acid, which reacts upon either copper or iron salts, again forming the corresponding sulphates.

The quality as well as the quantity of the cement copper obtained in a given time depends upon a number of conditions, which may be briefly enumerated as follows:

1. The amount of copper in solution.
2. The temperature of the solution.
3. The freedom of the solution from salts other than copper.
4. The rate of flow.
5. The kind of precipitating apparatus used. (Flumes are superior to towers in almost every way.)
6. The kind of iron used. (Almost any merchant iron or mild steel is suitable; old rails, merchant bar, and pipe of small diameter seems to be the best.)
7. The size and shape of the iron, and the way it is placed.
8. The proper handling and cleaning of the iron.

The author then describes the High Ore plant and its equipment. It produces approximately 2,200,000 lb. of pure copper annually, which amounts to the shipping of about ten cars of crude precipitate each month.

The iron consumed in the process weighs from 1.1 to 2 times as much as the copper produced. The lower the value in copper, or the higher the ferric sulphate or free acid, the greater the consumption of iron. Good, clean, compact iron will precipitate more copper than tin cans and similar light iron. At the Silver Bow plant, where a large percentage of railroad rails and similar desirable iron is used, 1700 lb. of copper recovered per ton of iron used is claimed. A fair estimate for normal conditions would be 1 lb. of copper recovered for each 1.5 lb. of iron consumed.

The author then discusses the other products precipitated. The presence of the ferric hydrate, or "ocher" in the precipitate is objectionable, and is to be avoided as far as possible. There does not seem, however, to be any satisfactory way to control its precipitation. The formation of this precipitate is due to oxidation of iron in solution, probably by aeration, which results in its being thrown down both as ferric hydrate and as basic ferric sulphate.

The mine waters of Butte also contain small amounts of arsenic, part of which is precipitated with the copper, although not proportionally with the copper.

An average of cost figures, extending over one year's operation of the Leonard plant, is as follows:

	Per lb. of Copper Produced
Labor	\$0.0352
Supplies and iron	0.0140
Sundries	0.0002
	\$0.0494

The above figures are upon the basis of the net pounds of copper contained in the precipitate shipped and paid for.

The author concludes with a brief description of the methods of protection of the pumps and water columns from the acid water.

The water ends of all pumps are constructed, as far as possible, of a phosphor-bronze alloy analyzing approximately 85 per cent of copper and 15 per cent of tin. Experience has shown that this alloy will stand the corrosive action better than any material as yet tried.

Pumpers and valves made of this bronze are not altogether proof against the action of the water, however, having to be replaced from time to time. The corroded parts are, upon removal, sent to the machine department for repairs, a large supply of interchangeable parts being kept on hand.

The columns are constructed of cast-iron pipe, lined with either lead or wood. The lead-lined pipes are flanged, the lead lining flaring, so that when bolted together, with a rubber gasket between, a water-tight joint is insured, and the iron casing is completely protected from the water flowing through.

The wood linings are made of narrow strips, the last strip being driven firmly home, forming a key, which holds the lining in place. The whole inside is then painted with acid-proof paint, and all joints are carefully calked and painted to prevent leakage. The exterior of all pump columns is protected with acid-proof paint.

In the discussion W. T. Burns gave a brief account of an attempt made at the Great Falls electrolytic plant to recover the copper from the mine water by direct electrolysis. His conclusions are that the recovery of the copper from the mine water by direct electrolysis does not appear to be possible from a commercial standpoint, chiefly on account of the low electric conductivity of the mine water, resulting in a high power consumption.

Electrolytic Refining of Copper Precipitate Anodes

A paper by W. T. Burns gives "notes on the electrolytic refining of copper precipitate anodes."

Attempts were made in 1908, at the Great Falls Works, to produce ingots direct from the Butte precipitate by smelting the material in a reverberatory refining furnace. The ingots produced in this manner average 99.4 per cent of copper and 0.40 per cent of arsenic and antimony.

The copper content being considered too low, it was decided to cast the copper into anodes and to treat the anodes in the electrolytic plant.

Details are given of the results of the treatment and the conclusions are summed up as follows:

Cathodes of wire-bar grade can be economically produced from copper precipitate anodes when treated at a current density of 17 to 18 amp. per square foot.

Cathodes, from which cake and ingots averaging 99.85 per cent of copper can be made, can be economically produced from copper precipitate anodes at a current density of 33 to 35 amp. per square foot.

The removal of the arsenic from the electrolyte is the principal item of electrolytic refining cost that is in excess of the cost to refine ordinary converter anodes.

It would then appear that if anodes can be produced as cheaply in the refining furnace as the precipitate can be treated in the smelter, there is little choice between the two methods, as the copper must eventually pass through the electrolytic refining.

MacDougall Roasters

A paper by Frank R. Corwin and Selden G. Rogers on "increasing the efficiency of MacDougall roasters at the Great Falls Smelter of the Anaconda Copper Mining Company," points out that since the first installation of MacDougall roasters at the Great Falls smelter of the Anaconda Copper Mining Company, the capacity of the furnaces has been more than doubled. During the first nine or ten years of the operation of the MacDougall department the tonnage treated by the furnaces remained practically unchanged. Experimental work carried out in 1906, however, showed that increasing the draft of the furnaces would bring about an increase in capacity. In 1909 the MacDougall department was connected to the new smelter stack and flue system, and the stronger draft caused by this change increased the capacity of the furnaces. The change in draft conditions made it necessary to operate differently, and systematic experimental work was started to adapt the furnaces to the new conditions.

This experimental work extended over a period of several years, and resulted in very largely increasing the capacity of the MacDougall furnaces, decreasing the percentage of flue dust, and also brought about other changes and improvements tending to raise the efficiency of the MacDougall department.

These changes, of course, resulted in reducing operating expenses. The improvement was brought about without building any new MacDougalls or enlarging the old ones, and was the result of the thorough study of the MacDougalls and numerous tests.

The paper gives a detailed account of the experimental work by which the increase in efficiency was brought about. An

analysis of this work must be reserved for a future occasion.

The following papers were finally presented:

The Metaline Plant of the Inland Portland Cement Company, Metaline Falls, Wash. By **Milo W. Krejci**.

Notes on the Metallography of Refined Copper. By **Earl S. Bardwell**.

The Compressed Air System of the Anaconda Mining Company, Butte, Mont. By **Bruno Nordberg**.

The program for Thursday included an all-day trip to the Southern Cross Mine and a banquet at the Silver Bow Club in the evening.

A magnificently illustrated pamphlet prepared by the local committees for this convention deserves special mention. The Montana section of the Institute was officially established by the directors in their session of Wednesday. The whole convention was greatly enjoyable, highly important, and most successful.

Synopsis of Recent Chemical and Metallurgical Literature

Iron and Steel

Electrolytic Prevention of Corrosion.—A preliminary report has been issued by the Bureau of Mines, *Technical Paper* No. 15, giving results of experimentation in electrolytic methods of preventing corrosion of iron and steel. The Bureau has undertaken a thorough investigation of the problem, and this report is merely an account of experiments made looking toward the development of a satisfactory method.

Experiments begun in 1911 by **Mr. F. M. Stanton**, then chemist at the experiment station in Pittsburgh, consisted in exposing steel plates in separate vessels in— H_2SO_4 . In one

vessel the steel plate was connected with the negative pole of a storage battery, the positive pole being connected with a carbon rod immersed in the solution near the steel plate. In another vessel the steel plate was left unprotected by any current. After 24 hours' exposure the unprotected plate had lost 1.81 g. weight, and the protected plate only 0.0012 g. In tests with plates submerged in the Monongahela river, the current density was less than in the laboratory experiment, and the protection against corrosion was less complete. The current was furnished by a 2-volt accumulator. The loss in 15 days was 0.7 g. for the protected plate and 10.9 g. for the unprotected. Mr. Stanton's experiments were not completed, and further work was undertaken by Messrs. **A. E. Hall, J. K. Clement and L. V. Walker**.

Their first experiments were not satisfactory, but indicated that a definite current density must be maintained in order to reduce corrosion to a minimum. Subsequent experiments showed that a current density of 0.4 milliampere per sq. in. would prevent 93 per cent to 94 per cent of the corrosive

action of— H_2SO_4 , when the solution is not stirred.

In experiments with stirred electrolyte, plates having a current density of 0.4 milliampere per sq. in. or less continued to lose weight during the entire period of experiment. Plates protected by current densities greater than 0.4 milliampere showed a marked initial corrosion, but thereafter no further loss in weight.

To determine the effect of oxygen in the electrolyte, experiments were made in which the solution was saturated with oxygen, and the results showed that corrosion was rapidly accelerated by increasing the amount of that element in the electrolyte. Corrosion increased also with the increase in the rate of stirring the electrolyte. Experiments to show the effect of acid concentration show that although a considerably greater current density would be required, the method would

be effectual even in— H_2SO_4 . With— acid the

results were practically identical, showing that between these limits of acid concentration there is no change in the rate of corrosion.

Gold and Silver

Vacuum Filtration in New Zealand.—An elaborate article on the plant and operating results with the Moore process at the Victoria mill of the Waihi Gold Mining Company, New Zealand, is published by **Mr. William Macdonald**, in the May, 1913, *Journal of the Chemical, Metallurgical and Mining Society of South Africa*. Filter presses were formerly used at this mill, but on account of their cumbersome nature and high cost of operation they were displaced in 1908 by the Moore filter.

A brief statement of the metallurgical process at the Victoria mill follows: The ore is crushed in water, amalgamated, and separated into sand and slime products, the former being leached. The slime is thickened by settlement, dewatered by a Moore filter, mixed with cyanide solution, agitated and again filtered by the Moore process. The slime is very finely divided, 91.33 per cent passing a 200-mesh screen. It contains about 1 per cent pyrite.

A complete cycle of filtration occupies 144 minutes, distributed as follows: Loading, 30 minutes; washing, 60; discharging, 24; transfer of baskets, 10. With a partial vacuum rising to 24 in., and a pulp of 1.15:1 consistency, a 1½-in. cake is formed in about 50 minutes. The routine work of the plant proceeds smoothly when the pulp is of the proper consistency, but with a thin pulp is often the cause of trouble. Thin pulp may be due to several causes, but probably is accounted for by poor settling in the collecting tanks.

Filter cloths have a life of 12 to 15 months. It is customary to place new cloths on the cyanide filter for three months, then remove and treat with a 5 per cent hot solution hydrochloric acid, and, after washing and drying, use again on the cyanide filter for three months. After the second removal and acid treatment the cloths are used on the dewatering filter for six to nine months.

Daily assay is made of discharged cakes. A sample is taken from every other leaf of each basket throughout the 24 hours. After cutting to about 20 lb., the slime is wrapped in clean filter cloth and squeezed between plates by screwing up bolts. The solution thus obtained is assayed, using 20.000 gr. The squeezed pulp is then broken up and washed by decantation, and finally assayed in triplicate, taking 1000 gr. for each charge. The amount of soluble gold left in the discharged cakes is less than 1 gr. per ton of dry slime. The cake is discharged with 30 per cent moisture. Filtering capacity per sq. ft. filtering area is about 10 lb. On a 144-min. cycle the maximum number of charges per day per basket is 10.

For the year 1911 the average assay of cake as discharged was 0.508 dwt. gold and 0.768 oz. silver per ton; average assay of washed cake was 0.55 dwt. gold and 0.746 oz. silver per ton. The extraction was 80.8 per cent of the gold and 71.5 per cent of the silver. The efficiency of displacement is given as approximately 90 per cent. The cost per ton of dry slime treated during 1911 was as follows: Labor, 5.54 cents; power, 2.88; supplies, 1.14; assaying, 0.4; total, 9.96 cents. The average cost of power from producer-gas plant was 15.974 cents per hp-day.

In conclusion it should be remembered that the process in use at this mill differs from the most modern practice in crushing in water, thus necessitating dewatering before cyaniding, and in relying on settling tanks for producing thickened slime.

Zinc and Lead

Bisulphite Process.—The British Metals Extraction Co., Ltd., is using the bisulphite process on zinc-lead ores at Llansamlet, Wales. According to a report made at the last annual meeting of the company the process is conducted as follows: The ore is pulverized and roasted slowly in mechanical furnaces. The sulphurous gases are conducted from the furnace to towers in which the extraction is made. The roasted ore is mixed with water to form a thin pulp which is

allowed to descend through the towers, meeting and mixing with the ascending gases. The interior construction of the towers is such as to cause intimate and effective contact of the gas and pulp. The solution of zinc bisulphite thus formed is separated and heated in a special apparatus, forming a precipitate which is calcined, yielding zinc oxide. The latter is then smelted, producing spelter assaying 99.85 per cent. zinc. No difficulty is experienced in the smelting operation. The extraction of zinc is reported to be 84 per cent to 89 per cent, depending on the efficiency of the roast. The lead residue contains about 8½ per cent zinc, and is smelted.

The Development of a New Optical Pyrometer

By C. G. Fisher

Since the introduction of Wedgwood's pyroscope in 1782, the matter of high-temperature measurements has received the attention of some of the most competent scientists of their times. Many valuable improvements in all forms of pyrometers have been made, but only those relating to the optical pyrometer are referred to in this article.

The goal toward which all manufacturers of pyrometers have been striving was to produce an accurate instrument which would be suitable for use in the industrial plant.

This requires an instrument which is not cumbersome, not too delicate to stand the ordinary usage under the conditions as they exist, an instrument which will not deteriorate rapidly when being subjected to the necessary operations, one which also could be satisfactorily handled by the workman of average intelligence. Each of the improvements in the optical pyrometer has been a step in this direction until the very latest improvement, the first announcement of which is this article, brings it into the position where it fulfills the requirements of a practical temperature-measuring instrument for use in the manufacturing plant.



FIG. 1.—WANNER PYROMETER

The general type of instrument which has made the greatest strides is the one based on Wien's law, utilizing the photometric measurement of light radiation of a given wave length of a definite portion of the visible spectrum. The Wanner and the Scimatco instruments are of this type and this is the type referred to here.

Wien's law, which expresses the physical principle upon which this type of pyrometer is designed, has been checked experimentally throughout the widest ranges of temperature and has received the most convincing verification.

In this instrument there are two adjacent fields of vision, one being illuminated by a standard incandescent lamp and the other by the object whose temperature is to be measured. This latter field must be adjusted to the same intensity as the standard field, and this is accomplished by turning the eye-piece.

It is generally understood that the optical pyrometer is not a low-temperature measuring instrument, its range beginning at about 650 deg. C. But as a high-temperature instrument it has no equal, and is now obtainable for taking accurate readings up to 7000 deg. C.

The first optical pyrometer of this type which has been used to any great extent is the one designed by Professor Wanner in Germany (Fig. 1). This pyrometer has the good points of the instruments of this type, such as accuracy, high range of temperature, etc., but it also has objectionable features, the most conspicuous being a separate table to convert readings on the dial to degrees of temperature, incandescent lamp unprotected against breakage and easily thrown out of adjustment, and no protection against dust in the factory, which interferes with its proper working. The metal slit which admits the light may change in size, due to the expansion of the metal when subjected to the heat in front of the furnace. The instrument is confined to use where the object, whose temperature is desired, is under "black body" conditions.

It would be proper to mention here just what is meant by "black-body" conditions. It is known that different substances

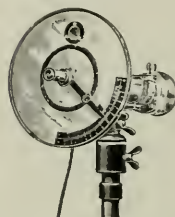


FIG. 3.—FURTHER IMPROVEMENTS IN CONSTRUCTION



FIG. 2.—DIRECT-READING TEMPERATURE SCALE

It is interesting to trace the successive improvements in the optical pyrometer, each one of which corrects a feature which interfered with the practical application of the instrument.



FIG. 4.—HOOK DIAL WITH OPEN SCALE

at the same temperatures when in the open, radiate different amounts of light, but should they be so situated that they are almost entirely surrounded by heated walls of uniform tem-

perature, as in a furnace, they then radiate the same amount of light, and are said to be under "black-body" conditions. The light radiation from a black body is due to temperature alone, and the physical principle on which the optical pyrometer was designed required that the object be under such conditions as make it a "black body."

The next important improvement was made by Dr. G. A. Shook in 1910, when he worked out the first direct-reading scale of temperature which was attached directly to the instrument, thus doing away with a separate reference table (Fig. 2). It might be well to mention that it has not been generally known that the credit for this improvement belongs to Dr. Shook, and such credit has never before been given him publicly.

This made the optical pyrometer the quickest reading instrument, as no time is required for the instrument to come up to temperature, and it requires only a few seconds to make the observations.

The question has been asked by many, "Is there not a great personal factor in matching the colors of the two fields?" having in mind, perhaps, the difficulty and accompanying complications of taking a piece of goods to town and buying a spool of silk to match. The matching of the two fields of vision is a great deal more simple, as the fields are adjacent, and as long as there is a difference in color, a line is visible between them, but just as soon as they match and one blends into the other the line immediately disappears. The operation, therefore, is simply turning the eye-piece of the instrument until the line disappears between the two fields, and as both are observed by one eye-piece and at the same time, any defect in sight is equalized.

Many experiments have been made with large numbers of different persons taking readings for the first time, and with the simple instruction to eliminate the line, the differences in readings were small, usually not more than 10 deg. between the highest and the lowest.

The next part of the pyrometer to be improved was the construction; the lamp and the wires leading to it were protected and the slit admitting the light was kept at a constant temperature so that its change of size due to expansion of the metal was eliminated. Dust also affected the size of the small opening and interfered materially with the operation of the instrument, and this was corrected by inserting a piece of special glass with known optical properties (Fig. 3). A much larger dial was designed for the instrument, allowing a larger temperature scale to be attached, which permitted more accurate readings, the dial also acting as a shield to protect the eye against outside influences. The whole instrument was now enclosed in a steel protecting tube which made it very substantial. This Scimatco pyrometer was adopted extensively by industrial works and used by workmen controlling temperatures in many different operations.

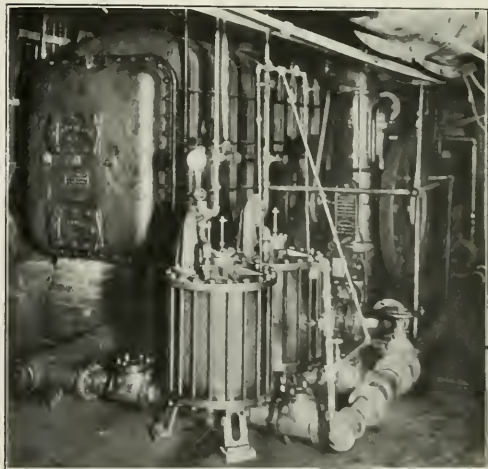
Dr. G. A. Shook of the University of Illinois has just perfected the latest improvement, a new optical pyrometer with a Shook open-temperature scale for the correct readings of temperatures of materials when not under "black-body" conditions. This new instrument is a universal one, having a scale for furnace temperatures in addition to the scale for open temperatures, and will, therefore, take the temperature of steel in the furnace, or while being poured from the ladle, the temperature of a billet, or a shape, while passing through the rolls, etc. This new pyrometer with the Shook open-scale has been based on the emissive power of light of different substances, calculated for the wave length, as used by the Scimatco instrument, and has been checked and found entirely correct by a number of scientists. The dial with the two scales is shown in Fig. 4. The first open-scale perfected was the one for iron and steel. Scales for the other metals came later, a different scale being required for different materials.

Dr. Shook has protected his new invention by applications for patents, all of which have been assigned to the Scientific Materials Co. of Pittsburgh, who have put the instrument on the market.

A Novel Application of Steam Traps for Evaporating Installations

For the last six months there has been an experiment tried out in the power plant of the Jacob Ruppert Brewery, New York, the successful results of which are interesting to all managers and engineers in factories where evaporators working under vacuum are used.

Part of the power plant of the brewery is run condensing, the condensing outfit consisting of two 1200-hp. Wheeler condensers, each equipped with a single Edwards air pump and an efficient oil separator on the inlet. The condensation from these condensers is intended to be used in the ice plant for making distilled water ice. By examining the adjoining illustration it will be noted that the condensers are located close to the floor in the basement of the engine room, the actual distance from the floor being 30 in. Now, instead of the air pumps delivering water suitable for ice, an emulsion was delivered which consisted of the condensed water and the air and non-condensable gases churned together in the pump. The problem was to overcome this condition. This was done in the following manner:



STEAM TRAP

The air pump, at a slight expense and without disturbing the installation in any way, was turned into a dry vacuum pump, which removed all the air and non-condensable gases. A connection was made to the bottom of the condenser and led to two special vacuum traps designed and built by the Lytton Manufacturing Corporation, which removed the condensation from the condenser and delivered it to the ice plant. On account of the air and gases being taken out by the air pump the clear condensation flowed to the trap quietly and instead of being churned up was delivered in the same condition to the ice plant.

After the installation was properly installed the vacuum on the condenser was increased to 28 $\frac{1}{2}$ in. The distance between the highest point to which the water rises in the trap, which is shown in the foreground of the illustration, and the bottom of the condenser is 6 in., and with this head two traps are able to handle a maximum amount of 400 lbs. per minute from the vacuum of 28 $\frac{1}{2}$ in. This is done without any heating of the condensation or re-evaporation, because the time it takes to equalize from a 5-lb. pressure to a 28-in. vacuum with the trap is through discharging is only three seconds.

The construction of the trap is a vertical cylinder with inlet and outlet at the bottom and one of the regular Lytton steam and vent valves installed on top and outside where it is get-

at-able. On the inside of the cylinder is a floating piston of a neat design, which operates the steam valve.

These traps are automatic and adapt themselves entirely to the amount of condensation flowing to them. They need no oiling, no packing, no regulating and very little attention. The two shown in the cut have a 4-in. inlet and outlet with a maximum capacity of ten tons of condensation per hour each.

These traps are ideal machines to locate on the drips of the steam chambers of multiple effects in place of the troublesome vacuum pumps, on account of their simplicity and automatic operation. They require very little space and a minimum foundation.

This trap is known as the Lytton perfect vacuum and lifting traps, and is made in sizes from 1 in. to 6 in. with capacities from $\frac{1}{2}$ ton to 25 tons per hour, and manufactured by the Lytton Manufacturing Corporation, of Franklin, Va., with sales office at 50 Church street, New York.

The experiments which were so successful at the ice plant of the brewery were conducted under the direction of Mr. W. E. Parsons, Consulting Engineer, by the Chief Engineer of the Brewery and Mr. J. W. Lytton, President, and Mr. D. J. Lewis, Jr., Sales Manager, of the Lytton Manufacturing Corporation.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

REFINING METALS (Continued).

426,789, April 29, 1890, Moses G. Farmer, of Eliot, Maine.

Relates to an apparatus for depositing copper ingots from copper matte or other impure copper, used as anodes, upon a revolving drum having a polygonal depositing surface, and removable insulating separators along the angles between the several facets so that square ingots of copper will be formed thereon. The drum is intermittently rotated by a pawl and ratchet device. The drum cathode is made of a lead-antimony alloy, having a smooth cylindrical inner surface by which it is supported upon a roller in a suitable electrolyte. The removable insulating separators provide for the deposition of a continuous band of copper uniting all the ingots, to prevent them from falling off; the ingots may be readily removed by cutting this band.

465,525, December 22, 1891, Edward S. Hayden, of Waterbury, Conn.

Relates to an apparatus for the deposition of metals, and resembles the apparatus used in the process covered by his prior patent No. 377,487. The apparatus consists of a number of tanks, each having grooved vertical pieces of wood on the sides to permit the insertion of crude copper plates, leaving a space underneath the plate for the circulation of the electrolyte. The anode, of the metal to be treated, is at one end of the tank, and the cathode at the other, the intermediate plates being secondary electrodes. The electrolyte circulates through the tank, or tanks in series if electrically connected in series, and in multiple if electrically connected in multiple. The electrolyte may be sulphate of copper. A storage tank for the supply of solution is provided with a steam coil to regulate the temperature of the electrolyte.

467,350, January 19, 1892, Otto Stalman, of Anaconda, Mont.

Relates to apparatus for refining metals, and consists of two concentric tanks having insulating material, such as asphaltum, between to prevent leakage. The electrodes are made of a plate of crude metal, serving as an anode, a layer of insulating material, and a superimposed sheet of pure metal serving as a cathode and electrically connected to the crude metal. These compound electrodes are used as secondary electrodes, between an anode of crude metal, and a cathode of pure metal in the tank above noted. The crude metal from one compound

electrode is dissolved and deposited upon the pure metal sheet of the adjacent compound electrode. Sulphate of copper is used as the electrolyte. The secondary electrodes are suitably supported in the tank, leaving a space below for the collection of slimes.

467,484, January 19, 1892, Otto Stalman, of Anaconda, Mont., assignor to Marcus Daly, of same place.

Relates to process and apparatus for refining copper, and consists in detachably connecting an anode of crude copper to a cathode of pure copper, suspending a plurality of them as secondary electrodes in a vat containing sulphate of copper. A pair of connected elements are separated by insulating knobs, such as glass, from adjacent pairs. Instead of connecting the two plates, they may be separately suspended, and then short-circuited by wires. The electrolyte is maintained in circulation.

484,416, October 18, 1892, Charles R. Fletcher, of Boston, Mass.

Relates to an apparatus for electrolytically refining metals, and consists in clamping plates of sheet copper to serve as cathodes, to plates of crude metal, as anodes, and using the compound plate as a secondary electrode. The anode is cast with curved edges, its attached cathode is both longer and wider than the anode, and also has edges which curve in the same direction as those of the anode. When placed in the electrolytic vat, the compound plate is so inclined to the vertical that the upper face of the plate is the anode, and its under face the cathode. By using such cathodes, so disposed as to the anode of an adjacent plate, it is stated that the "growths," or "warts," which form on cathodes are avoided. Electrolyte is squirted over the surface of the anode from a series of jets having flattened nozzles, in order to wash off the impurities; this is said to reduce the liability of the anode-surface to becoming honeycombed.

485,618, November 8, 1892, Franklin Farrel, of Ansonia, Conn.

Relates to the refining of crude metals by electrolysis, using the secondary electrode process. The electrodes are not supported in grooves in the side of the tank, but are preferably suspended by tanks at the top of the electrode resting in hooks of insulating material. By this means all the crude metal is dissolved, leaving none to be remelted, thereby avoiding the expense of continually reworking a certain percentage of the metal.

489,632, January 10, 1893, Frank Gruessner, of Chicago, Ill., assignor of one-fourth to Francis B. Badt, of same place.

Relates to regenerating electrolytes used in refining copper. In time the copper sulphate solution becomes impure from the presence of arsenic, antimony, bismuth, and the like, originally in the crude metal. It is stated that suitable quantities of metastannic acid, if boiled with the electrolyte, will combine with and precipitate the arsenic as an insoluble salt; and also that it probably removes some other foreign substances.

Book Review

The Colloidal and Crystalloidal State of Matter. By Dr. Paul Rohland, Professor at the Technische Hochschule, Stuttgart. Translated by W. J. Britland and H. E. Potts, M. Sc. 12 mo (12 x 19 cm.), 54 pages. Price \$1.25 net. New York: D. Van Nostrand Company.

The book attempts to give broadly and from a most general standpoint, the essential modern ideas in this field of research and speculation. The attempt is crude; many statements are too strong, others too indefinite, others almost meaningless. Part of this may possibly be due to literal translation, a following too slavishly of the literal German order of expression of ideas, but most of what is complained of is certainly inherent in the original. The publishers co-operated toward the weak features of the book as far as they could—by taking on 39 pages of advertisements of their publications to a book containing only 54 pages of text.

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Colorado Meeting of the American Electrochemical Society

In descriptions of the Cripple Creek railroad trip it is said that this trip "bankrupts the English language." With equal truth the same may be said of Colorado's overwhelming hospitality extended to the American Electrochemical Society at its meeting last month. A wonderful program had been arranged by the Society's Colorado friends, with the first meeting in Denver, the second at the University of Colorado in Boulder, the third at the School of Mines in Golden, a final session on top of Pike's Peak, an excursion to Cripple Creek, and a delightful series of social functions, sight-seeing trips, and visits to chemical and metallurgical works. Nothing but the "envy of the gods" could spoil this program, and when they sent a downpour of rain when there should have been a picnic dinner on Lookout Mountain, the ascent of the mountain was spoiled, but not the picnic dinner and not the enjoyment. In every other detail the program was carried out as projected, and those who attended the meeting will never forget it.

It was not the largest meeting, but it was the most spirited, the most lively, the most strenuous meeting the Society has ever held. This is true in every respect, not the least true with respect to the technical sessions. There were lively discussions and any casual onlooker would quickly have become aware of the fact that the Society is anything but a mutual admiration society. But in spite of freely expressed criticisms and spirited controversies, there never was a trace of personal ill-feeling, but due respect for the other fellow's personality. In the case of three or four papers, dealing with special phases of the relation of electrochemistry to metallurgy, the discussion lasted almost an hour. There would have been no possibility of finishing the program in the time allotted to the technical sessions if all the papers had not been printed in advance. Fortunately this is now a firmly established rule with this Society. As a detailed report of the proceedings, with abstracts of the papers, is given elsewhere in this issue, there is no necessity here to go into detail.

Certainly this meeting must have a lasting effect of far greater importance than would appear probable at first sight. The human reaction between the Western metallurgist and the Eastern electrochemist was vigorously started. This is the main thing. And what real use is there of a national scientific or engineering society but to act as the catalytic agent starting just such a human reaction? The spirit exhibited in the discussions in open sessions and outside of them indicates that the reaction velocity will be high. Of course the reaction must obey the laws of chemical mass action, and is reversible. As to what electrochemistry can do for the West, Western metallurgists have expressed their needs, hopes, and doubts; and time will show the outcome. As to what Western metallurgy

can do for Eastern chemistry, nothing was more suggestive than the inspection of the two cyanide mills in Colorado Springs and Victor.

Good luck to the Colorado Section of the American Electrochemical Society founded on top of Pike's Peak. It was founded not to detract from, or compete with, other societies. It has a glorious mission of its own.

Accidental Coincidences

A few weeks ago the *Engineering and Mining Journal* pointed out that during August there were, partly overlapping in their dates, the meetings of the American Institute of Mining Engineers at Butte, Mont., of the Lake Superior Mining Institute at Duluth, Minn., and the International Congress of Applied Geology in Canada; many persons who would have liked to attend all three of these meetings were unable to do so; it was urged that there ought to be a clearing house for society meetings, some central organization which might be consulted in the arrangement of dates, with the idea of avoiding coincidences.

This unfortunate condition of accidental coincidences now appears to become almost epidemic. In September on identically the same days the American Chemical Society held its meeting at Rochester, the American Electrochemical Society its meeting at Denver. This was purely an accidental coincidence. Probably it affected but little the attendance at either meeting. But, as a matter of principle, such a thing should be impossible.

In October there will be an overlapping of the meetings of the American Institute of Metals in Chicago with the iron and steel meeting of the American Institute of Mining Engineers to be held in New York on Oct. 16 and 17, and to be concluded by a banquet on the evening of Oct. 17, and on this same evening the New York section of the American Electrochemical Society will open the season with a meeting for which an excellent program has been arranged that should attract many iron and steel men.

Clearly something ought to be done. If a clearing house for society meetings can be created, let us have it as soon as possible. If any of our readers has a practical solution of this problem to offer, it will be welcome.

Handling Slime at Anaconda

As the result of a very elaborate series of experiments, it appears that the Anaconda Copper Mining Company has reached a conclusion as to the best method of handling the slime produced at the Washoe concentrator. We might almost use the term tentative conclusion, for any process adopted in this day represents only the present state of the art, and is subject to modification as further advances are made. As stated by Mr. Ralph Hayden, at the Butte meeting of the American Institute of Mining Engineers, the Anaconda problem is, briefly, to thicken daily about 20,000,000 gal. of slime pulp containing from 2 per cent to 3 per cent solids, producing simultaneously clear water for the subsequent concentration process, and a pulp containing from 10 per cent to 15 per cent solids. The second part of the problem is the concentration of the thickened pulp.

In the final decision as to the best method of accomplishing the desired end, two features stand out prominently: first, the adoption from the cyanide industry of a machine

originally developed for the thickening of gold-bearing slime for agitation and filtration; and, second, the improvement and adaptation of one of the very oldest types of concentrating machines. Hitherto it has been considered impracticable, in work of the magnitude which obtains at Anaconda, to thicken slime and recover water by any other means than settling ponds of large area. The recent series of articles in this journal by Mr. H. N. Spicer showed this to be a world-wide practice; and indeed it has been the custom at Anaconda. Mechanical contrivances have been left out of consideration as being of inadequate capacity.

It, therefore, becomes of considerable interest in the realm of concentration, and particularly of slime concentration, to know that the Anaconda tests determined the use of the Dorr continuous thickener and the buddle, respectively, for preparing and concentrating the slime pulp. The thickening plant will be of special design to meet the particular needs at Anaconda; and the buddles will be twenty decks high to provide maximum capacity on a small ground area.

This brief recital of fact should be suggestive to men engaged in any branch of industry. It shows how the developments made to meet the peculiar needs in one branch may be adapted to the requirements of another. The recovery of clear water from 20,000,000 gal. of thin slime at Anaconda suggests, for example, the possibility of using mechanical means, instead of settling basins, for the preliminary clarification, at least, of some municipal water supplies, especially in those cities situated on muddy rivers. If only 90 per cent of the suspended solid was thus removed, there would be a saving of much labor now expended on filter beds.

The work at Anaconda also emphasizes a point frequently suggested, that in our progress we sometimes find it advantageous to revert to the machines and practices of an earlier date. The buddle is an old device, considered by some as obsolete and a relic of past practice; but here we find it rehabilitated to perform a useful function.

The Trend of Tube Milling

Comparison of the monthly reports of the Transvaal Chamber of Mines emphasizes the increasing importance of the tube mill in the equipment of a Rand cyanide plant. According to the figures for last June, the Consolidated Langlaagte has 100 stamps and ten tube mills, the ratio of 10:1 being the lowest of any installation reported. The highest ratio was 90:1 at the Durban Roodeport, but there were several instances of such ratios as 12, 16, and 20 to 1. The effect of this increased number of tube mills is noticeable in the daily stamp duty. At the Consolidated Langlaagte, a twenty-four-hour duty of 21½ tons per stamp was recorded; and while other factors obviously enter into this high stamp duty, such as weight of stamps and mesh of screens, we see a tendency toward much higher stamp duties by reason of the use of the tube mill as a secondary grinder. As viewed by the *South African Mining Journal*, the tendency on the Rand is toward heavier stamps, coarser screens and more tube mills per stamp.

It is apparent also that the long tube mill, originally adopted in metallurgy from the cement industry, is finding less favor and is being shortened, with consequent advan-

tage in reduced power consumption. This is true not only on the Rand, but also in this country. At one time Rand tube mills were "standardized" at 22 ft. by $5\frac{1}{2}$ ft. More recently these dimensions have been changed to $16\frac{1}{2}$ ft. by 6 ft. In Mexico experiments were made on a long tube mill by boring discharge holes at intervals throughout its length, and examining the products obtained after passing through different lengths of the mill. It was found that the first 16 ft. of a 22-ft. tube gave a product as good for the purpose desired as did the entire mill. Hence the length was accordingly reduced.

Two other features of tube-mill construction and operation now being adopted in this country are the tire-discharge and operation of mills in tandem. The tire-discharge is 18 in. in diameter, fitted with reverse spiral to prevent discharge of pebbles. The use of tube mills in tandem, with intermediate classification, has been found, in some cases, superior to a larger number of mills in parallel, treating the same tonnage of ore and yielding as good a product for cyaniding.

Steel Under the New Tariff.

The opposition of the American steel manufacturers to the great reductions in the tariff is popularly misunderstood, though not from there having been any lack of adequate explanation. Repeatedly the question is asked why American steel manufacturers should fear foreign competition within our borders when we are able to export 2,500,000 tons of steel products annually, in competition with the world. Answers have been given, but they are not generally heeded.

The American steel industry is placed in a position which would be untenable in a military contest. Imagine a people with an impregnable citadel and outposts located at random, some of which are much more accessible to the enemy than to those upon whom the burden rests of defending them. The people are able to make successful sallies with certain forces, say with cavalry, while with infantry they must keep strictly within their own borders. In war a new alignment would quickly be created. Some of the outposts would be lost. The cavalry would carve a way for the infantry.

The American steel manufacturer cannot so wage his warfare in the territory which geography seems to have given him. Important consuming markets are much farther removed from his centers of production, in point of cost of carriage, than they are from foreign ports. Costs of production are so aligned between American and foreign producers that the American manufacturer can send some products to the uttermost parts of the globe, while others must stay at home, and the first cannot open the way for the second. Steel pipe is readily exported, but merchant steel bars, except in the Canadian market, can rarely compete with the foreign producer.

For many years prices of the great majority of steel products have been based upon Pittsburgh as a basis, plus the rail freight to destination. This structure has been regarded as good, and the practical operation has been eminently satisfactory—to the steel producers and to the railroads. On an actual count one could probably easily find a million tons of steel products consumed in the United

States in districts to which the freight from Pittsburgh exceeds the freight from foreign ports by a greater amount than the tariff duties hitherto levied upon such products. In the territory consuming those products the American mills are now, with the inconsequential duties enacted, placed at a great disadvantage, in that to meet foreign competition their realized prices at mill would be less than those of the best located foreign producers.

Again, there are great differences in the alignment of prices of different products. In the United States plates, structural shapes and merchant bars have lately been quoted at the same level, though until recently plates and shapes commanded \$1 per ton more than bars. German bars at Antwerp, however, are quoted \$3 a ton lower than German structural shapes, while the new tariff is lower on bars than on shapes, and the freights are the same on the two commodities, both our inland freights and those across the water. The fact that we can and do export structural shapes in large quantities cannot save use from a German sally into our territory with bars.

American manufacturers have been keener to improve their manufacturing methods than their selling methods. In the final analysis it may be that they feel much stronger in their mill cost structure than in their market structure. The market has been particularly sensitive to competition, and taking its whole history not infrequently savagely destructive competition. To protect themselves American sellers have resorted to artificial means, sometimes elaborate, sometimes relatively simple. Sometimes there has been no more control than that each manufacturer for himself desired to avoid departing from a price openly recognized as "the market." Such an adherence would be possible only were the price structure an extremely simple one, and thus easily understood, and the structure has been that of there being a Pittsburgh base price, delivered prices being in all cases the base price plus the rail freight to destination. Showing these rates large books have been published, whereby no misunderstanding can readily occur. With certain of the markets vastly more exposed than others to foreign competition, such a simple structure cannot be maintained, and adherence to regular selling prices by the mere volition of the individual manufacturer is made practically impossible by there being no recognized structure, unless the exposed markets are calmly resigned. Naturally the steel manufacturers have not been eager to emphasize that part of their objection to tariff reduction which the facts here indicated suggest, but that their dissatisfaction rests partly upon this foundation is rather plainly apparent.

However, we have been assured that to be faced with additional competition will brighten our wits and make us better men in the long run. The thought has been suggested chiefly with reference to our manufacturing methods, but we may find that our selling methods, which confessedly are not good in some respects, may also be improved even though the ordeal promises to be a trying one. Assuredly one of the effects of the lower tariff will be to increase the tendency toward competition between our own mills. No analysis of the prospects can avoid that conclusion entirely, though the superficial idea has been purely that additional competition would be forced between American mills as a class, and the foreign producers.

Readers' Views and Comments

Continuous Counter-Current Decantation

To the Editor of Metallurgical and Chemical Engineering:

SIR:—I note in your issue of September, in an editorial on the influence of South Dakota on cyanidation, a statement which would cause one to infer that the use of continuous counter-current decantation by John Randall in 1901 was a failure. Broadly speaking, if a plant shuts down it can be called a failure; but the following account of the circumstances connected with the plant will, I think, throw light upon the subject.

Mr. Randall, who is a very conscientious man, was approached by the owner of the property and asked to build a mill upon it for him. He protested that it would be much better to spend the money on further development of the mine; but when he found that the owner was intent on building a mill under any conditions, he finally consented to build it for him, and did so, developing there the counter-current method mentioned. After the mill had run for a month or so, the ore available was worked out and the rock furnished did not contain over \$1 per ton, so that the plant was naturally forced to shut down.

After this happened I met the owner one day, and speaking of the property he said: "When a mill shuts down there are only two things to do, blame either the mine or the chemist. As I want to sell the mine in this case I have to blame the chemist."

If Mr. Randall had been fortunate enough to install the process the first time at a plant with a good supply of ore, it is most probable it would have kept running indefinitely, as Mr. Randall's ingenuity would doubtless have solved the various small problems that arose in developing the use of this method. There is no question, of course, but that the Dorr thickener cuts out all the troubles met with in cones, and is far preferable, but it would be unjust to say that the method without it was a failure.

JOHN V. N. DORR.

Denver, Col.

"Hydrometallurgy—Joys of Its Theory, Woes of Its Practice"

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In the September number of this journal Dr. Regis Chauvenet has pointed out some pertinent facts and instances relating to hydrometallurgy. Inasmuch as this matter is of fundamental importance, I wish to present another perspective of the subject, in addition to pointing out some statements which, taken as representative, seem unfair to a branch of metallurgy along which so much has been accomplished, and of which so much is to be expected.

Notwithstanding the statements of the author to the contrary, the sum total impression one gains is that hydrometallurgy is a dangerous and fickle field—and one which offers particularly fertile soil to the charlatan, the ignorant chemist, or the unscrupulous promoter. There is no denying that, within the limits of poetic license, the facts as set forth are true, but the instances are scattered, and no process nor type is followed to a logical conclusion.

First of all, it should not be inferred that all of the impossible processes and inventions of a chemical nature have been in the field of wet metallurgy. A few years ago I witnessed a brilliant electric furnace operation which outclassed the broadest claims of either the Wynne process or the treatment of Herkimer sand.

Another example of the reduction of a staple product without a reducing agent has been so recently well advertised as to need no more than passing reference.

A Western genius not long ago devised a roaster which not only prepared the noble and base values for quantitative separa-

tion, but delivered the gangue ready for sale as cement. I think they hope to save most of the potash, too.

It is also within the memory of man that a process for the manufacture of diamonds was sold for a large sum; that a tragedy resulted from the belief of the claims that gold could be made out of baser metal; and that a very smooth gentleman convinced moneyed men that aluminium could be reduced with carbon.

The author specifically omits the cyanide process since "it constitutes a class by itself." Yet some of the greatest swindles ever devised were based upon the successful reputation of cyanidation. A rather large scale failure of a cyanide process (absolutely honest in this case) has just been announced. Another case of cyanide failure is in Boulder County, Colorado (there have been several failures in that locality). A very large mill was built at a cost reported to be in the general magnitude of a quarter of a million dollars. The pegmatite dyke which was to supply the ore contains, to my positive knowledge, as little gold as the sea water mentioned by the author. Almost all of these cyanide propositions begin in the same way—a large body (a prospect hole, in most cases) of low-grade ore, a 10-ton mill with a little tin-whistle roaster. The mill starts with a grand splurge, but screens refuse to work, the crusher breaks a jaw, the screw conveyor fails. During the weary months of getting started the manager convinces his stockholders that the mill will pay exactly three times as much on 30 tons per day as the 10 tons first proposed. Then a 50-ton unit, and finally, if the insurance companies do not get the mill, another ambitious promoter gets the machinery and the story begins again.

Yet the cyanide process has yearly added millions to our wealth.

It seems particularly unfortunate, therefore, that the author should have selected hydrometallurgy as the field in which impostors and half-learned chemists chiefly operate. There are unprincipled men and quacks in all branches of chemistry and metallurgy, and in mining. Keeley motors, gas burners which consume nine parts of air to one of gas, wonderful vacuum pumps which lift 2-ft.-lb. of water with 1-ft.-lb. of energy expended, marvelous tunneling machines, all attest the exploitation by fakirs of other fields as well.

The author's remarks on the relation of the laboratory process to the mill process are to the point, and they are true without exception so far as I know or can imagine. Yet if we accept the spirit of this article we shall be buried in a swamp of pessimism so deeply that only the most pressing need can force us to emerge. It is given no man to foresee the difficulties and pitfalls which lie between the laboratory and the finished mill. But by persistent labor, careful observation, and unflinching optimism, a successful end is not only possible but probable.

I disagree most thoroughly with the author when he speaks of fallacy. There is no magic in laboratory operations. The relations between cause and effect are the same wherever matter exists. The difficulty is that some men try to imitate laboratory apparatus in planning reactions on a large scale. A bubble of gas coming up through a beaker means only a million bubbles coming up in just the same way in a very large vessel. Boiling means a gas burner underneath. How queer to think of forcing hot gas or steam into a tank of liquor—especially if one never thinks of it. If a fallacy exist, here it is: the mind of a monkey trying to grasp original problems. We are all like monkeys in many ways. We can't quite forget how the other fellow does it.

An intelligent worker may, by treating quantities of one gram or less, draw valuable and correct conclusions. The experienced metallurgist will always avail himself of this opportunity. But here lies a dangerous trap for the tyro: he overrates and underrates the order of magnitude—he cannot prop-

erly control his imagination. On the other hand, the careful metallurgist will not be far misled by the phenomena attendant upon gram-lot treatments. But rather, having gained a large amount of information from working in a small way, he is well prepared to proceed with larger quantities. After one has worked with 1 gram, 10 grams, 100 grams, he may deem it profitable to work with a pound or 10 pounds.

These operations, while they check up the work done in the smaller way, do so only incidentally—they are for the purpose of obtaining additional data upon which further conclusions may be based. Broadly speaking, the gram experiments are qualitative, the pound experiments quantitative.

Also shapes, sizes and materials of apparatus begin to crystallize out, systems of heating and agitation may be tried, and a general idea may be had of ease of filtration and washing, together with a conception of the relative bulk and treatment of secondary products. Each experimenter will have different methods of approach and different technique, but there are certain well-defined endpoints which everyone must reach before proceeding to a finished process. The man who omits to study the solvent effect of his solutions upon everything with which they come in contact might as well go to selling lightning rods again.

Again, there are certain standard operations, the practice of which the metallurgist may correctly assume to fit into his proposed process without difficulty. Among these may be included crushing, screening, roasting, pulverizing, heating, agitation, decantation, filtration, absorption, drying, sampling, the application of power and electricity, and classes of materials. These things have all been studied and standardized. There is no need of inventing a vacuum filter if men who have spent their lives in the business offer you the best they have. I knew a man once who prided himself upon having invented every piece of apparatus in his plant. He had spent thousands of dollars needlessly, and had inferior stuff in the bargain.

In the laboratory a gram of material is digested in a glass beaker over a gas burner. Fortunately we have no large glass tanks to contend with in practice. And what a blessing we cannot put a burner under a 600-ton tube to heat it. But the chemical engineer knows that there are means of boiling solutions efficiently, and that there are to be had containers of any desired size or shape for the treatment of almost any solution, be it alkaline or acid.

A diligent perusal of advertisements in reputable scientific journals will keep the metallurgist well informed as to almost all standard apparatus. The successful metallurgist who does not read advertisements is a genius of the highest type who could increase his efficiency 25 per cent by the simple formula above.

I do not think the sarcastic remark "Multiply the last figure by 100—and there you are" is warranted by the conditions stated. If the author is speaking of fakirs and ignorant persons who call themselves chemical engineers, he should not blame the system of hydrometallurgy. If he wishes to criticise hydrometallurgy fairly, he should at least employ in his theory the services of men of experience and common sense.

In answer to his question, "How many experts per ton?" I should without hesitation say, "One expert per gram, and let him stay with that gram until he has finished." It is at the gram stage of operation that the expert density should be high, in order that time, money, and materials may be conserved.

"A stoppage—arose from the sudden discovery that a certain element was present in solution which had no business there." (This sounds like calcium sulphate in the Malm process referred to.) Assuming the explanation correct: "Possibly the solution—constituted the solvent" I should say the preliminary work in about the pound stage had either never been done or else had been done in a very slipshod manner.

If each and every reaction has been tested out thoroughly and independently before the process is correlated; if the results are entirely satisfactory; there still remains the wearing off of the rough edges of the machinery and apparatus, and

what is still more important, the training and discipline of the working force—from the superintendent to the beaker boy.

After this stage has been reached, the stockholders and friends may be invited to tea and cigars while the button is pressed for the long-expected opening. Generally the date is set for this button-pressing business about the time the concrete is poured for the foundations, and the general feeling is that the stockholders must not be disappointed for the *n*th time. When they do come, it is a fortunate superintendent who hasn't a half dozen such experiences as recited by the author.

The statement relating to low efficiencies and recoveries is, of course, correct, but it is not denied by anyone excepting those whose opinions do not matter. Because the actual recovery of values is low is no cause for alarm. Processes yielding as low as 20 or 30 per cent of the original value may be commercially very successful. I could name at least three processes in which the net efficiency is under 50 per cent, but which have netted millions of dollars in real money.

I consider the comparison which the author has made between wet processes and smelting processes to be extremely unfair. It is true that a smelter can absorb almost all classes of ore economically. It is also true that a wet process can be usually applied to not more than one class of ore. But one cannot condemn hydrometallurgy because it is not a cure-all. A smelter is necessarily a large unit—larger than the average mine or group of mines can supply. The wet process on the other hand may be applied with efficiency and economy to relatively small bodies of ore.

The honest mine with real ore in it can usually obtain a suitable process. It does not need to worry over the possible discoveries of other mines or other kinds of ore. If the ore is not blocked out, the erection of a mill is wildcatting. If it is blocked out, then work out a process for what exists. Of course, if it is desired to have a process which will absorb all kinds of ore, why not make it broad enough to save the aluminium, potash, and zirconium. It is merely a matter of degree, just the same as a high recovery of values. It is a simple matter to get a recovery of 99 per cent if one cares to pay the price.

For example, the Malm process spoken of is designed to treat sulphide ores. The theory (the most successful part of the process) excludes oxides and carbonates. What sane man, therefore, would try to chloridize an oxide or carbonate under the conditions described?

If the metallurgist has seen to it that he has a fair sample of the ore as it exists, and works out a process on that ore, he has been successful. If the mining engineer was mistaken, or if a new ore body is opened up, it is asking too much to require a process to be of such prophetic nature as to adapt itself to the new unknown conditions.

Excepting by inference, I have not spoken of the possibilities of the application of electricity to hydrometallurgy; yet in the years to come it may be predicted that the advance of these two, hand in hand, will be rapid. Given plenty of good water, salt, sulphuric acid, soda ash, and electrical energy, the metallurgist of to-day and to-morrow will assure hydrometallurgy an exalted position not surpassed by any other method of winning the values from raw ores.

Canonsburg, Pa.

WARREN F. BLECKER

New York Section of the American Electrochemical Society

The first meeting of the season will be held at the Chemists' Club, 52 East Forty-first street, New York City, on Friday, October 17, at 8.30, and will be preceded by an informal dinner at the same place at 7 p. m.

The subject for the evening is **some applications of the microscope to industrial work**, and the program is as follows:

Metallography of Carbon. By G. A. Roush, Lehigh University.

The application of the principles of metallography and petro-

graphy to the identification and analysis of carbon products, showing the development of characteristic structures and the utilization of these structures in identification and analysis.

The Microstructure of Raw and Manufactured Copper. By W. H. Bassett, American Brass Co.

A series of pictures showing structure of cast copper and how this structure is altered by rolling, etc.

The Microscope in Mineralogical Analysis. By Gilbert Rigg, New Jersey Zinc Co.

A series of Lumiere plates showing microphotographs of zinc ores with resolution of the minerals present.

Some remarks on Micrometry as Applied to Alloys. By Dr. C. H. Mathewson, Yale University.

The use of micrographic methods in investigating questions of constitution, qualitative and semi-quantitative interpretations; structural relations which are favorable to the development of micrometric methods; advantages and short-comings of such methods in ordinary analytical practice; the determination of cuprous oxide in copper; of iron in zinc.

These lectures will all be well illustrated by lantern slides and photographs. Any member is welcome to bring guests who may be interested.

Mr. Lawrence Addicks, of Chrome, N. J., is chairman of the New York Section of the American Electrochemical Society, and Mr. H. B. Coho, 111 Broadway, New York City, is secretary-treasurer.

Iron and Steel Meeting of the American Institute of Mining Engineers

The October meeting of the American Institute of Mining Engineers, which is to be held under the auspices of the Iron and Steel Committee, will be held on October 16th and 17th, 1913, in the United Engineering Societies Building, New York City. Members of friendly societies are invited.

The program for the meeting is as follows:

First Session, October 16, 1913, 10 A. M.

W. A. Forbes, Blast Furnace Gas Cleaning.

W. H. Blauvelt, The Slagging Producer.

F. Peter, The Generation of Steam by Waste Heat from Furnaces.

G. C. Stone, The Generation of Steam by Waste Heat from Regenerative Furnaces.

W. R. Shimer, Over-Oxidation of Steel.

Second Session, October 16, 2 P. M.

H. F. Miller, Jr., New Design of Regenerators for Open-Hearth Furnaces.

Richard K. Meade, The Use of Powdered Coal as Fuel.

H. R. Barnhurst, The Use of Powdered Coal as Fuel for Metallurgical Furnaces.

W. S. Quigley, The Use of Powdered Coal as Fuel.

E. Stuetz, The Scoria Process for the Manufacture of Fine-Ore Briquettes.

Felix A. Vogel, Briquetting.

Felix A. Vogel, Use and Advantages of Briquettes in Blast Furnace Practice.

R. H. Lee, The Use of Nodulized Ore in the Blast Furnace.

Third Session, October 17, 10 A. M.

H. M. Howe, Discussion of the Existing Data as to the Position of A₂3.

H. M. Howe and A. G. Levy, Determination of the Position of A₂3 in Carbon-Iron Alloys.

H. M. Howe, A₂3, the Equilibrium Temperature for A₂3 in Carbon Steel.

G. K. Burgess, J. J. Crowe and H. S. Rawdon, Thermal and Microscopical Examination of Professor Howe's Standard Commercial Steels.

H. M. Howe, The Divorcing of the Eutectoid in Meteorites.

Fourth Session, October 17, 2 P. M.

R. R. Abbott, The Influence of Various Elements on the Absorption of Carbon by Steel.

J. V. Emmens, Some Phase of the Practical Treatment of Tool Steel.

W. E. Ruder, Grain Growth in Silicon Steel.

J. H. Hall, Shock Tests of Cast Steel.

G. H. Clevenger and B. Ray, The Influence of Copper Upon the Physical Properties of Steel.

J. H. Hall, The Life of Crucible Steel Furnaces.

Mr. H. M. Boylston, Abbott Building, Harvard Square, Cambridge, Mass., is the secretary of the Iron and Steel Committee of the Institute. Mr. Bradley Stoughton, 29 West 39th street, is the secretary of the Institute.

Chicago Meeting of the American Institute of Metals

The next meeting of the American Institute of Metals will be held in Chicago, Ill., from October 13 to 17. The hotel headquarters will be at the Hotel LaSalle.

The arrangement of the meeting is in the hands of a committee with Mr. Gillett as chairman. Among the papers announced so far are the following: Soldering Fluxes for Soft Solder, by Walter Arthur; Core-Room Economies, by O. F. Flumerfelt; Report of the Official Chemist, by Arthur D. Little, Inc.; Brass Foundry of the Future, by G. P. Karr; the Boiling of Metals, by J. W. Richards; Producer Gas, as Applied to Brass Melting, by E. F. Bulmahn; Internal Strains in Castings, by J. E. Howard; the Work in Metals of the Bureau of Standards, by G. K. Burgess; discussion on the Nomenclature of Non-Ferrous Alloys, lead by G. K. Burgess; Approximate Melting Points of Some Commercial Copper Alloys, by Norton and Gillett.

Mr. W. M. Corse, Lumen Bearing Co., Buffalo, N. Y., is the secretary of the Institute.

American Institute of Chemical Engineers

The annual meeting of the American Institute of Chemical Engineers will be held in New York City from December 10 to 13.

Prof. John C. Olsen, Polytechnic Institute, Brooklyn, N. Y., is the secretary.

New York Section of the American Chemical Society

The first meeting of the season will be held in the Chemists' Building on Friday evening, October 10th, and will be opened by an address by the chairman of the Section, Dr. C. B. Hesse.

This will be followed by a paper by F. J. Pond, on a review of the pioneer work on the synthesis of rubber.

Mr. C. M. Joyce, Arlington, N. J., is the secretary-treasurer of the Section.

The Western Metallurgical Field

Status of Labor Strikes

Although conditions in the copper country of Michigan have been improved since the first disturbance incident to the labor strike called by the Western Federation of Miners, they are yet far from normal and it will be months before the various companies are operating at their usual capacities. Indications are that the strike has been broken. The Calumet & Hecla is hoisting and treating some ore, and other companies have resumed production on a small scale.

The operators held firmly to their decision not to recognize or treat with the Federation, and in a letter to the Governor of the State have set forth reasons for their refusal to accept his suggestion for a conference between representatives of the companies and the miners. The companies stated that they believed they were conserving the best interests of the miners as well as of the mining industry by refusing to accept the demands of the Federation. In defense of their attitude they cited the past record of the Federation and stated it as their belief that their miners had been misled by an organization which has been discredited in other mining districts.

Late advices indicate that the Federation is inclined to yield recognition of the union. President Moyer and some other officials have visited the Governor and suggested arbitration

of the points at issue, eliminating the question of recognition, but the outcome of the conference is not known. Disturbances of a minor nature have continued throughout the strike, some resulting in bloodshed. Troops are still in the district, and the mounted patrol has been increased in order to cope with the situation.

The labor strike in southeast Missouri is reported to have been settled, based on an increased wage to the miners without recognition of the union. The first demand was for an increase of 50 cents per day, but later this was reduced to 20 cents, which was considered more favorably.

Probable New Copper Smelter for Nevada

The Consolidated Coppermines Company is the name of the corporation into which were merged the Giroux Consolidated, Butte & Ely, Chairman Consolidated and Coppermines companies, all in the vicinity of Ely, Nev. Following the exchange of stock in the old companies for that in the new, according to terms of the merger, the new company has arranged a bond issue of \$2,500,000. The purpose of this latter fund is stated to be, among other things, "for carrying on simultaneously a comprehensive series of experiments with a view of determining the method of securing the highest percentage of extraction of metals and at the lowest possible cost; and when such methods are determined, for designing, constructing and equipping an adequate concentrating and smelting plant for the treatment of the ores of the Consolidated company and any customers it may have."

In consideration of what has been announced, and the later activity of the company's agents in securing a suitable site for works, it would appear that a new concentrating and smelting plant will be constructed. The old question of smelter fume and damage to country surrounding the smelter is entering into the selection of a site. Both the company and the inhabitants of Ely seem desirous of having the new works located near to that town. In an effort to protect itself against future trouble, the company has been securing from the inhabitants of Ely and vicinity, waivers against damage from smelter smoke. Little difficulty is met in securing such waivers, although objection has been made to the fact that the nature of the agreement does not obligate the smelting company to adopt any device now known or to be invented whereby the damage arising from smelter smoke could be abated.

Leaching Experiments

Interest continues to be manifested in the leaching of copper ores and more companies are constantly making experiments. The work at different points in Montana and Arizona has already been alluded to, and holds out promise of success. More recently the Miami copper company has announced that a leaching process is to be given a trial in an experimental unit of 20 tons capacity. This work will temporarily set aside any experiments with flotation, and all effort will be made to improve the present system of concentration supplemented by leaching. It is anticipated that success along these lines will reduce the percentage of copper in the tailings from 0.8 per cent to 0.2 per cent. The sixth unit of the mill is even now making a better recovery than is obtained in the other five, due to some alterations.

Another company which has announced its intention of experimenting with leaching is the Nevada Douglas Copper Company. According to a statement by the president, several carloads of ore will be shipped to Denver, where the experimental work will be done in a plant that is already partly equipped. Following the completion of the tests, a large plant is to be designed for erection at the mines. At present the company has a contract for the smelting of its ores by the Mason Valley smelter, and it is the intention to have the new leaching plant completed before that contract expires.

American Mining Congress Philadelphia Meeting

During the week of Oct. 20 the American Mining Congress will hold its annual convention in Philadelphia. The meeting promises to be of unusual interest, in that the Western metal

mining interests are to be as fully represented as the coal mining industry. Among the principal topics for discussion will be that of mine taxation, which is a matter that has agitated the mining industry for some time. It is expected that the discussion will elicit definite information on an equitable and scientific method of taxing mines and their output. Other matters of Western importance which will be discussed are a definite policy concerning the development of Alaska, the smelter smoke question and the disposal of debris from placer mining. The question of federal aid to mining education also will be on the program.

A new feature of the meeting will be the first mining exposition. This is a matter that has been agitated for previous meetings, but has never been brought to pass. At this meeting, however, arrangements have been made for the exhibition of mining machinery, demonstration of processes, methods adopted for the prevention of accidents in mines and mills, and provisions for rescue work in case of coal-mine accidents. The exposition will be held in Horticultural Hall, and is expected to add distinct interest to the meeting of the Congress.

Brown Cyanide Process Decision

The suit of Joseph A. Vincent vs. Tonopah Milling Company of Nevada and Desert Power & Mill Company, in the United States District Court for the District of Delaware, has been decided by Judge Edward G. Bradford in favor of the complainant, who alleged that the defendant was infringing the cyanide process patent of Alden H. Brown. The patent in question relates to an improvement in cyaniding precious metal ores, and has special reference to the sequence of the several steps by which the metals are recovered by cyanidation and concentration. Customarily, an ore requiring both concentration and cyanidation is subjected to treatment in the order mentioned. Brown's patent covered the reverse method, viz., cyanidation followed by concentration. It was the patentee's idea that his process would effect a greater recovery of finely divided gold and silver, by dissolving them immediately in cyanide solution, rather than attempt their recovery by concentration before cyanidation, and thereby subject them to probable loss.

The claim on which decision was given is as follows: "The herein described process for the treatment of ore, consisting in first, pulverizing the ore in the presence of cyanide solution; second, subjecting the ore to hydraulic classification by the introduction of cyanide solution at the bottom of an overflow tank to produce an ascending current; third, leaching the ore by the use of cyanide solution, whereby the finer metal content of the ore is dissolved; fourth, removing the dissolved metal from the ore in any suitable manner; and finally subjecting the residue to concentration."

The court held that in effect the process used at Tonopah was an infringement on this process, and sustained the patent in suit. On its face the decision may appear to be of wide importance, but it doubtless will be reviewed by a higher court, after which its true import will be more apparent. It will be observed, however, that one of the steps in the process is hydraulic classification of sand and slime, against which it would appear that mechanical classification could be successfully substituted. Again, it would be possible to evade the stated sequence of the steps of the Brown process, and still do good metallurgical work. Crushing might be done in water, with subsequent dewatering of the pulp before cyaniding; or the sequence of cyaniding and concentrating could be changed without material loss. In short, it would appear that the Brown process is so clearly defined as to successive steps that few mills would find it impossible to avoid infringement without material loss. General practice is not in accord with the Brown process, and in any event there would be only a few special cases amenable to the decision.

Company Reports

From the twenty-eighth annual report of the Mount Morgan Gold Mining Company, Mount Morgan, Queensland, for the year ending May 31, 1913, the following interesting facts are

taken. The tonnage treated, grade of ore and metal recovered are shown in the table:

	Tons treated	Copper %	Gold dwt. per ton	Copper tons	Gold oz.
Copper ore.....	203,777	3.41	9.50	6,958	96,852
Gold ore.....	19,895	2.66	18.93	530	18,831
Many peaks.....	88,396	1.52	15	1,346	658
Purchased ores.....	353	11.37	5.41	40	96
Miscellaneous products.....	9,677	4.19	4.24	406	2,051
Total and average.....	322,098	2.88	7.36	9,280	118,488

In addition to above gold, 7594 oz. was recovered from cleaning up the Mundic works which had been dismantled, as stated in the last half-yearly report.

Many minor improvements have been made at the smelting plant during the year, all of which have contributed to better operation. The large increase in wages, however, have shown their effect in an increased cost of smelting. New converters were installed during the year, but owing to trouble in the power plant, they have not been worked to the best advantage. The company has arranged for the purchase of 100,000 tons of iron ore for fluxing purposes, so that a large quantity of silicious ore in the mine can be treated. There is about 1,000,000 tons of such silicious material.

The old Mundic works have been completely dismantled, and the erection of a new 500-ton concentrating plant is well under way. It is expected that ultimately the capacity of the plant will be brought up to 1000 tons daily. Before designing this mill an experimental unit was run for some time and will continue for some months more. It has been found necessary to resort to finer grinding than usual on account of the intimate association of the minerals. The sum of \$15,000 has been spent on concentration experiments, and further investigations will be made, particularly with reference to American practice. Good progress is being made on the new smelting works and power plant.

The Iron and Steel Market

September has been a month of declining activity market-wise. August has shown a clearly perceptible improvement over July, apparently presaging further improvement in September owing to the close of the midsummer period, in which business is normally dull, for the first half of September showed slightly less activity than August, and after the middle of the month almost complete stagnation suddenly appeared. Orders grew less, specifications on old contracts decreased, and buyers ceased to show any interest at all in the future. Throughout the industry it was at once noticed that a change had occurred, though it was difficult to determine the cause.

The more common view is that the imminence of the passage of the new tariff bill was responsible for the change. For months the knowledge that the bill would eventually be passed has operated to curtail forward commitments, and the buying has been only against definite wants, the filling of which could not be deferred, but it appears that buyers still had a little leeway, and could curtail their orders further. There is no evidence that actual consumption has decreased materially, though in a couple of months the major part of the railroad business on books will have been completed and in other directions there will be curtailment on account of the closing of the outdoor season. These influences would not be sufficient to explain the present stagnation.

Since about the first of the year, or just after the culmination of the great buying movement of 1912, the inflow of new business in steel products has continuously been less than the current shipments, but the mills had an accumulation of old business from which to draw. While this supply is not exhausted the mills have caught up and the remaining business on books can only be shipped at certain rates, in accordance with the requirements of consumers. Furthermore, a considerable portion of the business on books is subject to cancellation if the buyer can secure a better price elsewhere, for most contracts in the finished steel trade are lightly regarded. Already there have had to be many adjustments on contracts in force, and with the outlook grown much more uncertain as to

prices buyers are still more chary as to placing specifications on contracts.

With mills now fully caught up on their old business, and with new business coming in at a relatively low rate, curtailment of output is practically inevitable. There have been few if any definite closings, but many mills probably operated under somewhat reduced pressure in September. Owing partly to weather conditions, actual output in the past three months has been at about 90 per cent of full rated capacity, and the outlook is that production will probably drop to between 75 and 65 per cent of capacity before the close of November.

Thus far the general level of prices have declined but slightly. Pig iron has not declined at all, while billets and sheet bars have declined \$2 to \$3 a ton in two months, and finished steel products have softened quite generally, though declines in openly quoted prices are recognized only in plates and shapes to the extent of \$1 a ton, and in black and galvanized sheets to the extent of \$2 a ton, such declines having occurred in September.

Both the advent of the new tariff and the fact that the mills can no longer operate at full capacity with the amount of business currently offered dictate that there shall be declines in prices and the next two months will probably witness some marked recessions. The tariff revision necessitates some important revisions, though how far they will extend is a question, the situation being a very complicated one. The American price structure rests upon a Pittsburgh basis price, plus rail freight to destination, these rates being many dollars per ton to the Pacific coast and to southwestern and southern points, while freights from foreign ports vary but slightly. Extras for size, section, etc., above base prices vary greatly between American and foreign products, and thus a great deal of study and negotiation will be necessary before the real competitive basis can be developed. The American mills do not intend to allow any large tonnage of foreign material to come into the United States, but as some semblance to the present price structure, in the matter of delivered prices, must be maintained there will probably be importations of some moment, chiefly for districts well removed from Pittsburgh.

Production of pig iron in the United States in September was at the rate of about 30,000,000 tons per annum, made up of about 21,500,000 tons by the steel works furnaces and 8,500,000 tons by the merchant furnaces. The total production thus shows a decrease of about 12 per cent from the maximum rate attained earlier in the year.

The steel works furnaces have lost about 8 per cent from their maximum rate and the merchant furnaces about 15 per cent from their maximum.

Even with a greater curtailment in output during the remainder of the year than now seems in prospect new tonnage records will be made by the calendar year in pig iron, steel ingots and finished steel products.

Rolled iron will probably show a further decrease. Its output has been decreasing for several years.

Pig Iron

The pig-iron market showed a slight advancing tendency in the fore part of September, and with the stagnant condition of late in the month such advances as had been scored were not lost, but at the close of the month the tendency is in the direction of stationary prices. Production seems well regulated to demand. Statistics privately gathered show a slight decrease in stocks during August, and total stocks relatively small. Southern iron advanced 75 cents a ton during September, while at Philadelphia and Cleveland there were advances of about 25 cents. The current market stands as follows: No. 2 foundry, f. o. b. Birmingham, \$11.50; No. 2 X, delivered Philadelphia, \$16; No. 2 X, f. o. b. Buffalo furnaces, \$14; No. 2 foundry, delivered Cleveland, \$14.75; No. 2 foundry, f. o. b. Chicago furnaces, \$15; at valley furnaces (60 cents freight to Pittsburgh and Cleveland) Bessemer, \$15.75@16; basic, \$14; No. 2 foundry, \$14; malleable, \$14.25, and forge, \$13.50.

Steel

The market in unfinished steel has been softening, particularly during the second half of September. Sellers had consumers fairly well booked on contracts, but upon these they have had to make a number of readjustments. Generally these have been effected quietly, and relatively little inquiry has come into the market so that prices are not well developed. Formal offerings have been refused of billets at \$24 and sheet bars at \$25, either Bessemer or open-hearth, f. o. b. maker's mill, Pittsburgh or Youngstown, so that the market is not quotable above this level. Rods are available at \$27, Pittsburgh or Youngstown, with few if any takers. In the east the market is quite at sea since billets are free under the new tariff and former asking prices of mills cannot be secured within several dollars a ton.

Finished Steel

Openly quoted prices on shapes and plates declined \$1 a ton in September, and on sheets \$2 a ton. Prices quoted below represent the open figures, with considerable shading in the majority of products, rails and sheets being exceptions. Prices are f. o. b. Pittsburgh unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, tank quality, 1.40 cents.

Shapes, 1.40 cents.

Steel bars, 1.40 cents, base.

Common iron bars, 1.55 cents, Pittsburgh; 1.25 cents at eastern Pennsylvania mill; 1.32½@1.35 cents delivered Philadelphia; 1.35 cents, delivered Chicago.

Wire nails, \$1.65, base; plain wire, 1.45 cents, base.

Sheets, blue annealed, 10 gage, 1.60@1.65 cents; black, 28 gage, 2.05@2.15 cents; galvanized, 28 gage, 3.10@3.20 cents; painted corrugated, 28 gage, 2.25 cents; galvanized corrugated, 28 gage, 3.15 cents.

Merchant steel pipe, 80 per cent off list for ¾ to 3 inch.

Steel boiler tubes, 3½ to 4½ inch, 60 per cent off list.

Standard railroad spikes, 1.65 cents, Pittsburgh; 1.70 cents, Chicago.

Button head structural rivets, 1.90 cents; cone-head boiler rivets, 2 cents.

The Non-Ferrous Metal Market

Since our last report the prices for non-ferrous metals have advanced appreciably, and at the time of writing the upward tendency is still noticeable. The settlement of the labor strike in Missouri has brought that district into production again, and the market for lead has been easier.

While the prices quoted herewith are all higher than those which were quoted a month ago, still higher prices have ruled in the interim.

Copper.—Business has been divided between foreign and domestic consumers. The quotations on Lake copper are only nominal, as there is none of this metal in the market except that offered by Calumet & Hecla.

Electrolytic copper is quoted at 16.65@16.75 cents, and Lake at 17 cents.

Tin.—This market was more active during the earlier part of the month, but was extremely dull toward the latter part. The last quotation is 42½ cents for September tin.

Lead.—This market is quiet, though prices rule higher than a month ago.

St. Louis lead is quoted at 4.60@4.62½ cents, and New York at 4.70@4.75 cents.

Spelter.—Prices for spelter have ruled higher than here reported. At present the market is quiet and prices are lower. St. Louis is quoted at 5.55@5.60 cents, and New York, 5.70@5.75 cents.

Other Metals.—Tariff legislation has affected conditions in the market for aluminium and antimony. Little business is being done. New York quotation on aluminium is 22@23½ cents per lb.; and on antimony the prices range from 7.15 to 8.50 cents

for various brands. Quicksilver is quiet and prices easier. New York quotes \$39 per flask of 75 lb.; and San Francisco \$39 for domestic, and \$36.50 for foreign orders.

A Study of the Oxidation of Coal and of the Process of Combustion*

By Horace C. Porter

It is well known from the work of several investigators in European laboratories¹ and from earlier work at the Bureau of Mines that all coals unite with oxygen more or less at ordinary temperatures. The action is at ordinary temperatures largely a simple addition, very little gaseous products of combustion being produced. There is, however, a development of heat by the simple addition of oxygen, and this heat development is the primary cause of spontaneous combustion.

As the temperature rises, the oxidation reactions increase rapidly in rate, and the simple-addition reaction becomes complicated with the development of water and oxides of carbon as products of the oxidation and with the production of water and of methane and other gases by destructive distillation.

Under whatever conditions coal burns, as for example on the fuel bed of a boiler furnace or in suspension as dust in the air of a mine, the earlier stages of oxidation proceeding at the lower temperatures have important bearing on the ease of ignition of the fuel and the degree of inflammability of its dust. In order to throw more light on the nature of the combustion reactions at successive stages and to determine how the various kinds of coal differ in their rates of reaction with oxygen, a laboratory study has been made of the oxidation of coal, at temperatures ranging from 80 deg. to 300 deg. C.

A series of experiments has been run with samples of different coals placed in atmospheres of pure oxygen at a pressure reduced so as to equal the average partial pressure of the oxygen in air. The rate of absorption of oxygen by the coal was determined by admitting pure oxygen at such a rate as to maintain constant pressure in the apparatus. Large differences in rate were found between different kinds of coal, and the comparative oxidation rates thus found conform in general to the known comparative degrees of inflammability of these coals.

At temperatures above 200 deg. C. the production of CO₂ and other gases causes a considerable error in an experimental method which does not remove these gases as they are formed. A rapid-stream method was, therefore, devised in which weighed samples of dry coal were heated at definite temperatures in a rapid stream of dry air for one hour, the change in weight of the coal being accurately determined as well as the amounts of H₂O, CO₂ and CO produced. From these data the total oxygen used and the oxygen fixed were calculated.

It was shown that the oxidation of coal consists in the earlier stages of the formation of an unknown complex of the coal substance with oxygen, and the more or less gradual dissociation of this complex as the temperature rises into water, CO₂ and CO. Below 200 deg. water is the chief product of this dissociation, the proportion of CO₂ in the products increasing as the temperature rises. Carbon monoxide is a primary product of the oxidation of coal at temperatures between 100 deg. and 300 deg. C. and possibly also outside of this range. Triangular diagrams representing the relationship of the three products H₂O, CO₂ and CO at different temperatures show interesting tendencies, e.g., toward constancy in ratio between CO₂ and CO within a certain temperature range.

The production of water by oxidation of coal at 100 deg. C or lower is of interest in connection with the devising of methods for accurate moisture determination in coal. The direct production of CO at moderate temperatures is of interest in connection with mining operations and the storage of coal.

* An address delivered at the annual meeting of the American Chemical Society, Rochester, N. Y., Sept. 9-12, published by permission of the Director of the Bureau of Mines.

¹Boudouard, Fagol, Richters, Anderson and others.

Corrosion Report Before the British Institute of Metals

The autumn meeting of the (British) Institute of Metals—the first to take place in Continental Europe—was held on August 28th and 29th in Ghent, Belgium.

The first session was opened by an address of welcome by President Schoentjes, of Ghent University, to which the president of the institute, Professor A. K. Huntington, replied.

The second report of the Committee on the Corrosion of Metals was presented by Dr. Bengough.

A summary of the second report is herewith given:

Details as to the nature of the practical problem have been ascertained from the inquiry forms returned to the institute. Important points brought out are:

1. The erratic nature of the trouble. Different users give as the normal life of a tube periods varying from 4 to 25 years.

2. The nature of the deterioration is in the great majority of cases dezincification, which results finally in a rotten and pitted tube. This type of corrosion is called "selective corrosion." Occasionally tubes fail by localized "complete corrosion," i.e., by the rapid removal of both Cu and Zn over certain areas.

3. It is not possible to correlate the failure of tubes with the presence, or nature, of the electric lighting system.

4. There is no obvious connection between any definite locality in a condenser and the frequency of dezincification.

The chemical and physical condition which exist in a condenser are considered; also the possible variations from point to point in the same condenser. It is the variations in these conditions which determine the locality at which corrosion commences. Engineers in charge of condenser plants are not usually in a position to study them, and consequently the information obtainable from them is of limited value.

There is an almost entire absence of experimental work showing how the speed and type of corrosion vary as the conditions vary. Consequently it has been considered advisable to carry out a detailed laboratory investigation into the matter.

I. Laboratory investigation on the standard tubes of the Corrosion Committee.

COMPOSITION OF STANDARD TUBES

	1. 70:30 Brass.	2. 70:28:2 (Pb). Special Brass.	3. 61:39 Muntz Metal.	4. 70:29:1 Admiralty Brass.
Copper	70.21	69.94	60.90	71.18
Zinc	29.17	27.60	38.21	27.28
Tin				1.07
Lead	0.27	2.08	0.46	0.28
Iron	0.27	0.28	0.33	0.21
	99.92	99.90	99.90	100.02

Summary of results of experiments carried out at 20° C.:

(1) Corrosion at this temperature is always of the approximately complete type and is slow and uniform.

(2) Carbon (in the form of coal, graphite and coke), sand, copper and a number of other bodies have, *per se*, little or no effect on the speed or type of corrosion. They do not give rise to any appreciable electrochemical action.

(3) Certain basic salts of zinc and ferric oxide appear to have some slight accelerating effect, but it is too small to be practically important.

(4) An increase in the concentration of the sea-water increases the speed of corrosion; a decrease in concentration decreases it. The type of corrosion is unaffected (but see below).

(5) The order in which tubes resist loss of weight is as follows:

1. Muntz metal (loses least weight).
2. 70:28:2 alloy
3. 70:30 brass
4. Admiralty metal.

(6) An increase in the speed of the water increases the amount of corrosion.

(7) Jelly tests have given inconclusive results.

(8) "Selective corrosion" could not be obtained at 20° C. except by the employment of free acid, or by the use of diluted sea-water for very long periods of time.

(9) A detailed account is given of the nature of the reaction which takes place during the process of corrosion.

(10) No evidence is forthcoming that "flaws" and "spills" have any influence on corrosion at the ordinary temperature.

Most of the experiments carried out at 20° were repeated at 40°:

(1) Corrosion at this temperature is of a distinctly "selective" type, i.e., the zinc is removed from the metal in larger amount, proportionately, than the copper.

(2) This action is by no means uniform but begins at certain points and spreads outward. The causes which determine the distribution of these points is discussed in detail. The action is inherent in alloys of the types considered, given the presence of sea-water and suitable temperature conditions.

(3) The order of resistance to "selective corrosion" at this temperature is:

1. 70:28:2 alloy (most resistant).
2. Admiralty alloy.
3. 70:30.
4. Muntz metal.

(4) Sea-water diluted with its own volume of distilled water exerts a greater selective corrosion effect than ordinary sea-water. Concentrated sea-water stimulates complete corrosion, but represses selective corrosion.

(5) The influence of carbon particles, etc., was no greater than at the ordinary temperature.

(6) The presence of zinc oxychloride on a tube gives rise to severe dezincification.

Inquiries have shown that it is highly probable that temperatures of 50° C. and upward are frequently attained in condensers. Consequently a series of experiments at 50° were conducted, mainly on 70:30 brass and Admiralty metal.

Results of experiments at 50° C.:

(1) Corrosion is of the selective type, and is concentrated almost entirely on the zinc. Mere traces of copper only were removed from the tubes, except in the case of the Admiralty tube.

(2) The action is severely localized.

(3) The order of resistance to selective corrosion is:

1. Admiralty metal (most resistant).
2. Muntz metal.
3. 70:30 brass.

The last two tubes showed signs of dezincification in 7 days. No signs of dezincification could be detected in the Admiralty metal in 4 weeks.

(4) The total weight of metal lost is lower at 50° than at the ordinary temperature for equal lengths of time. If the water be aerated the weight lost is enormously increased, and is greater than at the ordinary temperature.

(5) The effect of particles is the same as at the lower temperatures.

(6) Diluted sea-water (as above) and concentrated sea-water act as at 40° C.

II. Full details of the work carried out with the experimental condenser plant are given, and a comparison is made between the conditions obtaining in it and in an ordinary marine condenser. Three tubes of each of the standard compositions have been withdrawn and examined. The tubes themselves and the scale have been subjected to detailed examination. All the tubes show signs of dezincification with the exception of one of the Admiralty tubes, and one of Muntz's special brass. The extent and distribution of the areas of dezincification are considered in the light of the laboratory work. The agreement in the results obtained by the two methods is very striking.

III. Some preliminary experiments on the nature and mode of action of electrochemical protection methods are described, and the important part played by calcium carbonate scale is emphasized.

Continuity versus Atomism—The Limitations of the Scope of Physics and Chemistry

Presidential Address Before the British Association for the Advancement of Science*

By Sir Oliver Lodge

Natura non vincitur nisi parendo.

Let us ask what, in the main, is the characteristic of the promising though perturbing period in which we live. Different persons would give different answers, but the answer I venture to give is—rapid progress, combined with fundamental scepticism. Rapid progress was not characteristic of the latter half of the nineteenth century—at least not in physics. Fine solid dynamical foundations were laid, and the edifice of knowledge was consolidated; but wholly fresh ground was not being opened up, and totally new buildings were not expected. With the realization of predicted ether waves in 1888, the discovery of X-rays in 1895, spontaneous radio-activity in 1896, and the isolation of the electron in 1898, expectation of further achievement became vivid; and novelties, experimental, theoretical and speculative, have been showered upon us ever since this century began. Of the progress I shall say little—there must always be some uncertainty as to which particular achievement permanently contributes to it; but I will speak about the fundamental scepticism. To exhibit my meaning very briefly, I may cite the kind of dominating controversies now extant, employing as far as possible only a single word in each case so as to emphasize the necessary brevity and insufficiency of the reference. In physiology the conflict ranges round *vitalism*. (My immediate predecessor dealt with the subject at Dundee.) In chemistry the debate concerns *atomic structure*. (My penultimate predecessor is well aware of pugnacity in that region.) In biology the dispute is on the laws of *inheritance*. (My successor is sure to deal with this subject; probably in a way not deficient in liveliness.) And besides these major controversies, debate is active in other sections: In education, *curricula* generally are being overhauled or fundamentally criticised, and revolutionary ideas are promulgated concerning the advantages of freedom for infants. In economic and political science, or sociology, what is there that is not under discussion? Not property alone, nor land alone, but everything—back to the garden of Eden and the inter-relations of men and women. Lastly, in the vast group of mathematical and physical sciences, “slurred over rather than summed up as section A,” present-day scepticism concerns what, if I had to express it in one word, I should call *continuity*.

Still more fundamental and deep-rooted than any of these sectional debates, however, a critical examination of scientific foundations generally is going on; and a kind of philosophic scepticism is in the ascendant, resulting in a mistrust of purely intellectual processes and in a recognition of the limited scope of science. One thing is very notable, that it is closer and more exact knowledge that has led to the kind of scientific scepticism now referred to; and that the simple laws on which we used to be working were thus simple and discoverable because the full complexity of existence was tempered to our ken by the roughness of our means of observation. In most parts of physics simplicity has sooner or later to give place to complexity; though certainly I urge that the simple laws were true, and are still true, as far as they go, their inaccuracy being only detected by further real discovery. The reason they are departed from becomes known to us; the law is not really disobeyed, but is modified through the action of a known additional cause. Hence it is all in the direction of progress.

If we had to summarize the main trend of physical controversy at present, I feel inclined to urge that it largely turns on the question as to which way ultimate victory lies in the fight between continuity and discontinuity. On the surface of Nature at first we see discontinuity; objects detached and countable. Then we realize the air and other media, and so emphasize continuity and flowing quantities. Then we detect

atoms and numerical properties, and discontinuity once more makes its appearance. Then we invent the ether and are impressed with continuity again. But this is not likely to be the end; and what the ultimate end will be, or whether there is an ultimate end, is a question difficult to answer. The modern tendency is to emphasize the discontinuous or atomic character of everything. Matter has long been atomic, in the same sense as anthropology is atomic; the unit of matter is the atom, as the unit of humanity is the individual. Certainly, however, there is an illusion of continuity. We recognize it in the case of water. It appears to be a continuous medium, and yet it is certainly molecular. It is made continuous again, in a sense, by the ether postulated in its pores; for the ether is essentially continuous. Though Osborne Reynolds, it is true, invented a discontinuous or granular ether, on the analogy of the sea shore. Electricity itself—i. e., electric charge—strangely enough has proved itself to be atomic. There is a natural unit of electric charge, as suspected by Faraday and Maxwell and named by Johnstone Stoney. Some of the electron's visible effects were studied by Crookes in a vacuum; and its weighing and measuring by J. J. Thomson were announced to the British Association meeting at Dover in 1899—a fitting prelude to the twentieth century. Even magnetism has been suspected of being atomic, and its hypothetical unit has been named in advance the magneton; but I confess that here I have not been shaken out of the conservative view. Discontinuity does not fail to exercise fascination even in pure mathematics. Curves are invented which have no tangent or differential coefficient, curves which consist of a succession of dots or of twists; and the theory of commensurable numbers seems to be exerting a dominance over philosophic-mathematical thought as well as over physical problems. And not only these fairly accepted results are prominent, but some more difficult and unexpected theses in the same direction are being propounded, and the atomic character of energy is advocated. Then again radiation is showing signs of becoming atomic or discontinuous.

The forms of energy can be classified as either a translation, a rotation or a vibration of pieces of matter of different sizes, from stars and planets down to atoms and electrons; or else an *etherial strain* which in various different ways is manifested by the behavior of such masses of matter as appeal to our senses. Some of the facts responsible for the suggestion that energy is atomic seem to me to depend on the discontinuous nature of the structure of a material atom, and on the high velocity of its constituent particles. The apparently discontinuous emission of radiation is, I believe, due to features in the real discontinuity of matter. Disturbances inside an atom appear to be essentially catastrophic; a portion is liable to be ejected with violence. There appears to be a critical velocity below which ejection does not take place; and, when it does, there also occurs a sudden rearrangement of parts which is presumably responsible for some perceptible etherial radiation. Hence it is, I suppose, that radiation comes off in gushes or bursts; and hence it appears to consist of indivisible units. The occasional phenomenon of new stars, as compared with the steady orbital motion of the millions of recognized bodies, may be suggested as an astronomical analogue. The hypothesis of quanta was devised to reconcile the law that the energy of a group of colliding molecules must in the long run be equally shared among all their degrees of freedom, with the observed fact that the energy is really shared into only a small number of equal parts. For if vibration-possibilities have to be taken into account, the number of degrees of molecular freedom must be very large, and energy shared among them ought soon to be all frittered away; whereas it is not. Hence the idea is suggested that minor degrees of freedom are initially excluded from sharing the energy, because they cannot be supplied with less than one atom of it.

Now in all the debatable matters of which I have indicated possibilities I want to urge a conservative attitude. I accept the new experimental results on which some of these theories—such as the principle of relativity—are based, and am profoundly interested in them, but I do not feel that they are so revolu-

*Delivered at Birmingham on September 10, 1913.

tionary as their propounders think. I see a way to retain the old and yet embrace the new, and I urge moderation in the uprooting and removal of landmarks. And of these the chief is continuity. I cannot imagine the exertion of mechanical force across empty space, no matter how minute; a continuous medium seems to me essential. I cannot admit discontinuity in either space or time, nor can I imagine any sort of experiment which would justify such a hypothesis. For surely we must realize that we know nothing experimental of either space or time, we cannot modify them in any way. We make experiments on bodies, and only on bodies, using "body" as an exceedingly general term. We have no reason to postulate anything but continuity for space and time. We cut them up into conventional units for convenience sake, and those units we can count; but there is really nothing atomic or countable about the things themselves. We can count the rotations of the earth, or the revolutions of an electron, or the vibrations of a pendulum, or the waves of light. All these are concrete and tractable physical entities; but space and time are ultimate data, abstractions based on experience. We know them through motion, and through motion only, and motion is essentially continuous. Very well, then, what about the ether; is that in the same predicament? Is that an abstraction, or a mere convention, or is it a concrete physical entity on which we can experiment? Now it has to be freely admitted that it is exceedingly difficult to make experiments on the ether. It does not appeal to sense, and we know no means of getting hold of it. The one thing we know metrical about it is the velocity with which it can transmit transverse waves. That is clear and definite, and thereby to my judgment it proves itself a physical agent; not indeed tangible or sensible, but yet completely real. But it does elude our laboratory grasp. If we rapidly move matter through it, hoping to grip it and move it too, we fail: there is no mechanical connection. And even if we experiment on light we fail too. So long as transparent matter is moving relatively to us, light can be affected inside that matter; but when matter is relatively stationary to matter nothing observable takes place, however fast things may be moving, so long as they move together. Hence arises the idea that motion with respect to ether is meaningless; and the fact that only relative motion of pieces of matter with respect to each other has so far been observed is the foundation of the principle of relativity. It sounds simple enough as thus stated, but in its development it is an ingenious and complicated doctrine embodying surprising consequences which have been worked out by Professor Einstein and his disciples with consummate ingenuity.

Now the facts are that no motion with reference to the ether alone has ever yet been observed; there are always curious compensating effects which just cancel out the movement-terms and destroy or effectively mask any phenomena that might otherwise be expected. When matter moves past matter observation can be made; but, even so, no consequent locomotion of ether, outside the actually moving particles, can be detected. To detect motion through ether we must use an ethereal process. We may use radiation, and try to compare the speeds of light along or across the motion; or we might try to measure the speed, first with the motion and then against it. But how are we to make the comparison? If the time of emission from a distant source is given by a distant clock, that clock must be observed through a telescope, that is, by a beam of light; which is plainly a compensating process. Or the light from a neighboring source can be sent back to us by a distant mirror: when again there will be compensation. Or the starting of light from a distant terrestrial source may be telegraphed to us, either with a wire or without; but it is the ether that conveys the message in either case, so again there will be compensation. Electricity, magnetism, and light, are all effects of the ether. Use cohesion, then; have a rod stretching from one place to another, and measure that. But cohesion is transmitted by the ether, too, if, as believed, it is the universal binding medium. Compensation is likely; compensation can, on the electrical theory of matter, be predicted. Use some action not dependent on ether, then. Very well, where shall we find it? If the electrical theory of matter be true,

every material interaction will be electrical, i. e., ethereal; and hence arises our difficulty. Every kind of force is transmitted by the ether, and hence so long as all our apparatus is traveling together at one and the same pace, we have no chance of detecting the motion. That is the strength of the principle of relativity.

The changes are not zero, but they cancel each other out of observation.

The electrical theory of matter is a positive achievement, and has positive results. By its aid we make experiments which throw light upon the relation between matter and the ether of space. The principle of relativity, which seeks to replace it, is a principle of negation, a negative proposition, a statement that observation of certain facts can never be made, a denial of any relation between matter and ether, a virtual denial that the ether exists. Whereas if we admit the real changes that go on by reason of rapid motion, a whole field is open for discovery; it is even possible to investigate the changes in shape of an electron—appallingly minute though it is—as it approaches the speed of light; and properties belonging to the ether of space, evasive though it be, cannot lag far behind. The ether of space is at least the great engine of continuity. It may be much more, for without it there could hardly be a material universe at all. Certainly, however, it is essential to continuity; it is the one all-permeating substance that binds the whole of the particles of matter together. It is the uniting and binding medium without which, if matter could exist at all, it could exist only as chaotic and isolated fragments; and it is the universal medium of communication between worlds and particles. And yet it is possible for people to deny its existence, because it is unrelated to any of our senses, except sight—and to that only in an indirect and not easily recognized fashion.

I hold that science is incompetent to make comprehensive denials, even about the ether, and that it goes wrong when it makes the attempt. Science should not deal in negations; it is strong in affirmations, but nothing based on abstraction ought to presume to deny outside its own region. It often happens that things abstracted from and ignored by one branch of science may be taken into consideration by another. Yet none of these ignored things should be denied. Denial is no more infallible than assertion.

All intellectual processes are based on abstraction. Hence it is that science has no authority in denials. To deny effectively needs much more comprehensive knowledge than to assert. And abstraction is essentially not comprehensive; one cannot have it both ways. Science employs the methods of abstraction and thereby makes its discoveries. The reason why some physiologists insist so strenuously on the validity and self-sufficiency of the laws of physics and chemistry, and resist the temptation to appeal to unknown causes—even though the guiding influence and spontaneity of living things are occasionally conspicuous as well as inexplicable—is that they are keen to do their proper work; and their proper work is to pursue the laws of ordinary physical energy into the intricacies of "colloidal electrolytic structures of great chemical complexity" and to study its behavior there. What we have clearly to grasp, on their testimony, is that for all the terrestrial manifestations of life the ordinary physical and chemical processes have to serve. There are not new laws for living matter and old laws for non-living, the laws are the same; or if ever they differ, the burden of proof rests on him who sustains the difference. The conservation of energy, the laws of chemical combination, the laws of electric currents, of radiation, etc., etc.—all the laws of chemistry and physics—may be applied without hesitation in the organic domain. Whether they are sufficient is open to question, but as far as they go they are necessary; and it is the business of the physiologist to seek out and demonstrate the action of those laws in every vital action.

I observe that by some critics I have been called a vitalist, and in a sense I am; but I am not a vitalist if vitalism means an appeal to an undefined "vital force" (an objectionable term I have never thought of using) as against the laws of chemistry and physics. Those laws must be supplemented, but need by

no means be superseded. The business of science is to trace out their mode of action everywhere, as far and as fully as possible; and it is a true instinct which resents the mediæval practice of freely introducing spiritual and unknown causes into working science. In science an appeal to occult qualities must be illegitimate, and be a barrier to experiment and research generally; as, when anything is called an Act of God—and when no more is said. The occurrence is left unexplained. As an ultimate statement such a phrase may be not only true but universal in its application. But there are always proximate explanations which may be looked for and discovered with patience. So, lightning, earthquakes and other portents are reduced to natural causes. No ultimate explanation is ever attained by science: proximate explanations only. They are what it exists for; and it is the business of scientific men to seek them. So it is always in science, and its progress began when unknown causes were eliminated and treated as non-existent. Those causes, so far as they exist, must establish their footing by direct investigation and research; carried on in the first instance apart from the long recognized branches of science, until the time when they too have become sufficiently definite to be entitled to be called scientific. Outlandish territories may in time be incorporated as states, but they must make their claim good and become civilized first.

It is well for people to understand this definite limitation of scope quite clearly, else they wrest the splendid work of biologists to their own confusion—helped, it is true, by a few of the more robust or less responsible theorists, among those who should be better informed and more carefully critical in their philosophizing utterances. But, as is well known, there are more than a few biologists who, when taking a broad survey of their subject, clearly perceive and teach that before all the actions of life things are fully explained some hitherto excluded causes must be postulated. Ever since the time of J. R. Mayer it has been becoming more and more certain that, as regards performance of work, a living thing obeys the laws of physics, like everything else; but undoubtedly it initiates processes and produces results that without it could not have occurred. The behavior of a ship firing shot and shell is explicable in terms of energy, but the discrimination which it exercises between friend and foe is not so explicable. There is plenty of physics and chemistry and mechanics about every vital action, but for a complete understanding of it something beyond physics and chemistry is needed. And life introduces an incalculable element. The vagaries of a fire or a cyclone could all be predicted by Laplace's calculator, given the initial positions, velocities, and the law of acceleration of the molecules; but no mathematician could calculate the orbit of a common house-fly. A physicist into whose galvanometer a spider had crept would be liable to get phenomena of a kind quite inexplicable, until he discovered the supernatural—i.e., literally super-physical cause. I will risk the assertion that life introduces something incalculable and purposeful amid the laws of physics; it thus distinctly supplements those laws, though it leaves them otherwise precisely as they were and obeys them all.

What appears to be quite certain is that there can be no terrestrial manifestation of life without matter. Hence naturally they say, or they approve such sayings as, "I discern in matter the promise and potency of all forms of life." Of all terrestrial manifestations of life, certainly. How else could it manifest itself save through matter? "I detect nothing in the organism but the laws of chemistry and physics," it is said. Very well; naturally enough. That is what they are after; they are studying the physical and chemical aspects or manifestations of life. But life itself—life and mind and consciousness—they are not studying, and they exclude them from their purview. Matter is what appeals to our senses here and now; materialism is appropriate to the material world; and not as a philosophy but as a working creed, as a proximate and immediate formula for guiding research. Everything beyond that belongs to another region, and must be reached by other methods. To explain the psychical in terms of physics and chemistry is simply impossible; hence there is a tendency

to deny its existence, save as an epiphenomenon. But all such philosophizing is unjustified, and is really bad metaphysics.

But although life and mind may be excluded from physiology they are not excluded from science. Of course not. It is not reasonable to say that things necessarily elude investigation merely because we do not knock up against them. Yet the mistake is sometimes made. The ether makes no appeal to sense, therefore, some are beginning to say that it does not exist. Mind is occasionally put into the same predicament. Life is not detected in the laboratory, save in its physical and chemical manifestations; but we may have to admit that it guides processes nevertheless. It may be called a catalytic agent. To understand the action of life itself, the simplest plan is not to think of a microscopic organism, or any unfamiliar animal, but to make use of our own experience as living beings. Any positive instance serves to stem a comprehensive denial; and if the reality of mind and guidance and plan is denied because they make no appeal to sense, then think how the world would appear to an observer to whom the existence of men was unknown and undiscoverable, while yet all the laws and activities of Nature went on as they do now. Suppose, then, that *man* made no appeal to the senses of an observer of this planet. Suppose an outside observer could see all the events occurring in the world, save only that he could not see animals or men. He would describe what he saw much as we have to describe the activities initiated by life. If he looked at the Firth of Forth, for instance, he would see piers arising in the water, beginning to sprout, reaching across in strange manner till they actually join or are joined by pieces attracted up from below to complete the circuit (a solid circuit round the current). He would see a sort of bridge or filament thus constructed, from one shore to the other, and across this bridge insect-like things crawling and returning for no very obvious reason. If told that an engineer, a designer in London, called Benjamin Baker, had anything to do with it the idea would be preposterous. One conclusive argument is final against such a superstitious hypothesis—he is not there, and a thing plainly cannot act where it is not. But although we, with our greater advantages, perceive that the right solution for such an observer would be the recognition of some unknown agency or agent, it must be admitted that an explanation in terms of a vague entity called vital force would be useless, and might be so worded as to be misleading; whereas a statement in terms of mechanics and physics could be clear and definite and true as far as it went, though it must necessarily be incomplete.

And note that what we observe, in such understood cases, is an *interaction* of mind and matter; not parallelism nor epiphenomenalism, nor anything strained or difficult; but a straightforward utilization of the properties of matter and energy for purposes conceived in the mind and executed by muscles guided by acts of will. But, it will be said, this is unfair, for we *know* that there is design in the Forth Bridge or the Nile Dam, we have seen the plans and understand the agencies at work; we know that it was conceived and guided by life and mind, it is unfair to quote this as though it could simulate an automatic process. Not at all, say the extreme school of biologists whom I am criticizing, or ought to say if they were consistent, there is nothing but chemistry and physics at work anywhere; and the mental activity apparently demonstrated by those structures is only an illusion, an epiphenomenon; the laws of chemistry and physics are supreme, and they are sufficient to account for everything. Well, they account for things up to a point; they account in part for the color of a sunset, for the majesty of a mountain peak, for the glory of animate existence; but do they account for everything completely? Do they account for our own feeling of joy and exaltation, for our sense of beauty, for the manifest beauty existing throughout Nature? Do not these things suggest something higher and nobler and more joyous, something for the sake of which all the struggle for existence goes on?

Surely there must be a deeper meaning involved in natural objects. Orthodox explanations are only partial, though true as far as they go. When we examine each parti-colored pinnule

in a peacock's tail, or hair in a zebra's hide, and realize that the varying shades on each are so placed as to contribute to the general design and pattern, it becomes exceedingly difficult to explain how this organized co-operation of parts, this harmonious distribution of pigment cells, has come about on merely mechanical principles. It would be as easy to explain the sprouting of the cantilevers of the Forth Bridge from its piers, or the flocking of the stones of the Nile Dam by chemiotaxis. Flowers attract insects for fertilization, and fruit tempts animals to eat it in order to carry seeds; but these explanations cannot be final. We have still to explain the insects. So much beauty cannot be necessary merely to attract their attention. We have further to explain this competitive striving towards life. Why do things struggle to exist? Surely the effort must have some significance, the development some aim. We thus reach the problem of existence itself and the meaning of evolution. The mechanism whereby existence entrenches itself is manifest, or at least has been to a large extent discovered. Natural selection is a *vera causa*, so far as it goes; but if so much beauty is necessary for insects, what about the beauty of a landscape or of clouds? What utilitarian object do those subserve? Beauty in general is not taken into account by science. Very well, that may be all right, but it exists nevertheless. It is not my function to discuss it. No; but it is my function to remind you and myself that our studies do not exhaust the universe, and that if we dogmatize in a negative direction, and say that we can reduce everything to physics and chemistry, we gibbet ourselves as ludicrously narrow pedants, and are falling far short of the richness and fullness of our human birthright.

But if we have learned from science that evolution is real, we have learned a great deal. Surely evolution is not an illusion, surely the universe progresses in time. Time and space and matter are abstractions, but are none the less real; they are data given by experience, and time is the keystone of evolution. We abstract from living, moving reality a certain static aspect, and we call it matter; we abstract the element of progressiveness, and we call it time. When these two abstractions combine, co-operate, interact, we get reality again. It is like Poynting's theorem. The only way to refute or confuse the theory of evolution is to introduce the subjectivity of time. That theory involves the reality of time, and it is in this sense that Prof. Irgson uses the great phrase "creative evolution."

I see the whole of material existence as a steady passage from past to future, only the single instant which we call the present being actual. The past is not non-existent, however, it is stored in our memories, there is a record of it in matter, and the present is based upon it; the future is the outcome of the present, and is the product of evolution. Where inorganic matter alone is concerned, there everything is determined. Wherever full consciousness has entered new powers arise, and the faculties and desires of the conscious parts of the scheme have an effect upon the whole. It is not guided from outside, but from within; and the guiding power is immanent at every instant. Of this guiding power we are a small but not wholly insignificant portion. That evolutionary progress is real is a doctrine of profound significance, and our efforts at social betterment are justified because we are a part of the scheme, a part that has become conscious, a part that realizes, dimly at any rate, what it is doing and what it is aiming at. Planning and aiming are, therefore, not absent from the whole, for we are part of the whole, and are conscious of them in ourselves. Either we are immortal beings or we are not. We may not know our destiny, but we must have a destiny of some sort. Those who make denials are just as likely to be wrong as those who make assertions; in fact, denials are assertions thrown into negative form. Scientific men are looked up to as authorities and should be careful not to mislead. Science may not be able to reveal human destiny, but it certainly should not obscure it. Things are as they are whether we find them out or not, and if we make rash and false statements posterity will detect us—if posterity ever troubles its head about us. I am one of those who think that the methods of science are not so limited

in their scope as has been thought; that they can be applied much more widely, and that the psychic region can be studied and brought under law, too. Allow us, anyhow, to make the attempt; give us a fair field. Let those who prefer the materialistic hypothesis by all means develop their thesis as far as they can; but let us try what we can do in the psychical region, and see which wins. Our methods are really the same as theirs—the subject-matter differs. Neither should abuse the other for making the attempt.

Whether such things as intuition and revelation ever occur is an open question. There are some who have reason to say that they do. They are at any rate not to be denied off-hand. In fact, it is always extremely difficult to deny *anything* of a general character, since evidence in its favor may be only hidden and not forthcoming, especially not forthcoming at any particular age of the world's history, or at any particular stage of individual mental development. Mysticism must have its place, though its relation to science has, so far, not been found. They have appeared disparate and disconnected, but there need be no hostility between them. Every kind of reality must be ascertained and dealt with by proper methods. If the voices of Socrates and of Joan of Arc represent real psychical experiences, they must belong to the intelligible universe.

Although I am speaking *ex cathedra*, as one of the representatives of orthodox science, I will not shrink from a personal note summarizing the result on my own mind of 30 years' experience of psychical research, begun without predilection—indeed, with the usual hostile prejudice. This is not the place to enter into details or to discuss facts scorned by orthodox science, but I cannot help remembering that an utterance from this chair is no ephemeral production, for it remains to be criticized by generations yet unborn, whose knowledge must inevitably be fuller and wider than our own. Your president, therefore, should not be completely bound by the shackles of present-day orthodoxy, nor limited to beliefs fashionable at the time. In justice to myself and my co-workers I must risk annoying my present hearers, not only by leaving on record our conviction that occurrences now regarded as occult can be examined and reduced to order by the methods of science carefully and persistently applied, but by going further and saying with the utmost brevity, that already the facts so examined have convinced me that memory and affection are not limited to that association with matter by which alone they can manifest themselves here and now, and that personality persists beyond bodily death. The evidence to my mind goes to prove that discarnate intelligence, under certain conditions, may interact with us on the material side, thus indirectly coming within our scientific ken; and that gradually we may hope to attain some understanding of the nature of a larger, perhaps ethereal, existence, and of the conditions regulating intercourse across the chasm. A body of responsible investigators has even now landed on the treacherous but promising shores of a new continent.

Many scientific men still feel in pugnacious mood towards theology because of the exaggerated dogmatism which our predecessors encountered and overcame in the past. They had to struggle for freedom to find truth in their own way; but the struggle was a miserable necessity, and has left some evil effects. And one of them is this lack of sympathy, this occasional hostility, to other more spiritual forms of truth. We cannot really and seriously suppose that truth began to arrive on this planet a few centuries ago. The pre-scientific insight of genius—of poets and prophets and saints—was of supreme value, and the access of those inspired seers to the heart of the universe was profound. But the camp followers, the scribes and pliarisees, by whatever name they may be called, had no such insight, only a vicious or a foolish obstinacy; and the prophets of a new era were stoned. Now at last we of the new era have been victorious; we inherit the fruits of the age-long conflict, and the stones are in our hands. Let us not fall into the old mistake of thinking that ours is the only way of exploring the multifarious depths of the universe, and that all others are worthless and mistaken. The universe is a larger thing than we have any conception of, and no one method of search will exhaust its

treasures. Men and brethren, we are trustees of the truth of the physical universe as scientifically explored, let us be faithful to our trust. Genuine religion has its roots deep down in the heart of humanity and in the reality of things. It is not surprising that by our methods we fail to grasp it; the actions of the Deity make no appeal to any special sense, only a universal appeal; and our methods are, as we know, incompetent to detect complete uniformity. There is a principle of relativity here, and unless we encounter flaw or jar or change nothing in us responds; we are deaf and blind, therefore, to the immanent grandeur around us, unless we have insight enough to appreciate the whole, and to recognize in the woven fabric of existence flowing steadily from the loom in an infinite progress towards perfection, the ever-growing garment of a transcendent God.

Winona Stamp-Mill.*

By R. B. Seeber

From October, 1906, to November, 1907, the Winona mine shipped rock from one shaft to the Adventure stamp-mill. This rock showed a total copper content of about 20 lb. per ton stamped. A large part of the contained copper was fine or flaky, making extraction very difficult. Only a trifle over 13 lb. per ton stamped was recovered during this period. With such a low grade ore, it was obviously necessary to practice every economy if a profit was to be obtained. The transportation cost per ton stamped was 17.5 cents during this period, or considerably over 1 cent per pound on the copper obtained. If a stamp-mill were to be built on the mine location without otherwise increasing operating expenses evidently a large proportion of this expense might be eliminated.

There are two principal reasons for the location of stamp-mills away from the mine, namely: To provide ample water for washing and to provide room for disposal of tailings. To meet the water requirement, with the mill on the mine at Winona, a dam across the Sleeping River was necessary. For a two-head mill it would also be necessary to use at least 50 per cent of the water over and over, as the stream flow is only about 3,000,000 gallons per 24 hours.

To meet the sand-room requirements, some dewatering device and sand-stacking equipment would be necessary.

After considerable study it was decided that these requirements could be met without prohibitive first cost or undue operating expense and work on a two-head mill was begun May 22, 1909.

Selection of Mill Site

The location chosen was a hillside between No. 4 shaft, Winona, and No. 1 shaft, King Philip. On the line of the stamp-heads pipes were driven which, when stopped, indicated a layer of hard material at a depth of 40 feet below the intended plane of the stamp base. On this material the concrete foundation of each stamp-head was expected to rest. This foundation is cylindrical. A steel caisson, 15 ft. in diameter, was weighted with a concrete ring 3 ft. thick and sunk as a drop shaft. When the expected layer of hard material was reached it was found to be thin and to be underlain by at least 80 ft. of sand, some of which was wet. Sinking was therefore continued until a sufficiently hard layer of material was encountered at about 55 ft. below the intended plane of the stamp base. The concrete ring was blocked up and a mushroom foot of concrete built in. The top 7 ft. of the caisson and concrete foundation was made 22 ft. in diameter in order to accommodate the 16-ft. square base for the head casting. The castings below the rock (from mortar down) weigh about 80 tons. The total weight of the stamp is about 115 tons. When sinking the 15-ft. caisson it was found that 43 ft. of concrete ring would be held up by the friction of the sand against the outer surface of the steel shell. Including the 7 ft. of 22-ft. ring, this skin friction would care for at least 400 tons of total weight of about 1050 tons to be supported, (935 tons foundation; 115 tons stamp), leaving 650 tons to be supported on the

sand under the 20-ft. ring, 203 square ft. and the 20-ft. diameter base (314 square ft.) or a load of a little over 1.25 tons per square ft. on the total area of 517 square ft.

Velocity cards of the stamps show a blow of from 24 to 33 ft. tons. This is partly absorbed in crushing rock. As a foundation to assist in absorbing the balance there are supplied 80 tons of cast iron and 930 tons of concrete. After two years of use, no settling of foundations is in evidence and I am inclined to the belief that the foundations are much larger than is necessary.

Foundation walls were all of concrete, of a 1:3:5 mixture; all walls stand on dry, hard sand. The bin foundation walls were separated from the stamp foundations by tarred paper in order that any settling of the latter need not disturb the building foundations. The octagonal foundations for the rock bins were tied together, to a certain extent, by old wire cable.

Under the slime department floors eight tanks were built for settling the slimes and decanting the dirty water, to be reused in the mill. These tanks are 46 ft. long by 24 ft. wide, and 14 ft. deep. The walls are 14 to 24 in. thick and are reinforced by vertical iron rods $\frac{3}{4}$ in. in diameter, bent at the bottoms to make the joint into the floor.

It was very difficult to place and dump the concrete about



FIG. 1.—WINONA STAMP MILL

these rods, but the walls when completed showed but a few small leaks which were easily stopped and gave no trouble in operation.

The steel structure was made and erected by the Wisconsin Bridge & Iron Company. In general arrangement, it is the same as the usual copper country mill. Circular steel rock bins are used, two supplying each head, the openings being run together high up the sides thus allowing a large proportion of the contents to run out freely. The storage capacity of four bins is about 525 tons, or sufficient to last over night.

Instead of a trolley beam over the stamps a light crane beam was installed. This has proved very convenient in operating as well as during erection. This crane also handles all roll parts, etc.

Roof Construction

Over the slime department, at right angles to the line of the stamps, it was impossible to get in the usual step form of roof in a manner to allow sufficient light. Skylights of wire-glass were therefore tried and have proved satisfactory although snow sometimes accumulates over them. The roof is the usual form of matched flooring but is covered with Brooks' 4-ply asbestos roofing, in sheets about 3 ft. by 7 ft. During the first winter much trouble was experienced with ice on the eaves. As snow melts over the main body of the warm roofs the water runs down until it strikes the eaves, which are cold. It there freezes and makes a dam of ice which backs the water up until it runs through some opening or freezes and adds to the ice already formed. Immense icicles were sometimes formed

*Abstract of a paper read at the Lake Superior Mining Institute, Aug. 26-30, 1913.

down the sides of the building to the ground. This trouble has been entirely overcome by building wooden dams just back of the line of the eaves on each step of roof. These dams slope to holes in the roof leading to a piping system inside the mill which carries off the water. The warm air from the interior keeps these pipes free from ice, and as fast as snow melts it runs off the roof. This scheme was copied from a similar one at the Calumet & Hecla mills but I have seen no description of it in print.

The main slime department floor over the top of the settling tanks is made of concrete reinforced with triangle-mesh wire. The outside walls of the building are formed by a thick coat of cement plaster on a chicken-wire reinforcement. The inside face of the wall is made in the same manner, which leaves a good air space in the wall. The mortar is made of cement, sand, and a little lime.

Disposition of Tailings

Rough tailings from the tails of the jigs are de-watered by a large wheel and fed to a cross conveyor which discharges over the main conveyor running the long way of the mill. This main conveyor operates within a steel bridge supported on three steel towers and, when dumping, reaches a maximum height of 84 ft. above the track level at the bottom of the mill and 123 ft. above the ground. The conveyor is of the usual Robins type with troughing idlers on a 30-degree angle. The belt is balata, 20 in. wide. It has been in operation since March, 1911, and apparently has a long time yet to run before needing replacement. The conveyor is inclined $1\frac{3}{4}$ in. per ft., or 8 deg. 18 min.

A slime launder of steel extends 1200 ft. below the mill. This has a semi-circular bottom, with straight sides. We are now, (March, 1913) putting in the first steel liners. The wear is only around the rivet heads where eddy-currents are set up. This launder slopes $\frac{1}{4}$ on an inch to the foot and carries all the slime material. It empties into a ravine which joins the main river at a distance of about a mile and a half from the mill but on land belonging to the Winona Copper Company. If necessary, at any time in the future, a dam could be built across the river at this point and water pumped back to the mill.

A circular steel bin that can hold about four cars of coarse sand is situated over the track to the mineral bin. This can be filled from a separate belt conveyor parallel to the main conveyor and, during the summer months, much of the coarse sand is sold for concrete work and for railroad ballast.

The main trestle to the rock bins is used as a coal trestle. The coal plat is 30 ft. below the base of the rail. The floor is of concrete, about 4 in. thick. The usual coal adit under the main trestle is also made from concrete. The storage capacity of the plat is about 6000 tons. The trestle is of steel. The coal adit enters the boiler house at the same elevation as the ash adit in front of the boilers. An electric elevator elevates the car with ashes to an ash trestle and the car with coal to a trestle 8 ft. above the floor and running parallel to the boiler front. The coal supply is dumped on the floor and the boilers are fired by hand. The stack is of steel, brick lined. It is 150 ft. high and 6 ft. in diameter. The boilers were made by Parker, of Philadelphia. They are three in number, each 268 horsepower and are set in brick. They are equipped with Andrews' shaking grates, draft regulators and feedwater regulators.

The assay office is near the railroad at the bottom of the mill. It is supplied with the usual equipment for both fire and electrolytic assaying. A motor generator supplies the storage batteries for electrolytic work. A Tirrill gasoline gas plant supplies the gas.

Construction of Dam

A dam for the main water supply of the mill is situated on the Sleeping River, about 3900 ft. from the mill. It is 440 ft. long and 27 ft. high. The width of the bank on top is 15 ft. The slope on the water side is 2 to 1 and on the down stream side is $2\frac{1}{2}$ to 1. The spillway is 7 ft. wide and 8 ft. deep from the top of the concrete core wall. With all flash

boards out of the spillway, the storage capacity of the dam is about 77,000,000 gallons. Its volume is just about doubled by 5 ft. of flash boards. The core wall of the dam is concrete, 12 in. thick at the top and as much as 6 ft. at the very bottom. The sand fill was made mostly by the hydraulic process, a pump on the stream supplying water for washing sand from the banks. Both slopes are ripped up with coarse rock from the mine. A sheet steel intake 4 ft. square with a screen underneath, was first used but quickly became clogged with leaves, etc. A straight pipe 18 in. in diameter, the periphery filled with $1\frac{1}{2}$ -in. holes and rising above the highwater level, was then tried. This has given no trouble beyond catching in the ice once as the water level dropped.

The main pump is situated in a compressor house on the river bank. The pump was made by the Laidlaw-Dunn-Gordon Company. It has water cylinders $11\frac{1}{4} \times 24$ in. and steam cylinders 12 and 25 by 24 in. stroke. The cooling water for condensation is furnished by the main section of the pump. Water enters the suction under a head of 5 lb. Steam pressure is 200 lb. The pumping capacity, at $67\frac{1}{2}$ revolutions per minute, is 4,000,000 gallons per 24 hours.

The pipe line connecting the pump to the surge tank on top of the stamp-mill is spiral-riveted double galvanized pipe, with bolted points and rubber gaskets. The pressure at the pump is 86 pounds per square inch with the pump running. We have ruptured several lengths of this pipe, the rupture cutting clear across the steel. This was probably due to flaws in the steel. The joints have not given trouble. The pipe is 14 in. in diameter and 3900 ft. long. The surge tank on top of the mill supplies pressure for fire lines. All piping for the mill is direct from this tank.

Simple and Compound Steam Stamps

The steam stamps were made by the Allis-Chalmers Company. One is a simple stamp with a cylinder 24 in. diameter by 25 in. stroke and the other a compound with cylinders 16 and 32 in. diameter by 25 in. stroke. Both stamps have piston valves and only two eccentrics. The high-pressure cylinder of the compound stamp is on top and is removed bodily by the crane, if necessary to inspect the low-pressure cylinder or the piston. The rolls are of the rigid type and were made by the Allis-Chalmers Company. Four trommels are used instead of two.

The jigs are of the Woodbury system. One bull jig is used for the oversize and four four-compartment sets per head for the material through the trommels. Owing to the small percentage of No. 1 copper (medium-sized pieces) contained in the Winona rock, these jigs do not seem well adapted to the purpose. They do make an excellent separation of slimes for table treatment and provide a middling feed for regrinding mills. The jigs are supported on iron brackets instead of the usual timber supports. This makes it easier to get under the machines for adjustment and repairs and for washing floors.

De-watering Wheel

The principal machine developed for the operation of this mill is the de-watering wheel. As it is necessary to re-use the water, the tailings had to be separated from it. The water carrying the tails of the jigs is comparatively free from slime so that it is kept separate as "clear water" and re-used as wash water. The first de-watering device tried consisted of a screen tacked on a cylindrical frame which revolved slowly. The tails were drawn through spigots onto the outside of this screen the water falling through and the coarse material going over with the screen on to the cross conveyor belt. This scheme was not satisfactory as the spigots required a great deal of attention, having a tendency to either clog or to run water as the feed varied. An 8-ft. wheel along the lines of the present wheel was then developed. This worked well but the 12-ft. diameter wheel now used gives more room for launders, etc., and takes care of two heads. As shown by the drawing, Fig. 2, the de-watering wheel consists of a sheet steel water-tight wheel with radial partitions along the periphery forming pockets in which the sand is caught and lifted out of the water and

discharged at the top of the wheel over an apron on to the belt conveyor. The sand is run from settling tanks through spigots into the bottom of the wheel.

The water overflowing from the wheel is carried to the settling tanks and re-used.

The sand conveyor problem gave some little trouble before it was satisfactorily solved. The first cross conveyor used delivered sand to a bucket elevator which persisted in clogging and otherwise causing trouble. This was removed and the cross conveyor curved up to deliver directly to the main conveyor. Some trouble was experienced with the belts getting out of line but this was gradually eliminated until the sand

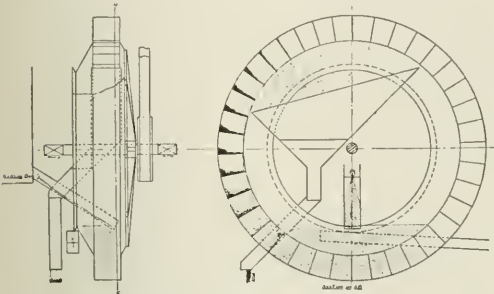


FIG. 2.—DEWATERING WHEEL

conveyor' equipment now has no special attendant. As a large pile of sand was piled up around the last tower of the sand conveyor the ground began to bulge up around the edge of the pile. Settlement of the outside tower came with this movement and finally caused shearing of rivets on the upper side of the conveyor bridge over the middle tower. These rivets were then cut out and bolts in slots substituted. A pin joint was placed in the bottom of the truss. This allowed for about 2 ft. of settlement in the outer tower. The settling continued until it became necessary to cut the conveyor house free from the outside tower. The conveyor bridge is now supported at the far end on blocking and jacks from the steel of the tower embedded in the sandpile. If settlement continues, the conveyor bridge will be jacked up to keep pace. As the sand accumulates the conveyor will be extended and supported on the sand.

As regrounding increases the amount of coarse sand to be stacked will decrease.

Regrinding with Hardinge Mills

Milling was started with one 8 x 30 in. Hardinge conical mill. During 1912 two more were installed, one a 6-ft. by 60-in. straight face and the other an 8-ft. by 18-in. straight face. For one period of January, 1912, no mill was at work. For most of the first six months two mills were in operation and for the last six months three mills were in operation. During the year about 36,000 tons of material was reground. This figure is of course not exact but from several tests and records of time, etc., I feel sure that it is very nearly right. From the reground material 215,248 lb. of refined copper was produced. The grade of this mineral will average better than 50 per cent. The total cost of regrounding during the year was \$9,697.09 or 26.93 cents per ton reground, and 4.5 cents per pound of copper recovered. With electricity costing 1.2 cents per kw-hour, the power cost was \$7,501.97 of the above total amount. Labor was \$402.85 and supplies cost \$1,792.27. Of the supplies, \$1,235.80 was for 172,030 pounds of French pebbles and \$280.68 for 19,342 pounds of silix lining. The balance was for oil and supplies incidental to repairs. The pebble loss figures nearly 5 lb. per ton reground as it includes the initial charge for two new mills. The pebble loss is now running a trifle under 4 lb. per ton reground, which is higher than usual in this district on account of the hardness of the rock.

Costs of Grinding 36,000 Tons

	Total	Per T'n	Per Cent	Units per Ton
Power 1.2c. per kw. hr.	\$7,501.97	20.84c	77.36	17.415 kw hrs
Labor	402.85	1.12	4.15	
Supplies—				
.7184c. per lb. pebbles	1,235.80	3.43	12.74	4.78 lb. pebbles
1.4512c. per lb. lining	280.68	.77	2.90	.537 lb. lining
Incidentals	275.78	.77	2.85	
	\$9,697.09	26.93c	100.00	

The 8-ft. diameter by 30-in. straight face mill has proved the most economical in power cost per ton ground and also has the largest capacity of any we have tried. We are now installing three more 8-ft. mills which have a 36-in. straight face and we expect still greater capacity. These mills have Falk's cut herringbone gears and there is only one speed reduction, which will materially reduce the power cost. The first mill installed has a silix lining but the subsequent mills are lined on the straight face with pebbles set in grooves in steel plates. The conical faces are lined with silix blocks. Steel-pebble linings have proved very satisfactory. A mill is out of service at least four days to replace silix lining as the cement must be allowed to set. With a steel-pebble lining, the mill can go into service again as soon as new plates are placed, a matter of hours only. So far, our experience shows that the steel-pebble linings are cheapest. All our new mills will be equipped with the steel-pebble linings on the cones as well as the straight faces.

The mill is lighted at night, principally, by 100 candlepower series Tungsten incandescent lamps on a 6.6 ampere constant current. These have been found satisfactory, the life of the lamps running above 2000 hours.

The Winona stamp-mill was built on the mine to reduce operating expenses per ton by reducing transportation charges. Seventeen and one-half cents per ton was cut out of the transportation charge and the following items were added:

Stacking sand	1.2c. per ton stamped
Transportation on the mine	3.6c. per ton stamped

This still leaves an important net saving due to location of mill on the mine. The unit costs given are necessarily high

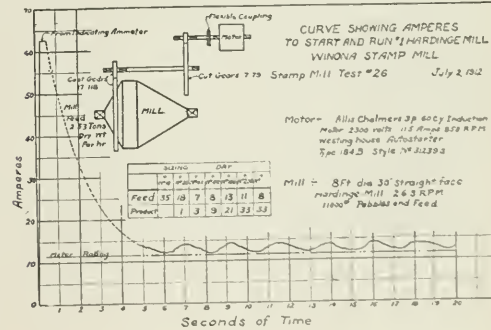


FIG. 3.—TEST CURVES

due to the small tonnage handled, namely, 181,184 tons for the year 1912. The detail of these costs follows:

Cost of belt and conveyor idlers, about \$4,000 erected:	
life 40 months; cost per month	\$100.00
Power at the rate of 8 kw at 1.2c. per kw-hour	60.00
Attendance	10.00
Oils, etc.	10.00
	\$180.00

Or 1.2 cents per ton on 15,000 tons stamped. These will of course be reduced materially with increased tonnage handled. Of the power used, at least two-thirds is in friction.

In addition, the following interest and depreciation charges might be listed against this operation:

Cost of conveyor bridge and towers, fully equipped with belt and machinery	\$12,000.00
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Interest at 6 per cent.....	720.00
5 per cent depreciation on \$8,000 of this amount....	400.00

The depreciation of the other \$4,000 is already accounted for in the working cost.

Pumping costs should not be increased over similar costs with a stamp-mill on Lake Superior as while the water re-handled is pumped with less efficient machinery the head against which it is pumped is very materially reduced. The pumping cost taken from our cost sheet for the year 1912 is 2.3 cents per ton stamped. This is materially higher than the usual figure on Lake Superior, owing, principally, to the smaller tonnage stamped.

There is undoubtedly considerable gained by the concentration of all operations at the mine. Some of this gain cannot be expressed in cents per ton. During part of the month of March, 1913, railroad service in Houghton county was very much hindered by heavy storms. Several of the mines were shut down temporarily and freight train service, at least, was cancelled for days at a time. Neither the Winona mine nor the mill was delayed on this account as our tracks are comparatively short and easily kept clear of snow.

Quantitative Spectrum Analysis

By G. A. Shook

II Spectrophotometers

Whenever white light, such as sunlight or the light from any incandescent solid, is passed through a prism, it is broken up or dispersed into a continuous band of colors which is known as the *spectrum* of white light.

The spectrum of a gas consists of bright lines which are more or less well defined; for example, when a salt of sodium is vaporized it produces a yellow flame and if this flame is examined by means of a small spectroscopic its spectrum is observed to consist of one bright yellow line.

If a colored solution, say copper sulphate, is placed before the slit of the spectroscopic we obtain the *absorption spectrum* of copper sulphate. The solution does not transmit simply one blue line, but to some degree all the colors, the blue region, of course, being the brightest. The intensity diminishes rapidly in the orange and yellow region and is relatively weak in the red region.

A cobalt solution, on the other hand, gives an absorption spectrum which is more intense in the red than in the blue.

In the case of most organic compounds, however, the intensity does not vary continuously from color to color, but the bands are more numerous and generally very much narrower than the bands observed in the spectrum of a simple solution like copper sulphate.

In the first article of this series, published in the September issue of this journal, it is shown that the concentration of a solution is directly proportional to the logarithm of that fraction of light transmitted by unit layer of the solution, that is

$$\log (I'/I) = kc$$

or

$$c = -\log (I'/I) = Ae \quad (1)$$

where I is the intensity of the incident light and I' the intensity of the transmitted light.

e is known as the *extinction coefficient* and A the *absorption ratio*. The concentration c will be expressed in per cent in all the calculations that follow. A 10 per cent solution here means that 10 grams of solute are dissolved in 100 c.c. of solvent.

Any notation may, of course, be used so long as one is consistent.

In many of the best instruments the intensity is controlled by means of a slit and the screw head which actuates the slit is divided into 100 divisions. When the head is turned to the 100 mark the slit is opened 1 mm. It will, therefore, be found convenient, in some cases, to express the fraction of the light transmitted, I'/I , in per cent.

From the theory of logarithms we have the following theorems:

$$\log A/B = \log A + \log B,$$

$$\log (A/B) = \log A - \log B$$

and

$$-\log (A/B) = \log B - \log A.$$

We may now write

$$e = -\log (I'/I) = \log I - \log I'.$$

Suppose, for example, that 20 per cent of the incident light is transmitted through a unit layer of a solution, then

$$e = \log 20/100 = \log 100 - \log 20 = 2 - 1.301 = 0.699.$$

Therefore, whenever (I'/I) is expressed in per cent

$$e = 2 - \log I' \quad (2)$$

and

$$c = Ae = A(2 - \log I') \quad (3)$$

These relations are only strictly true for homogeneous light, i.e., light of one color, so that in accurate work it is best to use a *spectro-colorimeter* or a *spectrophotometer* for the optical determination of concentrations.

In the spectro-colorimeter the light from the two containing vessels is rendered monochromatic either by means of a prism system or a number of monochromatic glasses. The latter method allows more light to be transmitted, but is not so accurate as the former.

In the type of spectrophotometer which is especially adapted for the determination of concentrations, the light from a suitable source of illumination such as a frosted globe, glow-lamp, or an Auer burner, is divided into two beams and one of these passes through a layer of the solution to be investigated. The transmitted beam and the uninterrupted beam then traverse a prism system which causes each to be drawn out into a spectrum or a band of colors and thence they are transmitted through some sort of a photometric screen to the eye.

By means of a diaphragm a narrow region, only, of the spectrum is transmitted to the eye. Somewhere in the path of the two beams is inserted a device for bringing them to equal intensity and this device gives us a measure of the relative intensity of the two. The ratio of the intensity of the transmitted beam to the intensity of the uninterrupted beam gives us at once the extinction coefficient of the solution in question for the particular color or wave-length of light employed. The commercial instruments accomplish these various ends in different ways and we shall therefore consider some of the typical instruments in detail.

The factor A is, of course, different for each color, so that if it is determined for a particular color by means of a known extinction coefficient and a known concentration, in all subsequent work that particular color must be employed.

There is no mathematical relation between the extinction coefficient and the wave-length of light, but a relation may be determined experimentally by means of a spectrophotometer.

Let us consider a simple blue solution, for example, a salt of copper. The general character of the absorption curves for different concentrations is shown in Fig. 1.

The curves illustrated in the figure show the relation between the extinction coefficient and the color. The extinction coefficient is a measure of the absorption of the solution.

There are several facts to be observed about these curves which are of fundamental importance to *spectro-colorimetry*. Since the solution is blue the blue region of the spectrum is of course least absorbed while the absorption in the red is the greatest.

A solution of zero concentration would transmit 100 per cent of the light incident upon it, neglecting the absorption of the solvent, and its extinction coefficient e would be $e = 2 - \log 100 = 0$. Its absorption curve would consequently be represented by the horizontal line A . A very dilute solution, say a concentration of 1 per cent, would transmit nearly 100 per cent in the blue but might transmit considerably less in the red, hence the shape of the 1 per cent curve.

Since e is directly proportional to c there is a constant difference between the successive concentration curves as shown in the figure. For instance, along the vertical line, B , there is the same interval between curve 1 and 2 as between 2 and 3, etc. The same is true for the vertical line C or D , but the

difference here in the red is considerably greater than in the blue region, as might be expected.

It is therefore clearly seen that in working with weak blue solutions more accurate results could be obtained if the red part of the spectrum were utilized instead of the blue. With the ordinary colorimeter, what the eye sees is the resultant effect of all these colors, and since the blue predominates the solution appears blue.

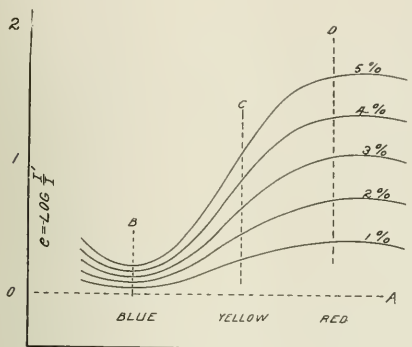


FIG. 1.—ABSORPTION CURVES FOR A BLUE SOLUTION

The advantage of the spectrophotometer or the spectro-photometer for accurate work is readily seen. The spectro-photometer has this decided advantage over the spectrophotometer, namely, that no standard solution is required. Moreover, only a small sample of the solution whose concentration is desired is necessary. The spectrophotometer may also be used for very dense and for very turbid solutions.

Cells

In making a photometric comparison of two light sources it is desirable that the two fields of view be uniform in intensity and that the boundary line between them be as small as possible. It is impossible to make an accurate setting when the two fields are separated by a line of any appreciable width.

Some instruments employ a simple rectangular glass cell; *a*, Fig. 2, and utilize the liquid surface of the solution to divide the incident beam. Due to the meniscus, however, the dividing line between the fields will not be narrow and this cell is therefore unfit for accurate work, without some modification.

A glass cube, *b*, Fig. 2, with optically plane sides is sometimes employed in which case the top surface of the cube forms the dividing line. This cell is often used where it is possible by special optical devices to separate the two beams a small amount before they reach the slit.

The third cell, *c*, Fig. 2, has a curved bottom and the cube which is used with it also has a curved bottom so that when

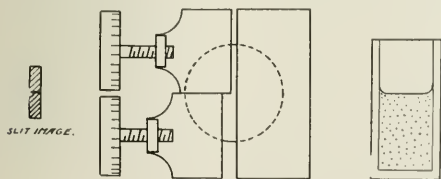


FIG. 3.—DOUBLE-SLIT APPARATUS

the latter is placed in the solution the sides of the cell and the sides of the cube will always be parallel.

The sides of these cells are either held together by cement or by means of clamps. The latter form of cell is more easily cleaned, and for some solvents this is an advantage.

In the case of volatile solutions it is well to use stoppered cells, *d*, Fig. 2.

The Krüss Universal Spectral Apparatus

This instrument is similar to the ordinary prism spectrometer except that the collimator slit is double and it is also provided with an ocular slit which excludes all of the spectrum except the desired color.

Each of the slits is actuated by means of a screw as shown in Fig. 3 and the screw heads are divided into 100 divisions. The cell is placed directly in front of and close to the slit face in such a manner that the level of the liquid is in line with the dividing line between the two slits.

Upon looking through the eye-piece, with the ocular slit removed, one sees two spectra separated by an ill-defined line. If the cell with the cube is employed this line may be made narrower and considerably sharper.

Now if the ocular slit is inserted and opened to a width of about 4 or 5 mm., two uniform fields of homogeneous light may be obtained. The ocular slit is, of course, magnified by the eye-piece, but in order to make a good setting it must not

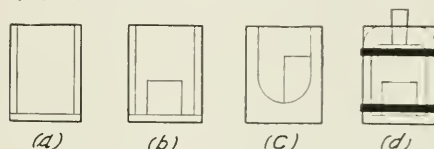


FIG. 2.—DIFFERENT TYPES OF GLASS CELLS

be too narrow. Moreover, homogeneous light is not obtained if the slit is open too wide. For example, if an observation were made upon a band narrower than the width of the slit then it is readily seen that the field formed by the band would not be uniform, but would fall off in intensity to either side.

Care must also always be taken to avoid setting the ocular slit upon the edge of a band where the intensity changes rapidly. For instance, suppose that a particular blue solution (Fig. 1) were so concentrated that the red region of the absorption spectra could not be used. The yellow region would, of course, transmit more light and consequently make the photometric setting possible, but a slight change in the wavelength would introduce a considerable change in the intensity so that if the instrument were, for subsequent work, reset for the same color, this adjustment would have to be made with extreme care.

In the red or blue region, however, the curves are nearly horizontal so that a considerable change in color is required to produce a significant error in the intensity.

With the type of slit shown in Fig. 3, a slight error will result if the difference in the widths of the upper and lower slit is too great and it is consequently advisable to use a symmetrical slit, *i.e.*, one in which both jaws of each slit open an equal amount, Fig. 4.

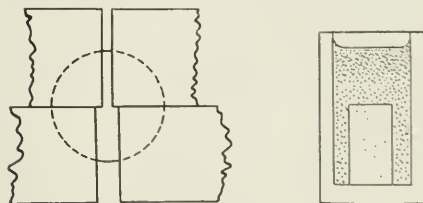


FIG. 4.—SYMMETRICAL DOUBLE SLIT

The scale which is attached to the telescope, or observing tube, of most of these instruments is either divided into circular degrees or arbitrary divisions so that the wave-length of light used cannot be read directly from the scale. This is not necessary, however, for it does not enter into any calculation and therefore an arbitrary scale is sufficient.

If, for example, the telescope reads 59.3 when the absorp-

tion ratio is determined for copper sulphate then the telescope must always be set for 59.3 when this particular absorption ratio is used for the determination of the concentration of a copper sulphate solution.

Sometimes the principal solar absorption lines A, B, C, D, etc., are also engraved upon the scale, but they only serve as a rough universal scale. When one observer makes the statement that in a particular process the absorption ratio for a given substance should be determined in the region of the D line, then the process may be readily duplicated by the user of any type of instrument.

The method of making an observation with any double slit instrument is briefly as follows:

Both slits are opened to any convenient width, say to the 30 mark, and then the source of illumination is adjusted until the two spectra, as observed through the eye-piece, are equally bright. This adjustment must then be verified by resetting one of the slits, say the upper one, several times.

If the average of the readings thus obtained is 30 or very nearly 30 then the initial adjustment is sufficiently close. If, however, it deviates too much from 30, then the lamp must be shifted slightly and another series of readings obtained by operating the slit. This operation must be carried out until the upper slit reads the same as the lower one.

Both slits are then opened to the 100 mark and the cell containing the solution is placed in front of the slit face and adjusted until the level of the solution is in line with the dividing line between the upper and lower slit.

If the light is not too bright both slits may be opened to 100 instead of 30 and moreover if, for subsequent work, all the photometric balancing is to be done with the upper slit only, then it is, of course, not necessary that the lower one read 100 or any other particular number, so that when the lamp is roughly adjusted the upper slit may be set to 100 and the final adjustment may be carried out by means of the lower slit.

This method of making the initial balance is evidently much easier than adjusting the lamp. In making this initial balance the ocular slit may be set on any convenient color since both spectra are due to the same source and are therefore identical.

When the cell, containing a colored solution, is in position the lower spectrum will be weakened and in order to bring both fields to the same intensity it will be necessary to narrow the upper slit.

Suppose that an intensity balance is obtained when the upper slit is set at 30; this means that 30 per cent of the incident light is transmitted by the solution and, therefore,

$$I'/I = 30/100 \text{ or } I' = 30$$

whence from (2)

$$e = 2 - \log I' = 2 - \log 30 = 0.523.$$

If the concentration is 4 per cent then the absorption ratio is

$$A = e/c = 4/0.523 = 7.65.$$

Now imagine that an unknown concentration of the same solute and solvent requires a reading of 24.0, on the screw-head of the upper slit for a balance, the extinction coefficient then becomes

$$e = 2 - \log 24.0 = 0.620$$

and the unknown concentration is, therefore,

$$c = A/e = 7.65 / 0.620 = 4.74.$$

This simple method admits of easy calculation but for wide ranges of concentrations it is sometimes necessary to manipu-

late both slits. For extreme concentrations, the upper slit would have to be made so narrow that an accurate adjustment is impossible, since a very narrow slit means, of course, a considerable loss of light.

If we had a dense solution which would affect only a 3 per cent transmission, instead of diminishing the upper slit to 3 we could set the lower slit to 300 and diminish the upper one to 9.

If several different widths of cells are used and if the absorption ratio is determined for each then all the adjusting may be done with one slit. Since the concentration varies inversely as the length of the absorbing layer, if the absorption ratio is determined experimentally for one cell, it may be calculated for any other. For example, if the absorption ratio, A , is 8.24 for a 10 mm. cell, then for a 20 mm. cell it would be equal to 4.12.

Whenever the glass cube is used, the adjusting is, of course, always done with the lower slit.

When the two uninterrupted beams are first brought to equality, it is not necessary to have the empty cell in place while adjusting the lamp, since it causes the same absorption of each beam. However, if the glass cube is used then its absorption will introduce a slight error which may be easily avoided by adjusting the intensity of the two spectra with the empty cell and cube in position. The error, however, is not worth considering when only approximate results are required since the transmission coefficient of 1 cm. of glass is about 98 per cent.

Let us suppose that the instrument is first balanced without the cell in position and that the filled cell, with cube, requires for a balance a reading of 40 on the lower slit when the upper slit is set at 100. Neglecting the glass cube we obtain for the extinction coefficient

$$e = 2 - \log 40 = 0.398.$$

But since the cube transmits only 98 per cent of the light incident upon it and not 100 per cent, the true extinction coefficient is

$$e = \log 98 - \log 40 = 0.380.$$

In refined work the cell should always be filled with the solvent to be used and should be placed in position when the initial balance is made.

In order to eliminate the trouble of carefully adjusting the cell each time so that the top face of the cube is in line with the division line between the slits, a Schottlander double collimator or a Hüfner reflecting prism is sometimes used in connection with the Universal spectrometer, but as there are more precise instruments which do not require this delicate cell adjustment, these rather complicated appliances need not be considered.

Lummer-Brodhun Spectrophotometer

The usual form of the Lummer-Brodhun spectrophotometer is shown in *a*, Fig. 5.

A and *B* are two collimators, the slits of which are illuminated by two frosted globe incandescent lamps (16 c.p., 110 volts). The telescope *D* is provided with a slit instead of an eye-piece so that on placing the eye close to this slit one sees the face of the Lummer-Brodhun cube, *C*, which has the appearance shown in *b*, Fig. 5.

The dispersive prism, *P*, spreads the light into a spectrum so that there is a succession of colored images of the cube in the plane of the ocular slit. Since, however, the slit transmits only a narrow region of the spectrum, the eye sees the cube illuminated in only one color.

The cell *S* may be placed before either slit and since the two slits are independent of each other the illumination may be controlled by moving either lamp to or from the slit it illuminates. Thus with the cell, filled with the solvent, in front of one of the slits both may be set to the 100 mark and the initial balance made by adjusting the lamps or by simply manipulating the slit which receives the absorbed light from the cell. The other slit must, of course, be kept at the 100 mark.

The disadvantage of this type of photometer over that in

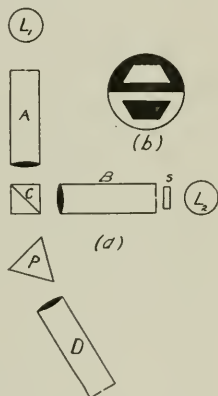


FIG. 5.—LUMMER-BRODHUN SPECTROPHOTOMETER

which both slits are illuminated by the same source is that the two lamps must be kept constant, but in the latest form of this instrument the lamps are rigidly clamped to the frame of the instrument so that the only change in intensity that may occur is that caused by fluctuations of the voltage. Although both lamps are on the same circuit and subjected to the same change in voltage, it does not follow, of course, that the corresponding change in intensity is the same in each lamp. This would be the case if both lamps were practically similar and it is not difficult to select two lamps which will fulfill all industrial requirements.

If the lamps appear equally bright to the unaided eye and if they balance for all parts of the spectrum there will be no appreciable difference in their intensity due to an increase or decrease of the voltage. The initial balancing of the two fields will then hold good for a number of hours of constant use.

With this type of Lummer-Brodhun spectrophotometer it is only necessary to have the solution cover one of the slits and the time saved in adjusting the cell more than compensates for the time required to rebalance the two fields, in order to make sure that a change of voltage has not disturbed the original balance.

In order to make a photometric balance with the type of Lummer-Brodhun cube shown in *b*, Fig. 5, the light from one source is varied by adjusting one slit until the semi-circular portions have the same brightness, and when this condition obtains the two quadrangles will be equally bright, but there is always a slight difference in their brightness and the bright-

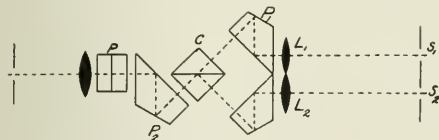


FIG. 6.—LUMMER-BRODHUN DOUBLE COLLIMATOR

ness of the semi-circular backgrounds. Such an arrangement is known as a *contrast photometer* and it is much more sensitive than the ordinary type.

The observing tube is provided with a scale which is read by means of a small telescope and miniature 6-volt glow-lamp. The lamp is usually run on three storage battery cells and as the scale cannot be seen without the aid of this lamp the arrangement is not practical since all commercial laboratories are not provided with storage batteries. The screws actuating the slits are not within easy reach when the eye is at the observing tube and moreover the divisions on the screw heads cannot be read from the position occupied by the observer when looking through the observing tube.

These disadvantages are overcome in some other instruments which lack the accuracy of the Lummer-Brodhun. To be sure these points have nothing to do with the working principle of the instrument; however, they are very essential to the operator whose time is valuable and the failure to appreciate their importance has in many instances retarded the progress of such methods in industrial laboratories.

For industrial purposes it is far better to employ an instrument by means of which one may obtain quickly a large number of fairly accurate observations than to employ a very sensitive instrument which requires more time to make a very few extremely accurate observations. In using any instrument which requires the eye to match colors or balance intensities the best method of making certain that your results are correct is to repeat the observations.

By means of a double collimator and reflecting prisms, the Lummer-Brodhun cube may be adapted to the double slit photometer. The optical system of the Lummer-Brodhun double slit instrument is shown in Fig. 6.

Light from a suitable source of illumination passes through the double slit S_1 and S_2 , is rendered parallel by means of the

two lenses L_1 and L_2 and thence passes to the Lummer-Brodhun cube C , by means of the reflecting prism P_1 . By means of a second reflecting prism P_2 the light is transmitted through the dispersive prism P which draws out the image of the Lummer-Brodhun cube into a spectrum.

If the lamp is fastened rigidly to the frame of the instrument then a change in its intensity, due to a change in voltage,

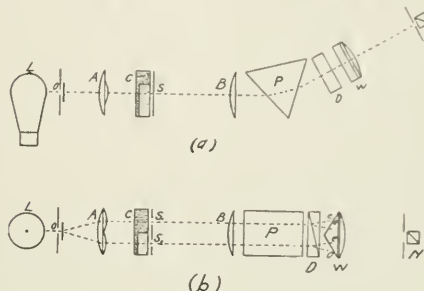


FIG. 7.—KÖNIG-MARTENS SPECTROPHOTOMETER

will not cause any error in the determination of e since the light through both slits is changed in the same ratio.

König-Martens Polarization Spectrophotometer

The optical system of this instrument is shown in Fig. 7; *a* is a horizontal section through the instrument and *b* a vertical section.

Light from a suitable lamp L passes through a small aperture O which is covered with ground glass. By means of a double lens A two parallel beams are produced which illuminate the two slits S_1 and S_2 . A cell of the type shown in *c*, Fig. 2, is placed in front of the two slits.

The slits are thus always equally illuminated regardless of the position of the lamp, since the two beams originate from the same source of light. If the lamp were moved slightly to one side, the ground glass would still be uniformly illuminated. In reality the two slits are one which is divided by means of a tongue, hence they are always open the same amount.

From the slits the light is again rendered parallel by a lens B and by means of a dispersive prism, P , the slit images are spread out into two spectra, *a*, Fig. 8, which are separated several millimeters.

These two spectra are next divided into four, *b*, Fig. 8, by means of a double image prism D . The ordinary and extraordinary images thus produced are plane polarized with their vibrating planes mutually perpendicular. The short horizontal and vertical lines in *b*, *c* and *d* indicate the direction of vibration.

By means of a wide angle wedge W , which is part of the telescope objective, the two inner images are thrown together, *c*, Fig. 8, while the outer images are excluded from the telescope.

When the eye is placed at the ocular slit, which is in the

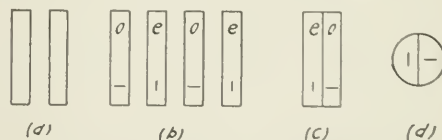


FIG. 8.—SPECTRA PRODUCED BY THE KÖNIG INSTRUMENT

focal plane of the telescope objective, it sees the wide angle wedge, W , illuminated by light of one color. As the telescope or observing tube is rotated about an axis through the prism P , thus causing the ocular slit to pass from one end of the spectra to the other, the field is alternately illuminated by the prismatic colors, red, yellow, green, etc. The field of view has the appearance shown in *d*, Fig. 8.

If a Nicol prism, *N*, Fig. 7 (*i.e.*, a prism which transmits only plane polarized light), is placed between the eye and the ocular slit with its transmission plane vertical, the left field will appear bright, but the right one dark, since it is illuminated by light whose vibrations are perpendicular to the transmission plane of the Nicol.

When the Nicol is placed with its transmission plane horizontal it will transmit light from the right field but exclude all light from the left one so that the left field appears dark.

Now if the Nicol is turned 45 deg., both fields will appear equally bright and for any other position of the Nicol the ratio of the intensity of the light from *S*₁ to the intensity of the light from *S*₂ is expressed by the simple relation

$$I_1/I_2 = \tan^2 \phi$$

If the cube covers slit *S*, then the cotangent is used instead of the tangent.

From the theory of extinction coefficients

$$e = -\log (I'/I).$$

But I_1/I_2 is the ratio of the transmitted light to the incident light if the instrument is balanced when the Nicol scale reads 45 deg. and the empty cell is in the position indicated, hence the extinction coefficient is

$$-\log (I_1/I_2) = -\log \tan^2 \phi = 2 \log \cot \phi.$$

The absorption ratio *A* is a constant so that the factor 2 may be dropped. For this particular instrument we may define the extinction coefficient by the equation

$$e = \log \cot \phi.$$

The concentration then becomes

$$c = A e = A \log \cot \phi.$$

The log cotangent of the angle, through which the Nicol must be turned in order to produce a photometric balance, may be obtained directly from a logarithmic table so that when the reading of the Nicol scale is obtained the extinction coefficient is readily determined.

The method of determining the concentration of a particular solution by means of observed data may be made clear by a simple calculation. Suppose a 10 per cent solution gives a reading of 26.4 (*i.e.*, 26 deg. 24 min.) on the Nicol scale. The extinction coefficient is, from the above equation

$$e = \log \cot \phi = \log \cot 26.4 = 0.304$$

and the absorption ratio is therefore

$$A = c/e = 10/0.304 = 32.9.$$

If an unknown solution gives a reading of 18.6 deg. its concentration is found to be

$$c = A e = 32.9 \log \cot 18.6 = 15.5 \text{ per cent.}$$

The Nicol prism is attached to a circular disc, *D*, Fig. 8, which is divided into angular degrees and this scale is observed by means of a small magnifier *M* which is close to the eye-piece *E*.

The telescope, or observing tube, is rotated by means of a screw *S*, the head of which moves over a scale divided into arbitrary divisions. This scale is used for indicating a particular position in the spectrum.

This instrument has several distinct advantages over the others described. The two slits have always the same width and this prevents any asymmetry of spectra. The device for varying the intensity is within easy reach and the attached scale may be quickly and accurately read by moving the eye a short distance. The screw that rotates the observing tube is also within easy reach and the observing tube scale may be easily

read from the position occupied by the observer when making a photometric balance. All of the optical parts are completely enclosed and this makes the instrument more robust. Moreover, all extraneous light is excluded.

Due to the rather complicated optical system, however, this instrument only transmits about one-third as much light as the double-slit spectrophotometer and this fact renders it unsuitable for very concentrated solutions. This difficulty may be overcome to some extent by using cells of short length. In the most improved form of this spectrophotometer, rather long cells may be used so that observations may be made on very dilute solutions. It is thus seen that the instrument has considerable range.

Both the double image prism and the wide-angle wedge may be rotated slightly by means of set-screws, *A* and *B*, Fig. 9, so that when the Nicol is set at 45 deg. the two fields may be brought to equality. It is never necessary to repeat this adjustment after the instrument is once set up, unless it receives some rough handling which might disturb the optical system.

The method of completely calibrating a spectrophotometer and some industrial applications of the spectrophotometer will be fully considered in a concluding article.

University of Illinois,
Urbana, Illinois.

Mill Physiology

By Stephen L. Goodale

Many technical articles on mills and ore dressing deal with plan and structure, machine details, etc., or what may be called mill anatomy; and the literature contains formidable masses of tabulated data on mill details from all over the world. All this is of the greatest value in the study of the subject, but it is not probable that too much emphasis has been laid upon the descriptions of machines in themselves, flow sheets, sizes of launders, grading analyses, screen sizes, stamp weights, etc.? Other types of discussions must doubtless be built upon the simple mill descriptions.

One kind of study, that of the mill in operation in its parts and as a unit, its "life processes," which I may call "mill physiology and hygiene," is occasionally emphasized in print as it is forced upon the millman's attention in practice; and is, I believe, of the greatest value to those students who can make real use of it—in some such way as a few good ladies are able to make successful pie from the cook book instructions without experience, and if they have a good cook book.

I am interested in this idea just now from the point of view of trying to offer students the most practical instruction along these lines. I am strongly of the opinion that much more can be learned of these practical affairs from "book larnin'" than is usually the case. The possibility of this depends, of course, largely on the student himself, and his ability to use books, but also in a very important way upon the literature available to the man.

In papers on gold milling there is often found a description of the make-up of the plant and not much more, and it is left to the reader to infer that because the selection and arrangement of machinery are of a certain kind, they are as they ought to be. Certainly this inference is not always justified, although perhaps not greatly in error if one is studying a successful mill.

Let us go a step beyond this, and inquire what the mill does in action. How do the various pieces of apparatus behave, each of itself, and how do they fit together in operation? What has been the reaction of the operation of the mill upon the mill itself, and how has it grown to what it is from what it was to begin with? For an instance, because of what overflows of launders or stopped-up launders or both have they been reconstructed to their present size, outline and grade?

In this paper I can but outline a few illustrations of this, leaving to the imagination of my readers the task of filling in such details as they wish. It is some time since I have been able to watch closely for a length of time the operation and development, the daily behavior and misbehavior, and

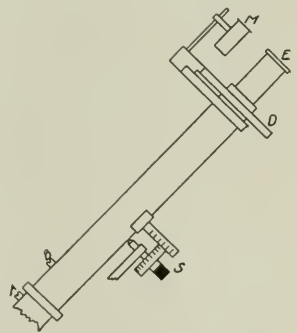


FIG. 9.—OBSERVING TUBE OF THE KOENIG-MARTENS INSTRUMENT

those little changes and improvements which are constantly being made in most successful plants, or, as we may say, the "physiology" of any gold mill.

Several months ago I spent a pleasant afternoon with a non-technical man who is trying to run a mill, and he entertained me with a story of his mill experiences and difficulties—and, by the way, such discussions as I have in mind would be largely connected with mill difficulties. My remarks on this plant are based almost entirely on his statements to me at the time as I recall them; and some of my inferences may not be justified.

This gentleman had but recently taken charge of the mill himself. Although a wide-awake and very intelligent and energetic man, he was without gold milling training or experience, and said he had no time to read either the text books or technical journals because the mill itself kept him so busy. He showed excellent ability in being able to secure ore to treat in a district where the competition among the mills for ore is very keen. That is, he was able to inspire confidence in the shippers of ore; which is especially essential where the mill is treating ore for the shipper, not buying it outright, but giving each producer just what amalgam and concentrates are obtained from his ore.

The plant had been operated at a loss. The new man attributed this loss mostly to theft of amalgam, and had consequently fired his superintending millman and taken charge himself. The business had been conducted on the basis of purchasing ore on sample to treat by amalgamation and concentration. On taking charge of the plant the new man ceased purchasing ore on sample, and started, according to the more usual custom of the district, to treat the ore for the account of the producer, this probably being a wise business move. A high tailings loss would not be so quickly felt in the mill balance sheet!

The day of my visit this operator was tearing out an excellent tube mill which had cost some \$3,000 in order to replace it with some type of grinding pan. Incidentally he had thrown out the classifiers which were in, and flooded the mill with others. He appeared to like to have things changing.

The mill had the usual crusher and grizzly equipment and some fifteen stamps. Very fine mesh screens and high discharge were used on the batteries. The coarse sand from the stamps was sent to the tube mill, which was, therefore, run on an extremely light feed, as there were only fifteen stamps to provide it. The tube mill was being rejected because the few stamps equipped for such fine crushing already could not keep it busy, and because of the large amount of power it consumed, this power being very expensive.

I believe that at this particular plant the equipment of tube mill in connection with stamps had not been thoroughly put through its paces before the change was decided upon to a grinding pan. The tube mill was probably being thrown out without a full trial of its possibilities of adaptation. It may be true that for this ore and treatment the two machines are not the best to work in series together, and that a grinding pan would be preferable to follow the stamps; but better commercial results can sometimes be secured by adapting a machine already installed than by throwing it on the dump heap to put in a new one at increased expense. This is true in spite of the great variety of machines made, each one better than any other in some small particular at least, and the fact that for a machine to be good, it must be good for the particular use in view.*

The old saw comes to my mind that "There are more ways than one to kill a cat besides choking it with cream." Some ways may be cheaper, and yet leave just as dead a cat. And in this mill it is probable that the tube mill could have been made to do well commercially by some adaptation of other equipment at slight cost to save and utilize the large capital

outlay in the tube mill. It seemed to me that this mill needed a business doctor instead of changes in its major equipment, a practical, technically educated mill man to put it through a period of experimentation with the machinery already installed, making any small changes necessary to see if a little ingenuity could adapt it commercially to the work to be done.

Now for the point of my remarks on this plant. A flow sheet of this mill with description of machinery taken before this change might get into a text book and start another man to copy it without due consideration of its work or what kind of ore it was best suited for—if any. By the time the description had gotten in print, the pattern would have materially changed. Or possibly the same might have occurred after the change, in which case a few remarks on the change would doubtless be inserted. It seems to me that the story of the change, including the reasons leading up to it why that particular change was adopted rather than another, and at least some results of the new arrangement, would make far more instructive and entertaining reading than a description of the mill either "before or after taking." True, the full history of this change is not yet made; no one knows yet but that the grinding pan may follow the tube mill to the waste heap.

A larger number of "physiological" chapters in our textbooks might contribute to lessen the apprenticeship necessary after school. Such chapters would give, for instance, a careful account of such a change as that mentioned including a recital of the symptoms of poor digestion, a statement of those methods of cure which prove defective, and many general details, etc. Students of this kind of thing can, if they will, profit by other men's experience and almost make it their own, so that a shorter term of apprenticeship than is now usually the case might fit them to take charge of affairs. I believe that we cannot do away with the necessity of experience for our young men. In fact, my observation of late has been to emphasize the need for our students—especially those in some Eastern schools—to undergo a real period of apprenticeship to round out their technical training.

I have in mind two other instances, in both cases successful mills, with which I was quite familiar at one time: I wish to cite these now, with close reference to each other, and this question.

One mill crushed wet with stamps, then amalgamated, classified roughly, reground the coarse sand in Huntingtons, re-amalgamated, concentrated, and cyanided the sand tailings, rejecting the slimes overflows which carried \$2 to \$2.50 usually—rarely up to \$4—per ton, and which amounted to some 35 tons or more per day. The work of this mill was excellent, the company was paying large dividends as the ore was high-grade, often \$30 or better per ton at that time. They were, however, sending down the creek in the slime overflows a large amount of money, of which, I am sure, a considerable portion could have been recovered with a fair profit. Of course, at the present time, scarcely any mill manager would think of wasting such material, easily amenable to cyanidation as it was and even at the time when I was familiar with the details slime handling machinery had been developed to a point where this material certainly could have been treated successfully. I mention this rather ordinary case to emphasize the next one which seems to me very striking.

This second mill crushed dry with rolls following a gyratory breaker, it being necessary to dry the ore as the mine was unusually wet. The drying acted also physically to render the ore somewhat porous. The finely ground dry ore was charged into vats, saturated from below with cyanide solution, and then leached for seven days, more or less, according to the ore supply and other circumstances. Strong sodium cyanide solutions—7 lb. KCN equivalent to the ton—were used because of the preponderance of silver. For a considerable period this ended the treatment. The tailings indicated more than usually satisfactory extraction, assaying only from one to two hundredths in gold.

But the idea was conceived of amalgamating after cyaniding, and after some trials this treatment was applied to all

*This is well illustrated in another mill I visited recently where Akins classifiers are used at one point where it is desired to secure a sand reject with especially low moisture content and Dorr classifiers are used where it is especially desired to get the sands free from slimes. These two machines are very similar, but have just about that difference in operation.

the tailings, the sands being sluiced from the leaching vats over amalgamating plates after leaching and washing. The expense of this treatment was slight, and every month a little gold brick was obtained from the plates sufficient to pay for the cost many times over, in spite of the low assay value of the material. The gold condition there was that most of it was very fine and easily dissolved, but that a little was coarse and rusty, being cleaned but only slightly dissolved by the seven days' treatment, and it was from this coarse gold that the added saving by amalgamation was made. This particular experience, by the way, has always seemed to me to indicate strongly the value of the microscope in ore dressing investigation.

The ore at this mine was of much lower grade than that at the former, seldom over \$8 to \$8.50 per ton in both gold and silver, and of course, closer economy had to be practiced here. The point of special interest to me is the development of a saving of some \$800 to \$1,000 per month at one plant from tailings assaying usually only 20 cents to 40 cents per ton, and the suggestiveness of this when thought of in connection with \$2 material in a fine state of division going to waste. Why not apply the experience at one plant to the condition at the other?

In the history of the treatment of Cripple Creek ores from the early unsuccessful attempts at amalgamation, through the earlier work of cyaniding, from that to chlorination, along with the development of the machinery for carrying on the process, the trials of chlorination in barrels and in open vats, the different methods of generating chlorine, etc., etc., the losing fight of chlorination against the better machinery of cyanidation, and the present development of cyaniding—in all this history of changes is ample opportunity for a number of papers which would illustrate not so much what the mills were at any particular time, but rather the daily operation and growth of milling processes, and would be of great value in teaching young men how to attack new problems. Some excellent papers of this kind, of course, already exist.

I may in closing mention very briefly some interesting recent developments of milling in the Cripple Creek district. For a long time certain mills there have been doing well commercially with what some persons might consider a very crude treatment. In those mills the ore is left very coarse, possibly being put through a 3- or 4-mesh screen, or even in some cases simply run through crusher and rolls and placed in vats for leaching without screening. This treatment is usually considered to be applicable only to the oxidized ores of the district, of which there seem to be very large supplies at certain of the mines. The success of the coarse crushing depends upon the location of the gold being in the seams of the rock, along which in crushing the rock breaks, thus exposing the gold minerals to the leaching solution without fine grinding.

The three latest operating mills present a very interesting development of this treatment—which is so much developed as to present almost a striking contrast. In all these mills fine grinding in tube mills is practiced on at least a part of the material. The details vary much in the three plants. In two cases of late practice the rock is broken down to about walnut size and then screened and only the fines saved, the coarse stuff being sometimes washed to recover any dust adhering. This treatment for the cost of crushing and screening secures from an ore running only about \$2 or lower a product which may carry \$7 to \$9 value per ton, which is then further treated by cyaniding. The further treatment at one plant includes grinding the above product in roller mills in a weak cyanide solution, and then concentrating on modified Wilfley tables. The tailings from these tables are dewatered, slightly washed and then discarded, as they are worth scarcely more than it would cost to treat them more closely. The concentrates are cleaned on a second set of tables and shipped to the smelter; and the middlings from this set of tables is reground to slimes and cyanided by agitation. The pregnant solutions are precipitated by zinc dust. The slime overflow from the sands of the first Wilfleys is also cyanided by agitation. This treatment

is seen to be really a logical development of the former in that it is still the fine breaking material which receives most attention; there having been added from another source the item of concentration on shaking tables.

The student can get from textbooks descriptions of many plants and thus familiarize himself with different types of mill adapted to different ore. No one can learn milling from textbook study or classroom lecture alone; but could we not further assist the student by relating to him more of "mill physiology," tell him of the "life processes" as it were of the mill in action, and thus prepare him better to take up the direction of such work? I think so.

University of Pittsburgh, Pittsburgh, Pa.

Determination of Oxygen in Copper and Brass

A paper by Mr. T. West on this subject was presented before the recent meeting of the (British) Institute of Metals.

The author has worked out a method for the estimation of oxygen in copper and brass, using carbon monoxide as the reducing agent. In the first instance the conditions for estimating oxygen in copper were determined by comparative tests with the customary method of reducing with hydrogen. A temperature of about 900° C. was found to be sufficient, the time varying between two and three hours.

Since any oxygen in brass probably exists in the form of zinc oxide a number of experiments were next carried out under varied conditions to ascertain how far the reduction of zinc oxide by carbon monoxide, and weighing the resulting carbon dioxide, could be relied upon as a test for zinc oxide. It was found that provided the temperature reached 1050° and the gas current was sufficiently rapid the amount of carbon dioxide formed corresponded closely to the actual weight of zinc oxide taken. Interesting variations in the method of condensation of the zinc were obtained. Experiments were then carried out on mixtures of copper (containing known amounts of oxygen) and zinc and these gave satisfactory results. Finally tests were made on samples of pure brass and these showed that ordinary brasses of good quality contain on an average about 0.002 to 0.003 per cent. of oxygen. Tests were also carried out on condenser tube brass containing about 2 per cent. of tin, and also on German silver. In these cases a rather surprising result was obtained, since a deposit of carbon was found adhering to the metallic surface after cooling down in an atmosphere of carbon monoxide. The method, therefore, can only be applied to the estimation of oxygen in brasses containing copper and zinc (with the ordinary amounts of commercial impurities) and not to spiral brasses containing tin or nickel, on account of their decomposing action on carbon monoxide.

Annealing of Gold

A paper on this subject was presented by Dr. T. K. Rose before the recent meeting of the Institute of Metals.

In the first part of the paper, the effect is shown of the presence of small quantities of impurities on the temperature at which gold can be annealed. This is the first investigation of the kind, and although the exact quantitative effects in the case of metals other than gold are doubtless different from those recorded in the paper, indications are afforded of the general results which may be expected. The effects are remarkably great. For example, the addition of only 0.002 per cent. of hydrogen raises the temperature of annealing of gold from 150° to over 300°, and 0.05 per cent. of copper raises it to 250°. Further additions do not raise the temperature proportionately.

It is further shown that the annealing of rolled gold begins by the complete softening and recrystallization of some of the laminated crystals of which the metal consists, the other laminae remaining unaltered until more time has elapsed or the temperature has been raised. Accordingly partially annealed metal consists of alternate strips of hard and soft material, having weak places where its rigidity and resistance to rupture are little higher than those of fully annealed material.

The paper is illustrated by curves and photomicrographs.

Induction-Furnace Notes

By Joh. Hårdén

In the following a few notes of details and readings on an induction-furnace plant, installed some short time ago, will be given, as an example of what can be done in this class of furnaces when used for high-class crucible steel making. The author ventures to surmise that the notes will be of some interest to the practical steel maker, since the facts given are taken from actual practice with a furnace plant which has now been in operation for some time and all uncertainties of the commencement are overcome.

Early in 1909, a large crucible steel making firm decided definitely to install a Kjellin induction furnace as a supplementary to their pot smelting. This decision was as much more recommendable as the raw materials consisted chiefly of Swedish bar ends, pig iron and high-grade steel scrap; consequently a most effective "melting machine" only was desired, refining from phosphorus and sulphur being unnecessary. Great importance, however, was laid on the question of elimination of gases and slag incrustations, as well as a high thermal efficiency, so as to keep the melting costs low.

The furnace was consequently designed accordingly; the writer was entrusted with the pre-calculation and general design, while a large engineering firm was called in for the construction of the tilting gear, staging and other auxiliary mechanical details.

The furnace was originally ordered for a tapping capacity of 1 ton; being fully acquainted, however, with the prevailing conditions, the writer thought it most advisable to provide for a considerable margin. This precaution proved fully justified, because the charge has subsequently been gradually increased up to over 2 tons.

Seeing that no particular refining was required, the bath section was kept rather deep and narrow, giving a smaller slag surface. In this way the thermal conditions could be kept better under control, and a small remainder only of the charge would be required for recharging.

The pre-calculation showed that the most economical dimensions for the hearth would be:

Middle diameter	66 in.
Height	11 in.
Width	63 in.

These dimensions were adhered to in the construction, with the exception that provision was made for deepening the bath so as to allow fully 2 tons of steel.

The above dimensions would give a capacity of 3960 lb., permitting of a tapping of 2700 lb.

The electrical data as calculated for this quantity proved to be very accurate when tested under running conditions; they were as follows:

Resistance of the bath = 18×10^{-4} ohms.

Inductance = 2324×10^{-9} Henrys.

Secondary current = 37,900 amp.

Power factor at $15\frac{1}{2}$ per cent 0.633.

Magnetic saturation, 8550 lines of force per cubic millimeter square.

As will be seen from the readings given below, the lowest power factor is somewhat under the calculated figure; this is due, in the first place, to the fact that the bath, as already said, was increased in bulk, while the other factors remained unaltered. Secondly, for certain mechanical reasons, the magnet core had to be suspended and stiffened by means of heavy tie rods and brackets which formed a magnetic by-pass which to some degree influenced the leak field. When the magnetic core and coil were tested on the bed, before the above-mentioned alterations were made, the readings gave an even higher power factor, namely, corresponding to 0.66 at full load.

The current supply was rather unfavorable, inasmuch as the current from an old 500-550 volts direct current machine was converted, by means of two slip rings, into single phase. This generator, which was of a larger size than actually required for the furnace, was also performing other duties at the same

time, which, of course, caused certain disadvantages as to regulation.

The normal single-phase voltage was 356, which could be raised at intervals to 400 by means of altering the excitation. The total maximum power available for the furnace was given as 800 kilovolt amperes.

Originally an air-cooling arrangement was provided, the air was supplied by two fans, large enough to allow one to be disconnected for cleaning for a couple of hours. The air pressure was 5 in. water gauge, delivered through 6-in. piping and cloth filter.

Later it was found that it was very difficult to obtain sufficient pure air, free from metal dust, in spite of the filters. It was, therefore, decided to substitute water cooling, as is the usual practice in most Kjellin furnaces, leaving the air cooling as a standby. This proved very satisfactory, there being no danger whatever, if a properly designed cooling jacket is provided, since any liquid metal coming into contact with it will instantly freeze and form a secure wall. Besides, a cooling mantle of this type is a very good indicator to the furnace man as to the conditions of his lining; long before the lining is worn out thin feelers of liquid metal will slowly penetrate the thinnest part of the lining, and gradually increase the temperature of the cooling water; by careful watching of the temperature by means of a long thermometer it is quite possible to foretell exactly the state of the lining and to disrupt the operations in good time, before matters have gone too far.

The quantity of air required for the cooling of a furnace of this size is about 250 cubic meters air per square meter of the cooler-surface and this surface ought to be about 1.14 decimeter square for each kilowatt of the furnace capacity. For water cooling, 3 to 5 gallons per minute, depending on the temperature, is quite sufficient.

The furnace was suspended on two trunnions, with a tapping spout near the edge. The tilting is effected by means of a large screw spindle operated by means of an electric motor and drum controller. This proved very satisfactory, especially as the arrangement was so made as to permit back tilting as well, which was especially advantageous when starting up, and for patching up of the lining. The charging platform was arranged practically in level with the upper edge of the furnace, for the sake of easy charging of cold scrap. The portion of the platform extending over the spout was arranged to swing on hinges when tilting. Cut-out switches were provided so as to prevent over-tilting in either direction.

The instruments and switchboard are arranged direct on the charging platform; they consist of voltmeter and ammeter, kilowattmeter, recording kilowatt-hour meter, main switch and field regulator, together with the fittings for light and auxiliary power (tilting, fans, etc.), the whole mounted on a compact casing-pillar of steel and expanded metal.

At the commencement, the lining as used by Dr. Kjellin in his first furnaces was put in.

This consisted of:

1100 kg.	of coarse, calcined magnesite (Styrian)
22 "	fine caustic magnesite.
90 "	white Holland clay soaked in
55 "	warm water.
11 "	boric acid.
33 "	boiling water.

The parts to be added in succession as given, and kneaded well together.

This is tamped in with hot rammers in one operation, care being taken not to interrupt the ramming more than absolutely necessary before it is finished.

This lining gives a rather good result, but is somewhat expensive, and if not handled with great skill and care, is most likely to develop vertical as well as horizontal cracks, which of course is rather dangerous. The first two linings of this kind in a new furnace are often at fault, as the men are not used to handling it.

Subsequently an ordinary tar-magnesite lining was employed; the mass consists of two parts of coarse, burnt magnesite, about $\frac{1}{2}$ in. to $\frac{5}{8}$ in. grain, one and one-half part medium, or $\frac{3}{8}$ in. to $\frac{1}{4}$ in. grain and $\frac{3}{4}$ part fine, or 12 to 16 mesh grain. Each kind is rabbled separately with 8 to 9 per cent of hot, water-free coke-tar on a heated plate, care being taken that each grain is enveloped in a film of tar. The parts as mentioned are then mixed on the hot tray, in small quantities as it is used up, and rammed in hot. Too much material must not be mixed at one time, since the tar oxidizes and a good binding is not secured.

As soon as the ramming-in of the bottom is completed, which is the easiest part, a second squad of men is employed, in order to insure a continuous working. The work is usually conducted in this way: One man is constantly preparing the mass and carrying it to the furnace, the foreman throws it by means of a scoop in even layers in front of the rammers, four to five men walk in the casing around the mold, using heated hammer heads on pneumatic hammers. Care is taken that both sides of the mold are equally tamped.

The ramming having proceeded up to the top of the mold, a thinner layer of fluxing material, consisting of fire clay, magnesite, and iron scale is used as a finish. The ramming of a 2 to 5 tons furnace should be completed in 8 hours.

The ring-shaped mold of cast iron or wood is made in three sections, screwed together; one of the sections is smaller than the other two and wedge-shaped at the ends, so as to permit an easy removal. Lifting eyes are screwed in holes in the top and the mold removed, section after section, commencing with the smallest.

After removal of the mold, it is well to face the annular hearth with a facing of ganister, Thomas slag, scale and water, by means of a stiff brush. Several rings of cast iron or welded wrought iron, about $2\frac{1}{2}$ in. high by $\frac{3}{4}$ in. thick, are placed in the furnace, the upper limb of the magnet is hotted down and current turned on. At first, only about 15 to 20 per cent of full current is used; as the fuming of the mass diminishes, more power is given until the rings gradually become red hot and show signs of melting.

If only one ring, having the full depth of the bath, is employed, it is likely to melt off on one spot, thus interrupting the circuit; this is avoided by using several lower rings which never break on one place. The bottom ring always melts first. As soon as this is observed, chips and borings of cast iron are filled in on the hottest part, thereby cooling this and procuring a bath proper. Soon the whole bottom is covered with molten iron, and punchings, scrap, etc., is added until the bath is full; a blanket of lime with some Thomas slag is formed and full power is turned on. The superheating is continued for about $1\frac{1}{2}$ hours, so as to harden the lining. Meantime ring-shaped grooves are formed in open sand boxes and the first charge is used for casting fresh rings for the subsequent restarting. In this way, practically no material is wasted and besides, it is always wise to have a few starting rings in stock, in case one should happen to drain off too much of the charge at any time.

This first charge should always fill up the whole furnace to its full capacity; a curious fact as to the power factor may be observed during its formation. Suppose a periodicity of 16 cycles is used in a 2 tons furnace; at first, with cold rings the cosine is about 0.70 which gradually rises to say 0.75, then decreases to 0.55 just before the rings begin to melt, when it again rises to about 0.70. As the filling of the bath proceeds it sinks to about 0.63 or even, at full bath, to 0.58-0.60.

This is explained in the following way: The cold rings have a fairly high ohmic resistance, which is gradually increased as the rings are heated; this tends to give a higher power factor. As, however, the permeability also increases rapidly with the temperature from a certain point, up to about 740 deg. to 780 deg. C., or the first recalescent point, the magnetic leak field is increased more rapidly and this again causes a lower power factor. This, however, ceases at the

above-mentioned point, which is shown on a sudden jump in the power factor curve, though it does not rise as high as from the start, as in the meantime more charge has been added and the metal is also well fritted together, thus somewhat lowering the total resistance. As the hearth is fully charged to its utmost capacity, the leak field is, of course, again increased, due to the counter inductive force of the secondary circuit.

It may be mentioned, by the way, that the average power factor for a given charge is a good indicator of the state of the lining. This fact is always appreciated by the experienced melter. It may be explained in the following way: Supposing the power factor is noted to be, say, 0.68 with a charge of 2 tons of about 0.5 to 0.7 per cent carbon steel during the first two or three melts; after say 150 or 200 charges it will be noted that this figure is slowly but steadily going down (sometimes earlier, at times later) and no attempt seems successful in restoring the original conditions. When it has reached say about 0.59 or 0.60, other conditions being equal, it is wise to stop and break up the lining as deep as the hearth proper, otherwise one may be fairly sure of a breaking through of the liquid steel somewhere.

Upon dismantling the lining, it will be found that a great number of fine threads of steel have slowly but surely penetrated the inner wall of the lining, until they stop and freeze against the cooling jacket. However hard the lining may be rammed in, small pores are always formed through the evaporation of the tar or binder, and the highly liquid metal is sure to find its way in. As this goes on, a sort of nest of solidified metal is formed through the inner wall around the cooler, and as this forms an excellent return for the lines of force of the leak field, this is increased and thereby the power factor lowered.

It is also necessary to take the temperature of the cooling water in the jacket once or twice during each charge, especially if the lining is "old." It should not be higher than 40 deg. to 50 deg. C. If it is found to rise higher of a sudden, it is a true sign that a larger crack has developed and a stem of steel has penetrated into the cooler.

If these precautions are always taken (which a skilled melter will do as a matter of instinct) no fear of breaking through need arise. In fact, it is to be regarded as either gross carelessness or inevitable accident, which need but very seldom occur, if a breaking through in a modern furnace is happening. The power factor meter and the thermometer, applied immediately in the outlet of the cooler, should prevent all such unwanted occurrences.

The thermometer should preferably consist of a thermocouple, connected to a millivoltmeter next to the power factor meter on the board. A mercury-thermometer inset direct in the cooler should be used as a check.

At times the millivoltmeter, calibrated in degrees C, is built as a contact instrument, ringing a bell when the temperature is too high, thus warning the melter of the danger. This was carried out at Roechlings works in Germany, but it seems to complicate the arrangement unnecessarily. Besides, the minute force acting upon the instrument does not always give an absolutely reliable contact, and it only serves to throw the melter off his guard. For this reason the arrangement is not recommended by the author.

A good lining, well built in, according to previous descriptions and carefully handled, should stand about 450 charges (463 has been reached in one furnace) when charged with liquid material; with cold charging, the life is naturally somewhat lower, on account of the mechanical strain caused by the throwing in of the larger pieces, but at least 150 to 200 should be made. The total cost of the lining, per ton of steel treated, is so small, that it does not count much when it is compared with the cost of pots and furnaces in ordinary pot-melting, especially with coke.

In former practice, one of the principal causes of ruining the lining, especially with stationary furnaces, was the treatment, or rather ill-treatment, of the taphole. Modern furnaces

are mostly made to tilt, but even here it is beneficial to the steel to make use of a "covered" taphole; considering that one has taken particular care to purify the steel from all obnoxious ingredients, among which the occluded gases are by no means the least, it is obvious that a long, open spout for tapping, in which the steel is exposed to the air, running as it is in a thin stream, thereby inevitably dragging smaller or larger lumps of semi-solid slag and crusts down into the ladle or runner, is evidently wrong.

Therefore, it is better to provide a bridge or barrier, so that the taphole is just below the normal steel level, whereby the furnace is greatly tilted after, so that the slag may run out last. In a stationary furnace, of course, the taphole must be of such level as to directly drain the hearth, leaving only about two-fifths of the charge behind for cold charging, or complete teeming when liquid charging is used.

Now, in the earlier days, a pointed steel bar was simply driven in, similar to the cupola practice, which is easy enough when the taphole is kept always "free." But, often enough, mixed slag—and steel—crusts froze in the slag hole, and considerable violence was exercised in breaking in through, which usually sooner or later led to a destruction of the lining round the taphole.

The author, therefore, adopted the practice known from carbide manufacture, to use a "live" bar, i.e., a pointed steel bar is connected to one transformer terminal, having about 65 to 70 volts, while the other is suitably connected to the metal bath, a choking coil of a few turns round an iron core, or a few turns round the back member of the furnace core, acting as an auxiliary transformer, in case only high tension is available, is equally useful. Obviously the other terminal is applied to the bath by means of a steel bar. Only about 100 to 200 amp. at 40 to 60 volts, applied for a few seconds, are required to break through the most stubborn taphole with greatest ease, if a slight hammer stroke is applied at first. It is surprising to see how clean a tap can be obtained in this way without the slightest injury to the lining; the bar is not worn away to any great extent, if it is quickly withdrawn as soon as the metal begins to flow. A further advantage is that the teeming can take place in the right and desired moment, while with the old style one had to play about the taphole at times for 15 minutes or more. It may be added that this method was not, to the author's knowledge, applied to the very furnace reported on here; but it has been used by the author in other furnaces with excellent results.

Recurring to the furnace in question, the following figures from a melting campaign at random may give an idea of the actual working:

The run covered 606 hours (including drying of the lining) with 120 charges, which gave 166 tons of finished steel of highest quality, mostly tool steel, but also steel of lower carbon content. The average power consumption over the whole run, including drying and starting, was 636 units per ton finished and degasified steel.

Generally 2 tons were treated in the furnace, teeming 1½ tons, leaving ½ ton for retesting. Two typical runs are given here, lasting 4½ hours each. No. 1 gave a tapping of 30 cwt. and No. 2 gave 28½ cwt.:

Hour	Volts	Amperes	KW.	Cos φ	No.
12 NOON.....	340	440	125	0.835	1
1.....	344	790	195	0.716	
2.....	336	950	208	0.650	
3.....	334	1060	210	0.590	
4.....	334	1105	212	0.573	
4:30 P.M.	Tapping 30 cwt.				
4:35 P.M.	320	340	85	0.79	
9 A.M.....	340	560	156	0.817	2
10.....	345	820	205	0.720	
11.....	330	955	210	0.670	
12.....	338	1090	228	0.621	
1 P.M.....	348	1175	250	0.610	
1-25.....	Tapping 28½ cwt.				

To the above figures may be added: In run No. 2 a slight mishap occurred, inasmuch as a certain quantity of metal was

splashed into the cooling jacket through the charging of a not quite dry lump of pig iron, hence the somewhat higher amperere readings in this case. This also somewhat lowered the power factor; no remark is made about the carbon content of the steel, but it is evident from the readings that the charge in No. 1 was of a low, and that in No. 2 of a higher carbon percentage.

It should also be noted that all the instruments were subsequently found to read somewhat too high, but as a full set of calibrating instruments could not be obtained at the time, it was thought advisable to leave the readings as obtained; the actual error cannot be very great, since all instruments showed a plus-reading of not a great magnitude.

The theoretical efficiency of the furnace, taking the average power consumption for one campaign as above, or 666 units per ton, inclusive of heating up, stoppages, etc., works out as follows: The melting, superheating and "killing" of 1 ton of steel requires in round figures 350,000 calories; 1 kilowatt-hour being equal in round figures to 864 calories. We thus find an efficiency of

$$\frac{350,000}{864 \times 636} = 63.8 \text{ per cent.}$$

This is to be regarded as exceptionally good; taking a greater number of runs, and a greater variety of steel, we get an average power consumption of 720 kilowatthours.

But, as pointed out in previous reports, it would be fallacious to judge as to the merits of the process by the theoretical efficiency only; this can only be regarded as an indicator of the furnace conditions. Apart from the quality of the product, which is stated to be of very high grade, the economical results must be carefully considered.

Assuming a power cost of ½ of a penny (one cent) per kilowatthour, a figure which in many places in England is on the high side (as low as 0.35 d. or 0.7 cent has been offered, to the author's knowledge, for a load factor of over 80 per cent), we find the approximate costs to be as follows: On a furnace as the one described, giving 1½ tons every 4½ hours, or 7.95, say around 7 tons in 24 hours.

	Per Ton
Power 720 Units @ ½d.....	£1 10.0
Lining: (retiring after first run):.....	
2 tons magnesite @ £4.10.....	£ 9. 0 0
1 ton dolomite @ £1.10.....	1.10 0
Tar.....	10 0
Bricks.....	1.10 0
Labour.....	3. 5 0
	£15.15 0
to last for say 175 tons.....	1.10
Labor:	
2 melters @ 8/- per day (2 shifts), 4 assistants @ 5/- per day, 1 labourer @ 4/- per day, or, per 24 hours equals 40/- and 7 tons, say.....	5 8
Extra labour occasionally.....	1.0
Management.....	3.6
Sundry expenses, daily repairs, etc.....	3.0
Depreciation.....	2.0
Losses 14½, say.....	4.0
Total.....	2.11.0 *
*Estimated melting cost per ton of steel.	

If we add to this a margin of, say 10/- per ton, to cover any accidents on bad linings, general mishaps, stoppage of power, license, etc., a figure which, it must be admitted, is rather generous, we still find that the actual melting does not materially exceed the figure of £3.00 per ton of steel, ready to sell.

It is generally admitted in Sheffield that the average cost of crucible melting is well up to double the above cost and generally more, taking everything into consideration.

The above figures are, of course, based on crucible steel melting only; although it may be quite possible to refine inferior raw material into high quality steel in induction furnaces of the Roehling-Rodenhauser type, as daily practice has shown, it is not intended to recommend such refining in a simple Kjellin furnace, on account of its hearth shape, which leaves only a limited area of contact between slag and steel, and also on account of the difficulty in keeping the slag blanket sufficiently hot. The furnace described was ordered as a melting machine only, to take the place of a crucible plant, capable

of executing all that is required of the latter, and experience has proved that these conditions were fulfilled, at a considerable reduction of cost, giving a steel fully up to the quality of the best crucible steel.

Luton, England.

Improvements in Smelting at the Consolidated Company's Works, Trail, B. C.

By E. Jacobs

The Consolidated Mining & Smelting Company of Canada, Ltd., continues to make improvements and additions to plant at its lead and copper-smelting works at Trail, B. C. That it is smelting a larger quantity of ore this year than last is evident from the following comparison, which shows also a generally higher value of ore smelted. During five months ended May 31, 1913, the quantity of ore and concentrates smelted was 134,660 tons, the gross value of which was \$3,526,436. This compares with 206,458 tons of material smelted during the company's last fiscal year, and a gross value of \$5,083,078. The monthly average of ore smelted this year was 26,932 tons, gross value \$705,287, as against 24,705 tons, gross value \$423,590, for last year.

Lead Smelting Department

In the lead sampling mill provision has been made for finer crushing so as to obtain a better product for roasting. Formerly the crushing was done by a Gates gyratory, one set of 42 in. by 16 in. heavy-duty Traylor rolls, and two sets of lighter rolls of the same make. Now the lighter rolls have been replaced with two sets of the heavy-duty type. Conveyor belts are being placed under all lead-ore beds. These will feed directly to the hoppers of the roasters and thus do away with the manual labor required for tramming from the beds to the roasters.

Part of the old building housing the Huntington-Heberlein sintering plant has been taken down, leaving but 175 ft. of the old structure standing. In place of the part removed, a wooden building has been erected, 61 ft. by 234 ft. The roof trusses are wood with iron tie-rods, making an excellent roof that withstands wind and rain, and carries the snows of winter. A similar roof on the company's lead refinery building has been serviceable and lasting. An electrically-operated Niles 20-ton crane is used in this building, the rails being spaced 58 ft. centers.

Seven Huntington-Heberlein roasters are in use, and preparations are well forward for putting in two Wedge roasters, early receipt of which is expected. The fire-brick to be used in their construction are now on the ground, and it is intended to have the furnaces in operation late in the ensuing autumn. In order to provide space for the roasters it was necessary to remove a large gravel bank, which was done by hydraulicking.

There has been a rearrangement of the sintering pots, of which there are 36 in the building. These have been placed in four rows of nine each, and in convenient proximity there has been erected a large concrete bin to receive the ore which has received a preliminary roast. A steel conveyor handles the roasted ore between the roasters and the bin. The sintering pots are filled from the latter and then lifted by the crane which places them on the converter stands. After the ore is properly sintered and cooled, the crane again removes the pot and dumps the contents on a floor, thus breaking the large cakes. A Hayward clam-shell bucket, the largest size made, lifts the broken sinter and dumps it into the hopper of a 24 in. by 36 in. Farrel crusher, from which it is removed to bins by a steel conveyor. At the upper end of the conveyor the sinter is passed over coarse and fine grizzlies which eliminate the fine portion and deliver a product suitable for the lead blast-furnaces. The fine part is returned to the bin and again sintered with a fresh charge of roasted ore.

A gas-producer is being installed to provide gas fuel for all the roasters, and for such other purposes for which it may be found advantageous to use producer-gas. A new building

has been erected adjacent the Huntington-Heberlein building to house the small Roots blower and a centrifugal fan giving 16 oz. pressure. In the upper story of this building are an office for the shift boss, and change and lunch rooms for the men, so that they may be away from the lead dust when eating and resting.

Three new lead blast-furnaces are being constructed; dimensions at the tuyeres, 45 in. by 216 in. They have wrought-iron jackets with 14 tuyeres; also cast-iron jackets with single tuyeres, as at present in use on the old furnaces. The height from center of tuyeres to feed-floor level is 17 ft. 6 in. These new furnaces will have their feed floor on the same level as that of the copper furnaces, instead of the higher level found convenient for hand feeding. With mechanical arrangements, one level for the feed-floor of all furnaces, both copper and lead, admits of more economical handling of furnace charges. To provide room for the new furnaces, the main building has been extended eastward by the addition of another bay of about 40 ft. with a lean-to to the south. The lowering of the tapping floor of the lead furnaces 10 ft. below the former level necessitated the removal of a lot of gravel, which was done by hydraulicking it out to the dump. Meanwhile the feed tracks of the two old furnaces, which are to be taken out, and the columns of buildings, etc., had to be supported while the changes were being made and until concrete piers and l-beams were put in place to form a new foundation and sub-structure.

Copper Smelting Department

In the copper smelting department, the improvements in hand include a conveyor to transport ore from the copper sampling mill to a stock-pile when there is more ore coming from the mill than can be held in the furnace bins. A new storage bin has been built, with sloping sides; and underneath is a tunnel into which cars are run for the ready withdrawal of ore from the bin. All stock-pile floors and bins now have tunnels under them, so that the removal of material is greatly facilitated and shoveling is entirely abandoned.

In place of the 4-ton motors formerly used, there are now two 7-ton electric locomotives for hauling to the charge bins, and larger cars are being made. It is the intention that these improvements shall permit hauling to the charge bins being done on one shift. One of the 7-ton locomotives is used also for hauling cars of lead anodes from the tapping floor to the refinery, instead of by railway locomotive as before.

The copper smelting plant has for some time included five blast-furnaces. Of these No. 2 has been replaced by a 42 in. by 35 ft. furnace with 28 standard tuyeres on each side. This furnace has an arched top and flat flue, the latter arrangement permitting the elimination of the goosenecks and the use of an overhead traveling crane on both furnace floors. It is expected that the new furnace will be found to possess such advantages that two similar furnaces, each of 450 tons daily capacity, will be substituted for three smaller furnaces, thus providing for the daily treatment of 1350 tons of ore, excluding fluxes, in three blast-furnaces.

On the new No. 2 furnace the feed-water pipes have been so arranged that all valves can be reached from the tapping floor, and discharge pipes so placed that the furnace-man can at all times see that waste water is coming from the jackets. The overflow trough has been made larger and placed farther out, so that no water will spill on the furnace-man when engaged in punching tuyeres.

The new furnace has center feed, the charge train being pushed into it by the electric motor. At present the cars run into the furnace on water-cooled rails on the track level; but after the tunnels under the charge bins shall have been enlarged wheels will be placed on the upper part of the charge cars and the latter will then run into the furnaces on rails placed at the proper height.

Crushing and granulating of copper matte is no longer done here as formerly, and the old plant used for that purpose has been taken out. This gives more room at the west end of the

furnace floor and permits a rearrangement of the tracks about No. 4 furnace. All low-grade copper matte is put through this furnace with silicious ore to raise the copper content and give a matte of 35 to 40 per cent copper. This product is shipped to works at Tacoma, Wash., for converting.

No copper-ore roasting is now done at Trail in the Huntington-Heberlein pots. Copper concentrates only, chiefly from the Rossland mines, are sintered on the Dwight & Lloyd machines. The gates of the latter have been changed from the herring-bone type previously used to straight-slot self-cleaning grates.

Blast is delivered to the furnaces at 32 to 34 oz. pressure. Six blowers have been in use, four Roots and two Connorsville. Another No. 11 Roots is being added to give 401 cu. ft. per revolution. It will be driven by two 300-hp induction motors.

The pyritic smelting of raw copper matte commenced in the early part of 1912 has been discontinued at Trail. While this practice was followed the charge consisted of 4200 lb. matte, 2000 lb. silicious ore, 13.5 per cent lime rock and 4 per cent Crow's Nest coke.

The ore now treated in the copper smelting department is obtained chiefly from the company's mines at Rossland. An approximate analysis of the ore shows: iron, 17 per cent; silica, 44 per cent; lime, 5.5 per cent; alumina, 15.5 per cent; sulphur, 8 per cent; magnesia, 3.5 per cent. This ore is smelted with 30 per cent lime rock and 16 per cent Crow's Nest coke.

Colorado School of Mines

The institution opens September 2 for its thirty-ninth annual session. The Summer school, held during July and August, had an attendance in excess of last year. This session offers the best opportunity for conditioned students to remove all obstacles to complete standing; and applicants for advanced standing, who are rarely prepared in every study, may also use the Summer school for completion of their requirements.

The expected registration at the School of Mines proper is 240, chances for a greater number depending on unforeseen admission to advanced standing. The graduation of seventy young men last May illustrates to what an extent in recent years these admissions have served to swell the numbers in the upper classes.

The working schedules of the institution have been largely revised during the Summer. The changes are all in the direction of greater simplicity in the courses. This has been accomplished without any material sacrifice of topics, nothing essential having been omitted in the new arrangement.

Mr. Harry J. Wolf, who acted as professor of mining during a part of last year, has accepted the position permanently for the coming year. The headships of all other departments remain as before. The total number in faculty, including associates and assistants, is twenty-four.

Ancient Egyptian Metal Objects

In a paper read at the recent meeting of the (British) Institute of Metals in Ghent, Belgium, entitled "Metallographical Researches on Egyptian Metal Antiquities," Mr. H. Garland describes his investigations into the crystal structure of Egyptian metal objects, all more than 2000 years old, made either of bronze or copper.

He gives the interesting case of a bronze knife 3500 years old in which the original cored structure has endured until the present day and is still perfectly stable at atmospheric temperatures. The view is held that the knife was hammered cold and so the best cutting edge was obtained. The crystal grains are very small, there having been apparently no crystal growth during the ages that have passed since the metal was first worked. This shows either that recrystallization does not take place at ordinary temperatures or if it does, then many times 3500 years is required to effect the change.

Mr. Garland, in his paper, gives other cases of the persistence of the "cast cored" structure in antique alloys, illustrating his views by means of photomicrographs of a small

bronze chisel, such as would be used by a jeweler in ancient Egypt, a copper arrow tip and of a bronze spatula.

An interesting photograph reproduced in the paper is of a Roman coin which still shows cores, though, as has been stated by Dr. Rose, of the Royal Mint, such coins were struck hot. Mr. Garland holds that the heat could only have been slight otherwise the cored structure would not persist.

Electric Furnaces—Their Design, Characteristics and Commercial Applications

By Woolsey McA. Johnson and George N. Sieger

Specific Design and Important Details

Electrodes.—To the electric furnace there come two conductors from the source of power; these lead to the electrodes (electrodes is a word derived from the Greek at the instance of Faraday from the words *electra*, electricity and *hodos*, path) and by these electrodes the electric current is delivered to the laboratory or working space of the furnace.

The electrodes must conduct electricity fairly well but should not conduct heat very well. Such a material does not exist, for every conductor of electricity is a conductor of heat, and quantitatively the two properties vary together and hence must be considered together in design. The two qualities, one good and one bad, must be made the subject of mutual adjustment.

By designing the electrodes of proper dimensions, it is possible to have an electrode that will conduct electricity with a minimum of loss and will conduct out a minimum of heat. This condition is not obtained by the ordinary current density design for in this design larger electrodes will run hotter than small ones owing to the fact that in the case of the larger electrodes the interior area has no chance to radiate its heat whereas in the smaller electrodes this is not the case. A much better principle of electrode design than that of current density is to keep the radiation surface constant per unit of current. The condition for minimum electrode loss is that when the temperature of the electrode is the temperature of the interior of the furnace the electrode will abstract no heat from the furnace.¹

Electric furnace electrodes can be made of the following materials:

- (1) Acheson graphite.
- (2) Molded carbon.
- (3) Metals such as copper or steel.

Practical Information about Electrodes

	Electrical Resistivity	Thermal Resistivity	Safe Carrying Capacity in amp. per sq. in.
	c. g. s.	c. g. s.	
Acheson graphite	0.000813	6.18	125
National Carbon Co.'s molded carbon.....	0.0038	61.8	38
Steel or iron.....	0.00009952	6.25	170
Copper	0.0000105	1.22	1000

Safe carrying capacity in amperes per square inch:

	Acheson Graphite	National Carbon
5" round	2350	720
6" "	3500	1000
8" "	6300	1900
10" "	9800	2000
12" "	14,100	4300
14" "	19,300	5850
16" "	25,000	7625
2" square	600	150
4" "	2000	600
6" "	4500	1350
8" "	8000	2430
10" "	12,500	3800
12" "	18,000	5450
14" "	24,500	7450

¹Carl Hering, Transactions Am. El. Chem. Soc., vol. 16, p. 265, 1909; E. F. Roeder, Transactions, vol. 16, p. 363, 1909.

	Voltage drop per foot at safe carrying capacity		RI ² loss per foot length at safe carrying capacity (KW)	
	Acheson	National Carbon	Acheson	National Carbon
5" round	0.49	0.68	1.14	0.48
6" "	0.47	0.64	1.64	0.64
8" "	0.44	0.665	2.77	1.26
10" "	0.48	0.636	4.70	1.84
12" "	0.42	0.645	5.90	2.77
2" square	0.58	0.68	0.35	0.10
4" "	0.48	0.60	0.96	0.36
6" "	0.45	0.68	2.02	0.91
8" "	0.48	0.68	3.84	1.65
10" "	0.51	0.68	6.35	2.37
12" "	0.36	0.70	6.48	3.85

Steel or copper electrodes are rarely used save for the bottom electrode of a furnace having a bath of metal. They must be water-cooled either by a spray or by a waterjacket. Usually it is well to design them at safe current-carrying capacity for cable or bus-bar, that of 1000 amperes per square inch of copper and 170 amperes per square inch of iron or steel. But 3000 amperes per square inch of copper and 500 amperes per square inch of iron and steel is permissible provided the electrode is well water-cooled.

Carbon electrodes of small sizes are made out of crushed petroleum coke or of anthracite slack in larger sizes mixed with one or several kinds of tar as a binder. Probably the best tar is gas-house tar containing carbon in a form similar to lamp black, in suspension.

A hot mixture of this is poured, molded, or extruded into the desired form. The green electrodes are slowly heated progressively, finally reaching a white heat. They are covered in the open-hearth furnace in which they are heated with sand to prevent oxidation. The temperature must be lowered slowly. Indeed, it is the practice at a certain German works to keep up the heat allowing the temperature to retrogress slowly.

As considerable gas is evolved in the process through the pores of the electrode and as the resultant product must possess a firm, hard structure, it is easy to see that this heating and cooling must be slow and uniform.

For an electrode of large cross-section such as is used in a Cowles type of furnace, a wooden form can be filled with hot "electrode concrete-mixture," made by mixing broken irregularly sized pieces of electrodes, fine dust and tar and tamping same hot into place. A long needle is run through so as to make holes spaced every 1 in. or 2 in. in order to allow the gases to escape easily. Then a wood fire is built and when the mass is dried out, the final cementizing carbonization is effected by passing an electric current through the mass.

GRAPHITE VERSUS AMORPHOUS CARBON ELECTRODES.

Graphite electrodes are made by the Acheson process by subjecting molded amorphous carbon electrodes in an electric furnace to a heat sufficient to graphitize the carbon, say, from 2300 deg. C. to 2500 deg. C.

The commercial comparison of Acheson graphite and amorphous carbon is interesting. The cost of Acheson graphite electrodes is from 11 cents to 16 cents per pound while carbon electrodes sell in this country for from 3½ cents to 6½ cents per pound, prices of each varying with size of order and shape of goods.

Both are resistant to oxidation, for the carbon is so molecularly dense² that it has but a slow affinity for oxygen. As most electric furnaces have a neutral or reducing atmosphere the oxidizing of the electrodes by the oxides of the charge is not great. There is, of course, a slight regular consumption of both. The writers cannot find that for rated loading, graphite electrodes are more resistant or oxidizing influences of the continuous zinc furnace than is the amorphous carbon electrode.

Acheson graphite, however, tends to increase the heat losses of an electric furnace unless conditions are such that they can be loaded up so that the heat generated in them is such that they are hot clear to where they protrude through the furnace roof.

As the Acheson graphite electrodes have been subjected to a second heating and annealing process, any electrode with small or concealed "check" has been passed to the scrap heap. As the heat conductivity is higher they are more resistant to the sudden changes in temperature than are the amorphous carbon product.

Both kinds of electrodes can be machined but the Acheson graphite much more readily and exactly.³ The ordinary metal-working or wood-working tools can be used to machine either sort of electrodes. They will turn down easier than brass or bronze and in the case of Acheson graphite the machine is always operated at highest speed. It has been found that the best joint suitable for electric furnace needs is made by using a cylindrical threaded plug (Fig. 1). This is screwed equal distances into the ends of the electrodes to be joined.

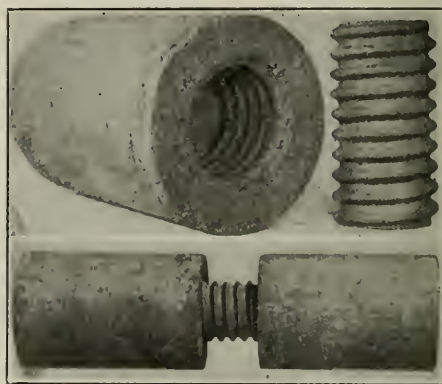


FIG. 1.—JOINING CARBON ELECTRODES

When joined electrodes are used care must be exercised so as to obtain good electrical contact, thus preventing the joint from becoming overheated. By the use of joined electrodes the expensive and wasteful butt ends are eliminated, but only round electrodes can be used for this operation.

Where a furnace working is made more difficult by the lowering of temperature in spots amorphous carbon is superior to artificial graphite.

For most purposes the present difference in price is such as to favor the use of amorphous carbon electrodes. In many places a price ratio of one to two and one-half about equals the value ratio of services of each commodity to user.

A practical smelting furnace should always be made with means to "fish" and pull out a broken stub electrode, through a side door.

Where necessary, electrodes can be sheathed by a tinsmith with sheet steel or copper so as to resist oxidation at spots where they protrude from the furnace. If a heilding is made in thick walls they will not be hot enough to burn at this point. Water-cooling can well be employed in this connection.

Square electrodes or those rectangular in cross-section are preferable in any furnace where the electrodes are permanently fixed, such as the Bailey bar-heating furnace, because of ease of laying in conformity with the courses of brick to round electrodes. Round electrodes are preferable to square electrodes in a steel-refining furnace or a furnace of the buried arc type such as is used in the manufacture of calcium carbide.

Summing up the practical knowledge about the electrode problem, it can be said that the electrodes should be designed as large as possible in order to have an excess carrying capacity

²A comparison of activity of carbon can be made by studying the reduction temperatures of different forms of carbon. See W. McA. Johnson, Bulletin Am. Inst. of Mining Engineers, Aug., 1913, "Reducibility of Metallic Oxides as Affected by Heat Treatment."

³International Acheson Graphite Co., Bulletin No. 6, C. L. Collins, Trans. American Electrochemical Socy., vol. 1, p. 53, 1902; W. McA. Johnson, Electrochemical Industry, Sept., 1904, p. 345.

for the periods when the furnace requires more power than during normal running conditions. But this size should be limited by the amount of cooling done to the "laboratory" of the furnace; also that a number of small electrodes is usually better in a smelting furnace than one or two large ones.

ELECTRODE CONSUMPTION AND ACCESSORIES

Electrode consumption is of two kinds, that lost in the "laboratory" of the furnace by accidental oxidation or incidental oxidation from the oxides in the charge or gases and that

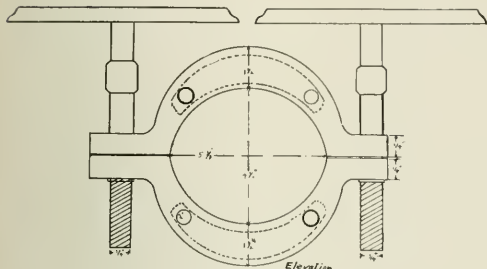
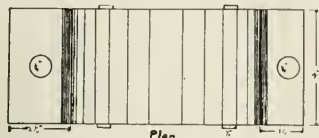


FIG. 2.—WATERCOOLED ELECTRODE HOLDER

lost by breakage or by reason of non-utilization of stub ends. As both amorphous carbon and graphite are much more chemically resistant to oxygen than carbon of coke, coal or charcoal of the charge used in the electric smelting furnace the loss by oxidation rarely amounts to more than 30 pounds per ton of primary raw material and often is less than 10 pounds. In steel furnaces of the Siemens type the total electrode consumption is often less than 25 pounds per ton of finished steel.

Electrode loss by reason of non-utilizable stub ends can be reduced to a minimum by using round electrodes and joining them together as heretofore described. When electrodes are thus joined they are slowly and continuously fed into the furnace at the rate at which they are consumed.

Electrodes can be placed in the furnace in all ways, horizontal, inclined much or little from the vertical, and vertical. Where the *slightest necessity exists*, they should be water-cooled, on connection clamps.

When all the above considerations are in the mind of the designer and when proper distribution of controlling factors of design is made, electrodes are relatively simple parts of electric furnaces. What is important is what goes on between the ends of the electrically opposite electrodes. When the metallurgy of the process is firmly and clearly understood, the electrode problem solves itself.

Where a movable electrode enters the furnace, something resembling a stuffing box must be provided. In an experimental furnace this can simply be proper cutting out of the brick or a form made out of No. 22 or 20 galvanized sheet steel made by a tinsmith and filled in with plastic fire cement.

For high temperature work a stuffing box of Acheson graphite is recommended. For regular operation the stuffing box had best be built of special shape fire bricks made by firebrick manufacturers, or a cast iron, cast steel, or water-cooled cast "composition" metal (a mixture of 95 per cent copper, 4 per cent zinc, 1 per cent tin is recommended) stuffing box used. The cast iron and cast steel stuffing boxes can be advantageously water-cooled.

The taper of the stuffing boxes should be from $\frac{1}{8}$ in. to $\frac{1}{2}$ in. to the foot. The clearance at the lower end should be from $\frac{3}{8}$ in. to 1 in. Where a gas-tight joint is desired, $\frac{1}{2}$ in. to $\frac{1}{3}$ in. asbestos rope can be wound about the electrode or stuffed into the box. (The Continuous Zinc Furnace Co. uses with satisfaction that asbestos fiber made by Johns Pratt Co., No. 688 ranging from $\frac{7}{32}$ to $\frac{1}{4}$ in. in diameter. Mineral wool, graphite, coal dust, fireclay or sand can be poured into fill up the interstices.

With the exercise of a little care and common sense a practical stuffing box for the low pressure existing in the electric furnace is quite easy to design provided some practical experience has been available.

The terminal connection of a small furnace electrode can be either a copper plate bolted or clamped to the flat surfaces of the electrode. In the case of a round electrode a portion of the electrode is cut off making a flat surface. The connection to the line is made through an air cooler. The air cooler consists of several pieces of sheet copper, say, a half dozen of sufficient width so as to be of proper carrying capacity, the sheets being soldered together at both ends. The central portions, for about six inches, are separated from each other so as to permit air currents to pass between them. The air cooler is about eight inches long, the one end being bolted to the main the other either bolted or clamped by an ordinary malleable iron carpenter's clamp. The purpose of the air cooler is to insure the solder of the terminal lug from melting. The use of a clamp is desirable should the contact become poor an 8 in. monkey wrench applied to the thumb flanges will remedy matters.

In any large furnace or experimental furnace where runs of over two days are made a bronze terminal connection with a piece of copper gauze between the terminal connection and electrode is the proper thing. The alloy used for this is "ounce" metal copper 85 parts, zinc 5 parts, lead 5 parts, tin 5 parts.

The form used by the Continuous Zinc Furnace Co. is shown

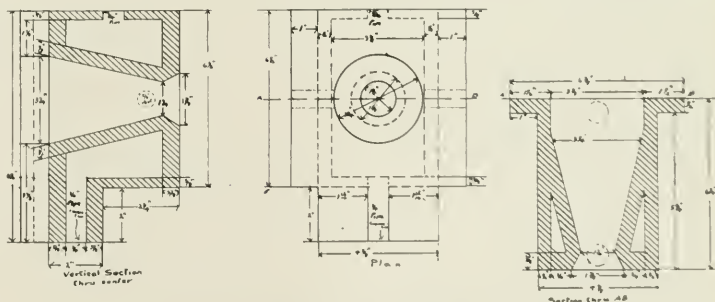


FIG. 3.—CINDER MONKEY OF CONTINUOUS ZINC FURNACE CO.

in Fig. 2. It consists of a connection built up of two exactly similar portions which when closed are just a trifle elliptical in shape to insure a firm hold on the round electrode. The portions can be hinged together and secured by means of a bolt or by two bolts as shown. Water-cooling connection from one portion to the other is provided by means of rubber tubing, or $\frac{3}{8}$ -in. piping. Mr. C. A. Hansen, of the General Electric Co., is an advocate of the use of wedges for tightening contact which is an easy and practical way, for the furnaceman can use a brick bat if a hammer is not handy.

Terminal connections should be "over designed." From 5 to 10 amperes per square inch of contact surface, is the authors' recommended figure.

TAP HOLES

In an electric furnace of the stationary type as compared with the tilting, producing a molten product, provision must be made for removing this product from the furnace. In the

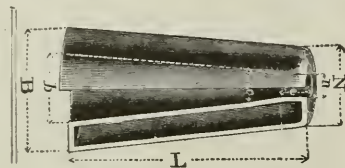


FIG. 4.—BRONZE CINDER MONKEY

tilting furnace this problem is solved, by tilting the furnace and simply pouring at will part or all of the fluid contents.

Some sort of a tap-hole must be made in stationary furnaces. Electric furnace processes follow ordinary processes of furnace tapping.

For slags and metals or metal products that do not attack copper such as copper matte, a water-cooled tap-jacket is decidedly the best. The illustrations give three types of tap-jackets. Fig. 3 shows the one now used by the Continuous Zinc Furnace Co.; the other, Fig. 4, is a standard tap-jacket or "cinder-monkey."

The "cinder or slag monkey" can be set in a "monkey-house." Fig. 5 shows the design of a monkey house for "monkey" shown in Fig. 3. It is made of cast iron, proved to be serviceable and cheap.

All water-cooled "monkeys" should have a good flow of water through them more especially during a tap. *The water must enter at the bottom and come out at the top*, otherwise pockets of live steam are formed and the "monkey" will burn out.

"Monkeys" can be made of an alloy 95 per cent. copper, 5 per cent spelter, or of 98 per cent copper and 2 per cent tin. Both alloys work satisfactorily.

STOPPING THE FLOWING OF SLAG

The flow of slag can be quickly stopped by inserting a tapered bar or a "dolly bar" $\frac{1}{4}$ in. larger than the diameter of the aperture through which the slag flows or by a clay "bott" on a botting bar. The botting bar consists of a disc $2\frac{1}{2}$ in. to $3\frac{1}{2}$ in. in diameter, the disc being $\frac{1}{2}$ in. thick. This is screwed to a $\frac{3}{4}$ in. gas pipe and can be made any length.

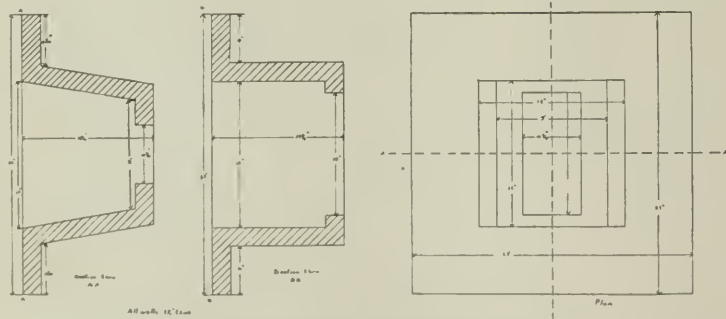


FIG. 5.—MONKEY HOUSE, CONTINUOUS FURNACE CO.

In case the slag buggy or ladle be placed in front of the tap hole as in the case of an experimental furnace the bars should be at least 9 ft. in length so that the furnace man can have a chance to reach over the ladle without having to stand too close to the ladle. A small "teat" or projection about an inch long is attached to the face of the disc so as to hold the bott of clay in place.

The clay bott is best made conical in shape, the base being somewhat larger than the face of the bar, and tapered so as to fit the taper of the slag monkey used. The bar is best moistened so that the bott will stick to it while being put into the monkey.

The furnaceman stopping the flow of the hot metallurgical liquid with either the iron bar or the conical clay bott should have an idea of where the slag hole is relative to where he is standing by co-ordinate sense of mind and body since he is partially blinded by heat. He should stand unanxious and unafraid with muscles and nerves relaxed.

When sufficient material has been tapped he should quickly apply his bar or bott to the "monkey" turning his face away so as to prevent the "sparks," if any, from burning him. These "sparks" are caused by the superficial moisture in the case of the clay bott being turned instantly into steam and are practically unnoticeable when an iron bar without bott is used. Three light taps of the sledge by a fellow workman complete the job. If possible there should be a stand for the workman on a level with the tap hole. By this means the workman looks down and is in a position to see the flow of the hot liquid. Also an iron bar transverse to flow of liquid is a good thing for it allows the workman to rest his botting bar previous to the final insertion.

The amateur had best wear a wide brimmed soft hat, goggles, woolen clothes including a high necked sweater, and leather gloves with gauntlets. (An old fencing mask is ideal for protecting the face and eyes.)

The clay mixture of which the "bott" is made consists of 4 parts crushed firebrick and 6 parts plastic fire-clay well worked up with water to a stiff mud. J. H. Gautier & Co., Jersey City, supply this material in barrels at a reasonable rate and it is better business for a small consumer, such as an experimental station, to buy such material from a manufacturer than to attempt to make it.

Tap-holes can be made of this material in a manner similar to the tap-holes used in the iron foundries, but they are not certain when used with slags or mattes while the water-cooled tap-holes made of metal can not be cut by slags when the water is turned on full. For metals such as steel, copper, lead, etc., a well built tap-hole of fire-clay or other refractory is the thing.

A good temporary tap-hole can be made of a stub graphite electrode or of an old carbon electrode. However, it cannot be drilled into.

GETTING INTO THE FURNACE

Getting into a furnace is usually an easy matter. A steel drill bar is driven into the tap-hole by the use of a 14 to 18 pound sledge hammer. The bar is best made of $\frac{7}{8}$ in. or 1 ft. octagon stuff about 6 ft. long. The end is drawn out to a drill-end as in a wall drill used by electricians so as to give a drilling effect. The "jabbing bar" used to break through the outer crust after a hole has been drilled, till a red color shows, should simply be pointed. When the tap-hole once shows a dull red a bar without much temper will be found satisfactory since it would not retain its temper any length of time under these conditions.

Should the bar stick upon attempting to remove it a "puller" will be found indispensable. The "puller" consists of a heavy piece of iron cut out so that it will just clear the sides of the bar upon which it is to be used. A second piece of wedge-shaped iron is inserted between the first piece of iron and the bar and tapped with a hammer. The "puller" should be made of at least inch stuff.

However, should the contents of the furnace be frozen about

the tap-hole it is clearly impossible to drive the bar in and the furnace must be heated up by putting in an extra amount of power for some time or a hole "arced in" by the use of an auxiliary movable electrode until a bright red is seen when an attempt should be made to drive the tap-bar.

But generally, the proper way is to see that the furnace is well heated and the lower contents liquid. The proper way to get stuff out of a furnace and how best to stop it up can only be learned by experience although the above hints will prove to be helpful as they are based on good theory and on good practice.

In the next article, the authors will deal with the ironing of an electric furnace, with water-cooling, with refractories, with feeds, and other important details and parts of electric furnaces.

Erratum.—We wish to make a correction of an error in the first article published in the last issue of this journal. An average coal with a heating value of 11,150 B.T.U. per pound (instead of 11,600 B.T.U. per pound) contains exactly the same heat as a horsepower year.

The Quantitative Determination of the Precious Metals—Gold, Silver and Platinum*

By Trenkner,

Assistant Assayer, Royal Mint, Berlin

Since the rise in the price of platinum the assay of coins and other alloys of the precious metals has needed considerable modification. The platinum alloys required a very high temperature in cupellation, causing a great loss of the precious metals. The button contained lead in varying amounts; it was brittle and could not be rolled in the usual way.

The platinum when alloyed with silver dissolved in nitric acid with a brown color, though not completely, while the residue fell to a fine powder which was hard to gather and did not agglomerate when heated, thus making direct weighing impossible.

The French Mint in Paris published in its annual report of 1901 its experience. The account showed, however, that the difficulties had not been surmounted.

According to the instructions of the Direction Générale des Contributions Indirectes of June 8, 1910, platinum in gold can only be determined with an accuracy of 10/1000 (or 1 per cent) and the error permitted by law for platinum was, therefore, fixed at 10/1000 (10 millimes).

In the following article a method will be described which allows of the determination of platinum and gold with an accuracy of 0.5/1000, or 0.05 per cent; with more extended experience a still greater accuracy may be reached.

With alloys containing more than 250/1000 gold plus platinum 500 mg. = 1000 A.T. (assay tons) are weighed out; with alloys containing less gold and platinum 1000 mg. = 1000 A.T. are weighed out.

For a preliminary assay the material is disintegrated by hammering and rolling. It is better to use granulations than cuttings obtained with the chisel, since platinum bars show considerable variation; it is also advisable to add silver to facilitate melting and to obtain a homogeneous alloy. Five hundred A.T. of this material are cupelled at a temperature of 900° to 950° C., with an addition of 3 to 15 g. of lead, according to the amount of base metal present. The button always retains some base metal, mainly consisting of lead. With large amounts of platinum present the button weighs more than the weighed out portion, due to the lead retained.

From the appearance of the button its content of platinum and gold may be estimated with some experience. If this fails, the button is treated in the manner described later, but without the caution which is required later on by putting the button into concentrated sulphuric acid, thereby dissolving the silver. The remaining gold and platinum is weighed.

In the regular assay 500 A.T. (generally in duplicate) are

weighed out and chemically pure silver is added, making the proportion of silver to platinum plus gold equal to 10 to 1. This is cupelled with the necessary amount of lead and after deducting the amount of silver added the total amount of silver plus gold plus platinum is obtained by weighing the button.

The loss in cupellation may amount to 5 to 35 A.T.; it depends on the temperature, the amount of lead used, the time of cupellation, the kind of cupel used, the amount of silver present, and the composition of the alloy. This loss must be determined by experiments with alloys of varying composition. More accurate results are obtained by cupelling simultaneously a check assay of approximately the same composition as the one to be assayed.

The alloying with silver has the following advantages: The button is free from lead (see Sharwood, Cupellation of Platinum Alloys Containing Gold and Silver, Jour. Soc. Chem. Ind., Vol. 23, p. 412, 1904), it is easily removed from the cupel and no appreciable amount of gold and platinum is lost by this cupellation, because this loss diminishes with the increase of the silver.

The button is put into a parting flask containing 25 c.c. concentrated sulphuric acid of 1.84 specific gravity; the button is neither hammered nor rolled.

The acid is heated with a round burner without boiling, avoiding uneven heating or overheating. The silver dissolves in from twenty-five to thirty minutes. The residue of platinum, gold and small amounts of silver always retains its coherence, which prevents mechanical losses on decanting. When the acid is heated to boiling considerable bumping takes place; this, however, does not cause the complete dissolution of the silver even when the acid is renewed. Furthermore, platinum is soluble in boiling concentrated sulphuric acid and boiling causes, therefore, a loss of platinum. After cooling somewhat, the acid is decanted carefully and the residue washed three times with boiling water; the wash water is conveniently passed through a filter. The residue in the flask is put into a porcelain crucible in the usual manner, by filling with water and tilting, and is then annealed together with the filter in front of the muffle.

In this way an addition of gold is avoided, also rolling and the making of cornets. The residue always retains its coherence. Cornets under similar circumstances retain their coherence only within very narrow limits so that the amount of platinum plus gold must be fairly well known beforehand.

The residue is dissolved in the smallest quantity possible of aqua regia, the free acid is evaporated as much as possible, and diluted with distilled water. The chloride of silver is filtered. We have now a solution of about 150 cc. containing platinum and gold.

The best and most convenient reduction reagent for the gold is "hydrazin hydrochlorid." It sometimes carries down some platinum (compare Jannasch and Mayer, Ber. d.d. chem. Ges., Vol. 38, p. 2129, 1905); but it is still the best method if certain conditions are observed. Its power of reduction is decreased by an excess of hydrochloric acid and when working at low temperatures.

The gold is precipitated in the solution after adding 15 cc. hydrochloric acid (1.19 specific gravity) and about 1 g. hydrazin hydrochloride. The solution is left standing at 18° to 20° C. for one hour, with occasional stirring, which causes the gold to agglomerate.

After filtering, incinerating, and annealing, the gold is weighed directly (without the crucible). The gold which frequently contains some platinum is melted with a little lead and five to eight times its amount of silver in a cupel in the muffle and then parted with nitric acid, thus dissolving the silver and any platinum that might be present. The remainder is gold.

In the filtrate from the precipitated gold the platinum is precipitated by boiling with an excess of ammonia in about ten minutes (compare Jannasch, Ber. d.d. chem. Ges., Vol. 37, p. 1080, 1904). If the platinum contents are low it is better to evaporate most of the hydrochloric acid present by gentle boiling, as ammonium chlorid in great excess prevents the pre-

*Translated from *Metallurgie*, vol. 9, p. 103 (1912) by A. F. Fuerst.

precipitation of platinum through the formation of complex ions. The platinum may also be precipitated by caustic potash or caustic potash and ammonia, in which case, however, the filter has to be washed very carefully with water containing hydrochloric acid. The platinum may then be weighed directly (without crucible), after incineration and annealing.

In order to show that the gold is precipitated quantitatively under the conditions given the following experiments were made:

(1) 500 A.T. of gold (1000 A.T. = $\frac{1}{2}$ g.) are dissolved in aqua regia and most of the acid evaporated, diluted to 150 cc., 10 cc. hydrochloric acid (1.19) and about 1 g. hydrazin hydrochlorid added, left standing at 18° to 20° C. for one hour, gave exactly 500 parts of gold.

(2) The same conditions, but with addition of 15 cc. hydrochloric acid, 499.7 Au.

(3) 20 cc. hydrochloric acid, 499.8 Au.

(4) 25 cc. hydrochloric acid, 499.5 Au.

(5) 20 cc. hydrochloric acid, 12° to 13° C., 498.8 Au.

(6) 1.5 cc. hydrochloric acid, 12° to 13° C., 499.8 Au.

In experiments (4) and (5)—0.5 and 1.2 parts were not precipitated respectively, while in the other experiments the results are within the limits of error.

(7) 500 pts. Pt, 1 pt. Au, 15 cc. HCl, 1 hour at 18° C., gave 3 pts. of gold containing platinum, which after treatment with nitric acid gave 1 pt. Au and 500 pts. Pt.

(8) 120 pts. Au, 5 pts. Pt, 15 cc. HCl, 1 hour at 18° C., gave 120.2 pts. Au and 5.0 pts. Pt.

(9) 300 pts. Au, 200 pts. Pt, 15 cc. HCl, 1 hour at 23° C., gave 303.1 pts. Au containing platinum, which after melting with silver and parting gave 299.6 Au and 200.4 Pt.

(10) 300 pts. Au, 200 pts. Pt, 15 cc. HCl, 1 hour at 18° C., gave 302.2 pts. Au containing Pt, after alloying with silver and parting 300.1 Au and 200.2 Pt.

Check assays:

(1) Weighed 120 A.T. (1000 A.T. = 1 g.) Au; 5 pts. Pt; 370 pts. Ag; 5 pts Cu. Cupelled with 1000 parts Ag and 3 g. lead. The button weighed 1489 A.T., loss in silver 6 A.T. Precipitated with hydrazin hydrochlorid gave 120.2 Au and 5 Pt.

(2) 5 A.T. Au (1000 A.T. = 1 g.); 5 A.T. Pt; 345 A.T. Ag; 245 A.T. Cu. Cupelled with 8 g. lead, gave 250 Ag + Au + Pt; loss in silver therefore 5 A. At 18° C. 15 cc. HCl, etc., gave 4.8 pts. Au and 5.3 pts. Pt.

(3) 100 A.T. Au (1000 A.T. = $\frac{1}{2}$ g.); 100 A.T. Pt; 50 Ag; 250 Cu. Cupelled with 8 g. lead, after adding 2000 Ag gave 2218 Ag + Pt + Au. Loss in silver, 32 parts. Precipitated at 18° C., etc., gave 103.2 Au containing Pt, which after alloying and parting gave 99.6 Au and 100.2 pts. Pt.

(4) 50 A.T. Au (1000 A.T. = 1 g.); 50 Pt; 20 ag; 380 Cu. Cupelled with 1000 Ag and 10 grs. lead gave 1104, showing a loss in silver of 16 A.T. Precipitated at 18° C., etc., gave 52.4 Au containing Pt, which after treatment with nitric acid gave 50 Au and 50.2 Pt.

The assay of ores and by-products is carried out by scorification or by the crucible method and subsequent treatment of the regulus by the method described.

Volume Changes in Alloys

In a recent paper before the (British) Institute of Metals Mr. J. H. Chamberlain discusses this subject.

The results of the research show that volume changes obtained in alloys are of both theoretical and practical importance.

Their relation to the crystallization intervals as plotted from the equilibrium diagram was clearly shown in the copper-aluminium series. The force producing the expansions, obtained in castings during solidification, was in the case of a copper-tin alloy (90 per cent. copper), a copper-zinc alloy (15 per cent. zinc), and pure aluminium shown to be quite considerable.

With the copper-tin alloys, and pure aluminium castings, the temperature at which the metal is poured appears to have some effect on the expansions observed. The extraordinary large and quiet expansions peculiar to the copper-zinc alloy in the

neighborhood of 15 per cent copper, are always preceded by an arrest; the expansion only taking place during the crystallization of the ϵ variety; microscopic examination and density experiments revealed cavities round the crystal boundaries of this alloy, and these are produced by the absorption of d by ϵ crystals.

On slowly heating up bars of a copper-tin alloy (90 per cent. copper), a copper-zinc alloy (15 per cent. copper), and pure aluminium in the electric furnace, contractions in the length of the bar were obtained in each case. On allowing the bar of the copper-zinc alloy to cool slowly a most extraordinary expansion was obtained, far larger than that yielded on casting, and on repeating the heating-up and cooling-down operations five times the bar was found to have grown in length from 10 inches to 11 $\frac{1}{2}$ inches, and also to have increased in sectional area. The author is of the opinion that the contraction of the copper-zinc alloy when slowly heated may be due to the liquidation of ϵ crystals, the liquid filling up the cracks existing in the casting, the skeleton crystals left intertwining. The large expansion and the continued growth of the bar on repeated heatings may be due to this filling up of the cavities, followed by the crystalline change from d into ϵ . With the copper-tin alloy (90 per cent. copper) the expansion is apparently produced by the chilling of the liquid round alpha dendrites, since it is not observed when once the bar has been annealed. With aluminium the volume changes observed on heating and cooling are probably due to the presence of some gaseous impurity.

Phosphorus in Copper—Aluminium Alloys

A recent paper read by Professor A. A. Read before the (British) Institute of Metals discusses the influence of phosphorus on some copper-aluminium alloys.

Some mechanical and physical properties of a number of copper-aluminium alloys with 5 or 10 per cent. of aluminium and varying amounts of phosphorus have been investigated. The effect of small quantities of phosphorus, up to 0.2 per cent., on the 5 per cent aluminium alloy is to diminish its capacity for hot rolling, though its behavior in cold drawing is improved. The yield point and maximum stress of the annealed metals are slightly raised, and the reduction of area is lowered without, however, affecting the elongation. The resistance to alternating stress is slightly increased. Tested in the form of chill castings with varying amounts of phosphorus, up to 1 per cent., the yield point was found to rise considerably; the ultimate stress attained a maximum at 0.52 per cent., but the elongation and reduction of area fell rapidly. Phosphorus has a hardening effect above 0.25 per cent.

The 10 per cent aluminium alloy suffers considerable loss in its mechanical properties by annealing; 0.1 per cent. of phosphorus slightly improves the character of the annealed metal. There is, however, no improvement in the properties of the cold drawn metal or of the chill castings.

Phosphorus cannot be added to this metal in greater amount than about 0.2 per cent. on account of the formation of aluminium phosphide, which gives off phosphine on exposure to the moisture of the atmosphere. Phosphorus lowers the melting point and conductivity for electricity of both the alloys. When present in the rolled metal it reduces the loss due to corrosion of these alloys by sea-water. The presence of phosphorus up to 0.11 per cent. does not affect the characteristic alpha and beta structure of the 10 per cent aluminium alloy. Above about 0.2 per cent., however, the cast metal shows specks of phosphide in the beta constituent. The phosphorus in the cast alloys containing 5 per cent of aluminium separates out as a phosphide in the boundaries of the crystals of the alpha constituent. On annealing these boundaries ball up, and if less than 0.25 per cent. of phosphorus be present the phosphide passes completely into solution. The structure of the annealed and cold drawn metals, with 0.2 per cent. of phosphorus and under, is precisely similar to that of the pure 5 per cent. aluminium alloy, except that the alpha constituent containing phosphorus assumes a deep bronze color on etching with an acid solution of ferric chloride.

Colorado Meeting of the American Electrochemical Society

THE twenty-fourth general meeting of the American Electrochemical Society, held during the week of Sept. 8, at Denver, Boulder, and Golden, Colo., with a final session on top of Pike's Peak and an excursion to Cripple Creek, was the first meeting of the society west of Chicago and will forever be remembered as a most successful and lively meeting. It will be remembered for the most interesting and lively discussions in the technical sessions, for the equally spirited and enjoyable social and sight-seeing functions, and for the overwhelming hospitality which was extended to the Society by Colorado. As we comment on the scope of the convention in our editorial pages, this report may enter at once *in medias res*.

The attendance from the East was less than had been hoped for and expected. But while disappointing at the start, this fact did not detract in the least from the success of the meeting. As a matter of fact, it helped to produce quickly that close social intercourse by which in a short time everyone knew everyone and was everyone's friend and brother.

Denver

The total registry list contained 109 names, given in alphabetical order at the end of this report. Out of these there were fifty-two from states outside of Colorado. The Eastern delegation arrived in Denver on two special cars in the early afternoon of *Monday*, Sept. 8, and were received at the railroad station by the local committee enforced by other Denver engineers and business men, and a score of automobiles.

The success of the local committee in making all arrangements was really extraordinary and not enough credit can be given to their work. The committee consisted of the following members: H. C. Parmelee (chairman), W. W. Case, Jr., C. W. Comstock, W. H. Davis, H. C. Eastman, J. B. Ekeley, H. S. Evans, S. Fisher, O. P. Fritchie, F. S. MacGregor, J. L. Malm, G. H. Sethman, C. D. Test, H. J. Wichman and H. J. Wolf.

It was a grand idea to take the party from the station not right to the hotel, but to show them first the beauties of the Queen City of the Plains. After the long railroad journey this crisp automobile ride was wonderfully refreshing. The party went through the big Welcome Arch up Seventeenth Street to the new extensive boulevard and park system, where the greensward produced by constant irrigation was duly admired, back through another residence section of the city, out to Inspiration Point, and finally to the headquarters, the Hotel Shirley, where registration began.

On *Tuesday* morning, the first technical session was held in the Shirley Hotel. A report of all the technical sessions will be found later on.

On *Tuesday* afternoon visits were paid to the plants of Sutton, Steele & Steele Company, the Western Chemical & Manufacturing Company, and the Henry E. Wood Ore Testing Company.

Smoker

In the evening of *Tuesday*, an invitation smoker was held in the Pipe and Bowl Room of the University Club. It was the liveliest smoker of the many lively smokers the society has held.

The committee consisted of C. W. Comstock (chairman), H. C. Eastman, F. S. MacGregor, H. C. Parmelee, and F. A. Lidbury, who, as representative of Section Q, was in charge of the proceedings.

A program had been printed, the only weakness of which is that it cannot be reprinted here. There were refreshments and songs, much of the "poetry" having been prepared to idle away what would have been dull moments on the railroad trip in coming to Denver. There was a glorious revival of the Electrochemists' Lovesong: "Sue, dear, turn out the tungstens." There was a wonderful cartoon of the Metallurgical

Cow. There were more refreshments and more songs. There were speeches and the home talent of the Denver University Club shone brilliantly. There were limericks. There was a lecture in rhymes, by Dr. Chauvenet, on ferromanurium. There was not one dull second till the climax was reached and the party broke up with the song of "Auld Lang Syne."

Boulder

On *Wednesday* morning special cars were taken to bring the party to Boulder, where the society was shown through the power plant, Professor Ekeley's chemical laboratory, and Professor Evans' electrical laboratory. In the latter the "Elderkin rectifier" was exhibited, which is essentially a commutator driven by a synchronous motor.

The second technical session was held in the University of Colorado. A report of this session will be found below.

After the meeting the society was entertained at luncheon by the university and was cordially addressed by President James F. Baker, of the University. The very enjoyable luncheon was concluded by a hearty vote of thanks to the University and President Baker.

Following the luncheon the members and guests of the society, together with their ladies, were taken by automobiles, provided by the courtesy of the Boulder Motor Club, up beautiful Boulder canyon, where the power plant of the Central Colorado Power Company was inspected. A few of the party extended the trip further up Boulder canyon into the tungsten district at Nederland. It was a truly wonderful "joy ride."

The whole party returned in the evening to Denver.

Golden

On the morning of *Thursday* some visits were made in Denver to the United States Mint, the American Zinc Ore Separating Company, the Globe Smelter of the American Smelting & Refining Company, and the plant of the Screenless Sizer Company.

At 11 a. m. special cars were taken for Golden, where the party were received by professors and officials of the Colorado School of Mines. The society was cordially entertained by the Colorado School of Mines at luncheon in Guggenheim Hall. In an after-luncheon speech the President Emeritus of the School of Mines, Dr. Regis Chauvenet, spoke in poetry of the work of the mining engineer and then expressed the hope that this first Western meeting of the American Electrochemical Society might help to electrify the Rocky Mountains. The hearty thanks of the society were then extended to the School of Mines and Dr. Chauvenet.

After some of the laboratories had been visited, the third technical session followed, a report of which will be found below. After the session the new ore-dressing and metallurgical plant of the Colorado School of Mines was visited. In our issue of May, 1912, pages 269 to 272, we gave a detailed account of the purpose and equipment of the plant; and in a later issue, March, 1913, page 148, we published a statement of the charges for use of any part or all of the plant by private engineers.

Since that time the plant has been brought sufficiently near completion to make the set of interior views of the plant on pages 587 and 589 of additional interest. Some additional equipment is contemplated by the School authorities and will be added as funds become available.

Picnic Dinner

For the evening an informal picnic dinner, tendered by the Colorado Scientific Society, was on the program. The Colorado Scientific Society, of which Mr. G. E. Collins is the president, is the intellectual center of the Rocky Mountain region, and whoever has come in close contact with the big men of that



PHOTOGRAPH TAKEN AFTER FIRST SESSION

region, has heard of the famous yearly dinners of the Colorado Scientific Society at the Denver University Club. They say a man would rather give up Christmas than miss such a dinner.

Now, it was significant that the Colorado Scientific Society undertook to entertain the American Electrochemical Society at dinner on the last day of the Denver-Boulder-Golden meeting; and it was equally significant that a formal full-dress dinner was tabooed and an informal picnic dinner was arranged behind Golden, on top of Lookout Mountain. They say at night Denver presents a wonderful sight from Lookout Mountain—a mass of lights on the plain.

It was to be the crowning success of the entertainment of the American Electrochemical Society by its Denver friends. It was not so to be. The rain came down in torrents. It clearly made the ascent of Lookout Mountain impossible. But it could not spoil in the least the happiness of the party. It could not spoil in the least the picnic dinner, which was immensely enjoyed in Guggenheim Hall. Once more was heard the song of "Auld Lang Syne" and the party went back to Denver.

Colorado Springs and the Pike's Peak Session

On Friday morning the party was taken to Colorado Springs, where the members were received at the station by President William F. Slocum, of Colorado College.

Shortly before noon the ascent of Pike's Peak began via the Cog Wheel route from Manitou. The atmosphere was clear after the rain of the night before, though there were clouds enough. But the view from Pike's Peak was glorious through the shifting clouds. When the party arrived by train, they were received by Messrs Bailey, Crosby, and Hinckley, who had walked up to the top the night before. Snow-balling was enjoyed, some light refreshments were taken and innumerable snapshots.

Then the last session of the meeting of the American Electrochemical Society was held on top of Pike's Peak. It is believed that this was the first official session held by any scientific or engineering society at this place or elsewhere at such altitude (the books usually state the altitude as 14,108 ft., but the bench mark of the U. S. Geological Survey in the room where the session was held says 14,083 ft.).

The only business done at this session was the launching of the Colorado Section of the American Electrochemical Society. This was done on motion of Mr. Parmelee, Professor Fkeley, and Professor Wolf, with great enthusiasm, and thus ended officially the Colorado meeting, with the heartiest invitation extended by the East to the West to attend the next spring meeting in New York City.

On the return to Manitou a large number of the party enjoyed an extended ride through the Garden of the Gods.

Cripple Creek

On Saturday morning the party went by a special train to the Cripple Creek district. The party was joined by quite a number of the members of the Public Health Association, which had just concluded its meeting in Colorado Springs.

The trip on the Short Line up to Cripple Creek is truly described in the advertisements of the company as a "trip that bankrupts the English language," so we will not try to describe it, but the immense enjoyment of this trip must be recorded.

Before the trip began a large number of the party visited the Golden Cycle Cyanide Mill in Colorado Springs. In the Cripple Creek district the majority of the party visited the Portland Cyanide Mill in Victor, while the remainder enjoyed a trip over the electric lines to Cripple Creek.

In the evening the party broke up. A large majority of the Eastern members left at 11 o'clock via the Santa Fe for Kansas City and Chicago.

Visits to Plants.

Visits were made to a variety of metallurgical plants; and although these were not of an electrometallurgical nature, nevertheless they held considerable interest for the visitors, to many of whom the processes were new and suggestive. It would not be a bit surprising, if, as a result of these visits, some of the apparatus and methods worked out for Western metallurgy should find application in the Eastern chemical industries; especially various apparatus originally devised for the cyanide process, suggested to the visitors new fields of application in various chemical industries.

The first plant visited was the experimental mill of the *Sutton, Steele & Steele Company*, where through the courtesy of Mr. W. S. Myers, the president, special demonstrations were made showing the separation of zinc and iron sulphides by electrostatic means, and the dry concentration of zinc-lead-iron sulphides on pneumatic tables. The grading of grains and seeds also was shown, using the same tables as for ore concentration.

At the *Western Chemical Manufacturing Company*, Mr. L. B. Skinner, superintendent, conducted the party through the plant. This company manufactures commercial and C. P. acids and ammonia, liquid carbonic acid and anhydrous ammonia. One of the features of the plant is the gas-producer power plant, the exhaust from the gas engine being purified and converted into liquid CO₂. The company operates also a 100-ton concentrating mill, treating complex sulphide ores by a process which involves roasting, magnetic separation of the iron constituent, and wet dressing of the lead-zinc residue. Sulphuric acid is made from the roaster gases.

The *Henry E. Wood Ore Testing Company's* mill was of



IN SHIRLEY HOTEL, DENVER

interest on account of the variety of methods available for testing different kinds of ore. In addition to the standard methods of concentration, cyanidation, etc., a method of flotation of sulphide minerals invented by Mr. Wood was demonstrated for the benefit of the visitors. This process has already been described in this journal.¹

Electrostatic separation by the Huff system was demonstrated at the laboratory of the *American Zinc Ore Separating Company*, by Mr. A. M. Plumb, who also exhibited and operated a new dry jig of his invention. The latter is entirely new in the field of concentration and treats dry ore-pulp by pneumatic jigging. In operation, the machine is comparable to the ordinary hydraulic jig, except that a pulsating current of air is used instead of water.

The *Globe Plant* of the *American Smelting & Refining Company* was visited by some of the members who were particularly interested in seeing a typical western lead smelter. Mr. Frederick Roeser, superintendent, conducted the party through the plant. The principal points of interest were the Huntington-Heberlein and Dwight-Lloyd processes of sinter-roasting, the recovery of blast-furnace dust and fume by the bag-house, and the volatilization and purification of arsenic trioxide from bag-house fume.

The *United States Mint* was also visited and attracted much interest and attention, although the electrolytic refining of gold, in use at this plant, had been temporarily discontinued.

Considerable interest centered in the visit to the *Screenless Sizer Company*, where Mr. B. F. Rice gave demonstrations on the McKesson-Rice screenless sizer. Three types of machines were operated; one each for coal, crushed rock and ore. The screenless sizer is a mechanical device for sizing dry granular material without the use of screens of any sort. Detailed reference has been made to this machine in an earlier issue of this journal.²

At Colorado Springs an opportunity was given to visit the cyanide mill of the *Golden Cycle Mining Company*, under the direction of Superintendent A. L. Blomfield. This mill is the largest of Colorado's cyanide plants, having a capacity of 1000 tons of ore per day. It treats only Cripple Creek ore, and is operated not only on company account, but also as a custom mill. The process consists in roasting to oxidize the sulpho-tellurides, after which the coarse gold is recovered by concentration on blankets. The residue is classified into sand and slime, the sand being leached with cyanide solution and the slime thickened, agitated and filtered by vacuum. The resulting gold-bearing solution is precipitated with zinc, and the gold precipitate fluxed and melted.

A different type of cyanide mill was found in the new *Portland Mill* at Victor, in the Cripple Creek district, where Mr. L. W. Lennox, assistant superintendent, conducted the party and explained the steps in the process. Here only low-grade mine and dump ore is treated, having a gross value of not more than \$3.25 per ton. The ore is ground in cyanide solution in Chilean mills, and concentrated on Wilfley tables to remove the sulpho-tellurides which are shipped to the smelter. The remaining pulp is classified into sand and slime, the former being discarded and used for stope filling in the mine. The slime is thickened and agitated, and finally filtered on a rotary type of continuous vacuum filter. The gold solution is precipitated by zinc dust and collected in filter presses, from which it is removed from time to time, fluxed and melted.

Entertainments

In addition to the social functions mentioned above several dinners and luncheons were given for visiting members by local engineers. Mr. G. E. Collins was host at a lunch at the Denver Club. Mr. C. W. Comstock tendered a dinner at the University Club. Mr. J. V. N. Dorr entertained a party at dinner at the Antlers, Colorado Springs. Mr. J. P. Evans, Mr. H. C. Parmelee, and Mr. L. Searing entertained other parties in Denver.

Ladies' Entertainment

The entertainment of visiting ladies was in the hands of a local committee consisting of Mesdames G. F. Collins, I. B. Skinner, J. V. N. Dorr, and H. C. Parmelee, of Denver, Mrs. J. B. Ekeley, of Boulder, and Mrs. C. D. Test, of Golden.

On the first day of the meeting a well-appointed luncheon was served at noon in the "blue room" on the fourteenth floor of Daniels & Fisher's tower, in Denver. In the evening Mrs. Collins entertained at dinner, following which there was a theater party.

At Boulder an automobile ride was arranged, giving a wonderful view of mountains, foothills and plains.

During the afternoon session at Golden, a reception and musicale was held at the home of Mrs. Test.

All of these functions were delightful, and were thoroughly enjoyed by the ladies who were fortunate enough to attend the meeting.

In the following we give a complete report of the papers presented at the technical sessions, together with discussions

Technical Session at Denver

The first technical session was held in the Shirley Hotel, Denver, on Tuesday, September 9.

The meeting was called to order at 10 A. M. by the president of the Society, Dr. E. F. Roeber, who expressed the sincere thanks of the Society for Colorado's wonderful hospi-

¹March, 1912, p. 132.

²October, 1911, p. 553.

tality. Mr. H. C. Parmelee, the Chairman of the Denver local committee, followed with a number of announcements on the arrangements made for the afternoon and the following days.

The secretary of the Society, Professor Joseph W. Richards, read a telegram of good wishes from the National Electroplaters' Association.

The following four papers were then read and discussed: Some Aspects of Heat Flow, by E. F. Northrup; Flow of Heat Through Furnace Walls and the Shape Factor, by Irving Langmuir; the Heat Resistivity of Graphite and Carbon, by J. W. Richards; the Electrolysis of Cyanide Solutions, by E. F. Kern. In the absence of the authors the papers of Dr. Northrup and Dr. Langmuir were presented by Mr. Hansen, the paper of Dr. Kern by Professor Schoch.

Some Aspects of Heat Flow

The elaborate paper by Dr. E. F. Northrup, of Princeton University, on "Some Aspects of Heat Flow," emphasizes in the introduction the advantages of viewing a problem in its various phases through the true analogies which it bears with similar problems. After a discussion of the general equations expressing analogies, the author takes up specifically heat flow analogies and gives a long table of the physical analogies and dissimilarities between flow of heat and flow of electricity.

The author then discusses why measurements of thermal conductivity are difficult. The first difficulty is that in producing a flow of heat with which to make the measurement, by creating a temperature difference, we vary more or less the quantity we desire to measure.

The second difficulty is the extreme slowness with which

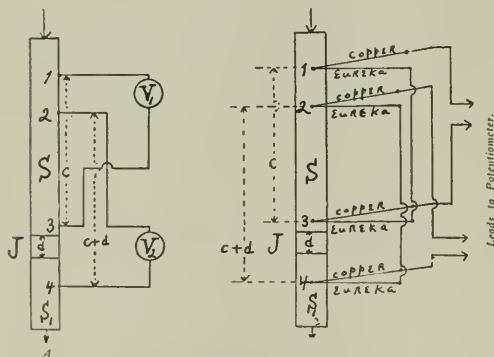


FIG. 1.—AN ANALOGOUS METHOD FOR MEASURING ELECTRIC AND THERMAL CONDUCTIVITIES

temperature equilibrium is re-established after once being disturbed. The third and perhaps the greatest difficulty results from the impossibility of giving to any path of heat flow a thermal insulation which even approximates perfection. This difficulty increases as the thermal conductivity of the material being measured decreases.

A fourth and less understood difficulty which if not surmounted will introduce very great errors into the measurements, is the very large surface film resistance which is always found at the surfaces of thermal conductors.

In the light of the analogies between heat and electrical phenomena, given in the first part of the paper, the author discusses in the second part how the above difficulties may be in part overcome, by using methods of measurement analogous with the fundamental electrical methods for measuring electrical resistance.

The four fundamental methods of measuring electrical resistance are:

(a) By measuring the fall of potential and the current which flows,

(b) By comparing the fall of potential over a standard resistance and the specimen,

(c) By the balance method of the Wheatstone bridge, and

(d) By the balance method of the Kelvin Double Bridge.

In adapting any of these methods to the measurement of thermal resistance, a thermocouple is substituted for the voltmeter or potentiometer. But there is no single thermal device, as a substitute for the ammeter. When it is required to measure heat flow, it is, therefore, necessary to determine the change per unit of time produced by the heat in some material substance. The quantity of water raised one degree or the quantity of ice melted in the unit of time are methods commonly employed to determine the rate at which the heat flows or is delivered. Of course, the rate at which heat is produced by the source of heat can be very precisely measured and if the heat which flows through to the opposite side of the specimen is also measured the quantity of heat which does not pass between the temperature points can be determined.

Method (a). The analogous of the electrical method (a) stated above is to determine the mean thermal resistance by measuring the three quantities. The temperature difference, the heat which flows in the unit of time, and the elapsed time.

As this is the least precise of the electrical methods it is even less so in the thermal case. The time and the temperature difference can be measured with high precision by the quantity of heat which flows through a selected path cannot. The difficulty is largely, of not chiefly, due to the impossibility of knowing the paths in which all the heat flows.

Method (b). An analogue of the electrical method (b), applied to determinations of thermal resistivities, has been carefully tested by the author in its application to measuring the thermal resistance between two plane surfaces which press together.

Referring to A, Fig. 1, let a rod of homogeneous material and uniform cross-section be joined end on with another rod of like material and cross-section. At the surface of union J let there be either a contact resistance or another short rod of different material and of length d.

Let 1, 2, 3, 4, be potential points so located that the length C of the material S included between points 1 and 3 is exactly equal to the length of the material S included between points 2 and 4. Let d_1 be the deflection of the instrument V_1 and d_2 the deflection of the instrument V_2 . Both instruments must have the same constant but they need not necessarily read in volts.

Then $d_1 \propto C$ and $d_2 \propto C + x$ which gives by division

$$x = \frac{d_2 - d_1}{d_1} C$$

This equation gives the resistance of the length d of the material at J in terms of a length x of the material S. If c is taken in centimeters x is in centimeters. If the length d of the material J is made zero as nearly as possible then x is the contact resistance of the point at J in equivalent lengths of the material S. This method will be recognized as the one employed by F. A. J. Fitzgerald and A. T. Hinckley⁸ for measuring the resistance of electrode joints.

Refer now to B, Fig 1, and we have the exact analogy for the thermal case. The gap at J may be filled with a film of air or liquid, or the gap may be closed by pressing the piece S tightly against the piece S_1 . In the former case x gives the thermal resistance in terms of equivalent length of S of a film of liquid of thickness d, and in the latter case the thermal resistance of a joint between S and S_1 for the particular pressure employed.

In the apparatus constructed S and S_1 are round rods 3.8 cm. in diameter, of electrolytic copper. The ends at J are very true surfaces. The separation at J can be measured by a micrometer, and the length C is made very accurately 10 cm. The rods are carefully heat-insulated with baked lavite. The lower end is kept at zero degrees in ice water, and the upper end is heated electrically.

It may be mentioned here, however, that the thermal contact

⁸Trans. Am. Electrochem. Soc., 23, 333 (1913). MET. & CHEM. ENG'G, May, 1913, vol. 11, p. 279.

resistance of the two plane copper faces when pressed together with a moderate pressure is equal to 31.2 cm. of length of copper. It will be noted that the precision of the method is unaffected by any leakage of heat between the points 2 and 3, and only the difference in the leakage between the points 3 and 4 and the points 1 and 2 can affect the measurement. As the distances $1=2$ and $3=4$ are very small, the former being only 0.6 cm., and as the lavite insulation is very good, errors from this heat leakage are thought to be very small.

The method of measuring thermal resistances by comparing

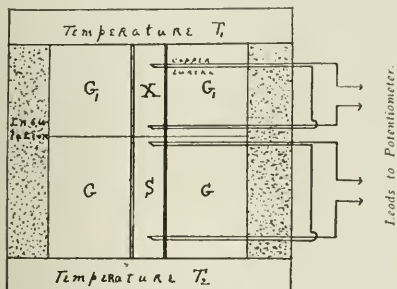


FIG. 2.—METHOD OF MEASURING THERMAL CONDUCTIVITIES

the fall of temperature over a standard material and the material to be measured should prove a very precise one if properly carried out. The diagram, Fig. 2, suggests in outline the way the author would propose to execute the method for either metallic materials or so-called heat insulators.

The standard material S should be copper for the metals and a substance like alundum of a standard grade and composition for high thermal resistance materials. G, G is a guard cylinder of the same material as the standard, and G, G , a guard cylinder of the same material as the sample X .

The results obtained should give the ratio $X/S = E_1/E_2$ with excellent precision. The proposed method is exactly analogous to the electrical method (b).

Method (c). No exact analogy to the electrical method (c) occurred to the author as having much value in thermal measurements, but the method of the Wheatstone bridge suggested a method for obtaining the relative thermal conductances of similar coils of wire when not impregnated and when impregnated with different insulating compounds.

This method was tested under the supervision of the author by Mr. William Eves, 3rd, and described, together with the results obtained, in an article by Mr. Eves in the September, 1913, issue of this journal, page 503.

Method (d). Concerning the thermal analogue to electrical method (d) Dr. Northrup speaks as follows:

"The thermal analogue of the Kelvin double bridge method (d) of measuring low resistances and of obtaining the conductivity of low resistance, large temperature coefficient wires in terms of a standard of conductivity (Hoopers' Bridge Method) has not been tried out by me. I have, however, tested experimentally the principle of the method and I believe it is capable of being developed into a very useful method. The diagrams *A* and *B* of Fig. 3 below are self explanatory.

"In either of the diagrams (*A*) or (*B*), the galvanometer shows no deflection and the bridge is balanced when $S/X = a/b$, provided the ratios a/b and a/b are maintained the same. The connection in *B* shown in dotted line may or may not be made.

"In the thermal case, thermocouples are the sources of current which operate the galvanometer. The method is a zero method in every respect and it is not necessary to know the quantity of electricity or the quantity of heat which flows. By maintaining the cold end as much below as the hot end is above the surrounding temperature it is evident that heat losses from the sides of the thermal path would not introduce

much error. Side guards through which the temperature gradient is made the same as that through the standard and the sample would effectually reduce this source of error."

In conclusion, Dr. Northrup expresses the following opinions on the use of standards of thermal conductivity:

"It will be at once recognized and perhaps objected to, that the above methods, taken as the analogues of the electrical methods for measuring resistance, give only comparative results.

"But why not adopt standards of thermal resistance, and standards of thermal conductivity the same as in electrical measurements we adopt standards of electrical resistance and a standard (Matthiessen's Standard) of electrical conductivity? It seems to me that great advantages would result from this procedure. Some of these advantages are apparent from the suggestions regarding measurements given above. Others are.

"The corrections for losses of heat through the insulation of the path of the heat flow, could be made negligibly small.

"The principles of the Kelvin bridge, which avoids all contact resistances (surface film resistances in the thermal case) could be applied.

"The measurements could all be made by accurately measuring only temperature-differences, the absolute temperature being only roughly required.

"The measurements could be carried out at absolute temperatures chosen through a wide range.

"The measurement of a quantity of heat is never required.

"To be sure, all results would be given in terms of a ratio of thermal resistance to the thermal resistance of the selected standard. But when by long and carefully conducted investigation the specific thermal resistance of the material selected for

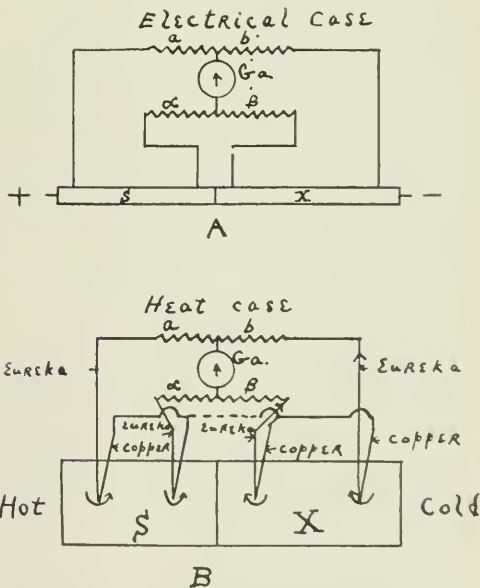


FIG. 3—APPLICATION OF THE KELVIN DOUBLE BRIDGE METHOD TO THE DETERMINATION OF THERMAL CONDUCTIVITIES

the standard is obtained, all comparisons made with this standard can be expressed in the usual thermal units.

"I would suggest that two standards be adopted—one for low thermal conductivity materials and one for high conductivity materials. The first might consist for example of a standard grade of alundum cement packed under a given pressure and baked to a maximum given temperature. The second might consist of 100 per cent electrical conductivity copper. The thermal conductivity of this last is already pretty

well known, and one careful investigation would determine the other.

"I offer this suggestion in the hope it may meet with favor and prove to be as good in practice as it seems to be in theory.

"Interest in the properties of matter at temperatures above those at which organic life can exist, but at which very many industrial processes are carried on, is constantly increasing. I venture to say that no method in learning how to investigate those properties will prove more fruitful of results than the method whereby phenomena are viewed in the light of their natural physical connections and the analogies which join them together in groups and classes."

Dr. Northrup's paper was briefly discussed by Dr. J. W. Richards and Mr. C. W. Comstock with reference to a few details in the author's table of comparison between electricity and heat flow.

Flow of Heat Through Furnace Walls—The Shape Factor

The paper on this subject by Irving Langmuir, E. Q. Adams and G. S. Meikle, of the Research Laboratory of the General Electric Company, Schenectady, is an important continuation of Dr. Langmuir's paper presented at the Atlantic City convention of the Society (this journal, May, 1913, p. 278). The ordinary formula for the flow of heat through plates or rods of solid materials may be written:

$$H' = A/tk(T - T_0)$$

where H' = the heat flow, expressed in watts,

A = area of plate or cross-section of the rod,

t = thickness of the plate or length of the rod,

$T - T_0$ = the difference of temperature between the two sides of the plate or between the ends of the rod,

k = heat conductivity of the material of the plate or rod in watts per cm. per degree.

In most practical cases of heat flow the problem is more complicated than this. Usually the heat is not flowing between parallel surfaces, or at least the areas of the two bounding surfaces are different. The above equation, as it stands, applies only between parallel surfaces through a body of uniform cross-section. For bodies of other shapes, A will vary along the path of heat flow. However, if the area of inflow and outflow are fixed, the ratio A/t will have a definite value, depending only on the shape of the body. This quantity we shall call the shape factor, and will represent it by S . Our formula thus becomes:

$$H' = Sk(T - T_0)$$

In case the heat conductivity is a function of the temperature, the equation should be written:

$$H' = S \int_{T_0}^T k dT$$

S , shape factor, is a quantity of the dimension of a length which depends only on the shape and size of the body and the position of the surfaces by which the heat enters and leaves the body.

It is with the determination of this shape factor that the present paper deals. The first part of the paper discusses in detail the theoretical determination of the shape factor, the second part describes an experimental determination, based on the analogy between flow of heat and flow of electricity. The chief results are summed up as follows:

Only in three simple cases, planes, cylinders and spheres, can S be determined rigorously by calculation. In other cases certain simplifying assumptions must be made, so that the results are of limited accuracy.

There are three types of assumptions that are useful in calculating the shape factor:

A. Assuming some given shape for the isothermal surfaces. This is equivalent to placing in the body perfectly conducting but infinitely thin surfaces. The shape factor calculated in this way is, of course, too high, and must be considered an upper limit. Many different assumptions may be made regarding the shape of the isothermal surfaces, and since the results will always be too high, that assumption is the best which gives the lowest result

B. Assuming some given shape for the heat flow lines. This is equivalent to compelling the heat to flow in a given way by interposing infinitely thin surfaces of perfect heat insulating material. The shape factors calculated this way will be too low, and the best assumption is that which gives the highest value.

By applying both methods *A* and *B*, we can usually show that the shape factor must lie between quite narrow limits.

C. Very good approximate results may often be obtained by imagining parts of the surfaces of inflow and outflow of heat to be replaced by spheres or cylinders of equal area.

Experimentally the shape factor may be very conveniently determined for bodies of any shape by constructing small models of copper and glass and determining the electrical resistance of a saturated copper sulphate solution placed in them.

The values of the shape factors of heat insulation placed on bodies of various shapes have been determined both theoretically and experimentally. The results are given in the following formulas:

Parallel Planes: $S = A/t$ where A is the area of the planes and t the distance between them. This applies rigorously when the planes are infinite in extent.

Concentric Cylinders:

$$S = \frac{2\pi l}{\log_e \frac{b}{a}} = \frac{2.73 l}{\log \frac{b}{a}}$$

where l is the length of the cylinders, b the diameter of the outer cylinder and a the diameter of the inner one. This formula is rigorous for infinitely long cylinders.

Concentric Spheres:

$$S = \frac{2\pi}{\frac{1}{a} - \frac{1}{b}} = \frac{\pi ab}{t} = \frac{\sqrt{AB}}{t}$$

where b is the diameter of the outer, a the diameter of the inner sphere, B the surface of the outer and A the surface of the inner sphere, t the thickness of the insulation, i. e., $\frac{1}{2}(b - a)$. This formula is rigorous.

Square Edge: The shape factor is calculated for the inner surface up to the edge by the formula $S = A/t$, where A is the surface inside the insulation and t the thickness of the insulation. To this is added $0.54 \Sigma l$, where Σl is the sum of the lengths of all the square edges. The result is the shape factor for the whole insulation. The above factor 0.54 was experimentally determined. The theory gave 0.477 as a lower limit.

Square Corner: To the shape factor of the plane faces (inside the insulation) and the square edges add that of the square corners, which is equal to $0.15 n t$. Here n is the number of corners and t is the thickness of the insulation.

Plane Edge: Where two square edges are so close together that their distance apart is less than one-fifth of t , the thickness of the insulation, then the shape factor is found by adding to the shape factor of the faces that of the "plane edge," which is equal to $0.93 \Sigma l$. Here Σl is the total length of the plane edges. This factor 0.93 was determined experimentally. The calculated value was 0.95 .

Plane Corner: To the shape factor of the plane faces (inside the insulation) and plane edges add that of the plane corners, which is $0.087 n t$. Here n is the number of such corners and t is the thickness of the insulation. The factor 0.87 was experimentally determined. Calculation gave 0.15 .

Small Square Rod: Where all the dimensions except the length are less than one-fifth the thickness of the insulation, then the shape factor is obtained from the formula:

$$S = \frac{6.4 l}{\log_e \frac{b}{a}} = \frac{2.78 l}{\log \frac{b}{a}}$$

where b/a gives the ratio of the surfaces of the outer and inner prism. The coefficient 2.78 was experimentally determined. Calculation shows that the coefficient must lie between 2.73 and 3.47 .

Small Cube: Where all the dimensions of a rectangular

parallelepiped are less than one-fifth of the thickness of the insulation, the shape factor should be calculated from:

$$S = 0.79 \frac{\sqrt{AB}}{t}$$

where A and B are the surfaces of the inner and outer boundaries of the insulation and t is the thickness of the insulation. The coefficient 0.79 was found by experiment. Calculation gives 0.724.

Rectangular Parallelepipeds Covered with Uniform Thickness of Insulation: The following formulas apply to all those cases where the furnace is of rectangular shape with uniform wall thickness:

1st Case. All three dimensions are greater than $1/5 t$. For this case the shape factor can be calculated simply:

$$S = A/t + 0.54 \Sigma l + 1.2 t$$

2nd Case. Two dimensions greater than $1/5 t$ and one dimension less than $1/5 t$. We then have:

$$S = A/t + 0.465 \Sigma l + 0.35 t$$

3rd Case. One dimension greater than $1/5 t$ and two dimensions less than $1/5 t$. We have:

$$S = \frac{2.78 l}{\log \frac{b}{a}}$$

as explained under the heading "Small Square Prism."

4th Case. All three dimensions less than $1/5 t$:

$$S = 0.79 \frac{\sqrt{AB}}{t}$$

as explained under the heading "Small Cube."

General Formula for the Cube. For the cube a single formula can be made to cover all four of the above cases:

$$S = \frac{6a^2}{t} + 6.5a + \frac{1.2t}{\sqrt{1 + 0.16 \frac{t^2}{a^2}}}$$

where a is the length of the side of the cube and t is the thickness of the insulation covering it.

Any actual problem may usually be solved by considering the insulation made up of various parts for each of which the shape factor is given in the table. An actual problem will illustrate.

Given a rectangular electrically heated oven, with inside dimensions $8 \times 10 \times 20$ in. The walls are of a uniform thickness of 8 in. This makes the outside dimensions $24 \times 26 \times 36$. The shape factor is calculated from the formula:

$$S = A/t + 0.54 \Sigma l + 0.15 n t$$

Here A is the area of the inside of the wall of the oven. This is 880 sq. in., thus:

$$\begin{aligned} 2 \times 8 \times 10 &= 160 \\ + 2 \times 10 \times 20 &= 400 \\ + 2 \times 20 \times 8 &= 320 \end{aligned}$$

The thickness of insulation is 8 in. Hence, $A/t = 110$. The sum of the lengths of the edges Σl is 152 in., thus:

$$\begin{aligned} 4 \times 8 &= 32 \\ + 4 \times 10 &= 40 \\ + 4 \times 20 &= 80 \end{aligned}$$

$$152 \times 0.54 = 82$$

The third term $0.15 n t$ is 9.6, whence the shape factor is the sum of the three terms:

$$\begin{array}{r} 110 \\ 82 \\ 9.6 \\ \hline 201.6 \end{array}$$

It is interesting to compare this result with that obtained by the usual methods.

The most common method is to calculate the shape factor from the formula $S = A/t$, where A is the arithmetical mean of the inner and outer surfaces of the oven. The outer surface is 4848 sq. in. This would give for the shape factor:

$$S = \frac{808 + 4848}{2 \times 8} = 353.5$$

This is very different from the 201.6 obtained by our new method, and well illustrates the serious error that may be made by using the ordinary rough method.

If the geometric mean be employed instead of the arithmetic, the result is:

$$S = \frac{\sqrt{808 \times 4848}}{8} = 247.4$$

This is much better than 353, but is still quite different from 201.6, being 23 per cent too high.

The paper was discussed at some length by Messrs. C. W. Comstock, C. A. Hensen, F. A. Lidbury and J. W. Richards. A communicated discussion by Dr. E. F. Northrup was read in which a high compliment was paid to the scholarly treatment of the subject by the authors in developing their "sound theory which henceforth must form the starting point in any practical calculation or scientific tests in the difficult subject of heat flow." Dr. Northrup points out, however, that in the calculation of the heat flow from actual furnaces there are other factors, such as surface resistances, variable and uncertain conductivities, departures from the state of temperature equilibrium, heat losses by the outflow of hot gases, etc., which have such large relative importance that it is immaterial that the shape factor be known to a close precision. Dr. Northrup also gave some formulas which are useful when it is required to determine the temperature of any point in the insulation, the temperature of two other points being known. They relate to the cases of parallel planes, concentric spheres and concentric cylinders.

Heat Resistivity of Graphite and Carbon

A paper by Professor Jos. W. Richards, of Lehigh University, on the heat resistivity of carbon and graphite, gave the results of an experimental investigation carried out in the following way: Two blocks each $20 \times 20 \times 10$ cm. ($8 \times 8 \times 4$ in.) were placed on each other so as to form a cube 20 cm. (8 in.) on the side. In the center a cubical cavity was made 2.5 cm. (1 in.) on the side. Two passages 1 cm. (0.4 in.) in diameter were then made from the outside to the inner cavity, and a smaller hole 5 mm. (0.2 in.) in diameter was drilled from the middle of an outside face to within 1 mm (0.4 in.) of the inner cavity. The larger passages were for the introduction of electric-light carbons, passing through glass tubes which fitted tightly into the passages; the small hole was for a thermo-couple (iron-nichrome) for measuring the temperatures when thermal equilibrium was reached. Currents up to 1000 watts were used in the arc.

From several careful determinations the following figures for thermal resistivity (in thermal ohms, i. e., degrees per watt) of these two materials were obtained:

National Carbon Company carbon (202° - 853°) = 30.8 thermal ohms for a cm. cube.

Acheson graphite (240° - 420°) = 3.60 thermal ohms for a cm. cube.

Ratio = 10:1.

From a discussion of the possible errors it is concluded that the ratio in question is at least 10 to 1, but possibly higher, though the ratio is believed to be accurate within 10 per cent.

In the conclusion of the paper a comparison is given with the results of other investigators.

Dr. Richard's paper was discussed at some length by Messrs. C. G. Fink, C. A. Hansen, F. A. Lidbury, J. W. Richards and E. F. Roerber. Dr. Carl Hering, in a long communicated discussion, questioned the accuracy of the experimental method in various respects and protested against the comparison of the author's results with his own determination of "thermal conductivities" under electrode conditions, which are really of different quantity.

Electrolysis of Cyanide Solution

The paper on this subject by Dr. Edward F. Kern, of Columbia University, gives the results of an investigation undertaken with the idea of determining the validity of some of the conflicting statements which were found in reviewing the

extended literature on the electrolysis of cyanide solutions.

A review of the literature on the subject is first given and the published information is summed up as follows:

Electrolysis has, under favorable conditions, been utilized as a means of precipitating and recovering gold and silver from cyanide leach solutions by electrolyzing the solution with suitable electrodes and with current density of 0.01 to 0.6 amp per sq. ft. (0.11 to 6.5 amp per sq. m.). The metal was deposited as a dense film when the current density was properly regulated with regard to the composition, the concentration and the circulation of the solution. The deposit formed is pulverulent and non-adherent if the solution is dilute, the circulation sluggish, and the current density high.

Iron anodes dissolve in cyanide electrolytes causing consumption of cyanide and contamination of the solution by precipitates of iron compounds.

Carbon and graphite anodes are satisfactory in cyanide electrolytes when used at low current density, but they disintegrate at high current density, causing "fouling" of the solution.

Peroxidized lead anodes are not dissolved during electrolysis of cyanide solutions, and the consumption of cyanide by their use is less than when metallic iron anodes are employed.

Fused iron oxide anodes are permanent in cyanide electrolytes, and the cyanide consumption by their use is small.

Oxygen, supplied to cyanide leach solutions either by additions of chemicals or by electrolysis, is beneficial for the treatment of certain refractory ores, in that the extraction is thereby accomplished more rapidly and more efficiently. Electrolysis is claimed to be a cheaper and more efficient method of supplying oxygen to cyanide leach solutions than by the addition of chemical oxidizers.

Regeneration of cyanide "work" solutions, which contain cyanogen compounds, is possible by electrolysis. There is controversy as to whether alternating current can be used for this purpose.

Electrolysis of solutions containing sulphocyanide and cyanamide by means of direct current at high current density (50 amperes per square foot, 540 amperes per square meter) produced solutions of cyanide. Fused iron oxide anodes were found to be the most satisfactory for this purpose.

Electrolysis of solutions containing only cyanide produces cyanates, which will not dissolve gold and silver.

Telluride gold ores were effectively treated by use of solutions which contained an halogen-cyanide compound, which was generated by electrolysis of solutions containing an halide salt and cyanogen compounds, employing direct current at high current density for producing the active halogen-cyanide compound.

An account of the experimental investigation carried out at Columbia University is given in the second part of the paper under the following three headings: permanence of anodes in cyanide electrolytes; electrolysis of cyanide solutions with insoluble anodes; electrolysis of cyanide solutions containing sulphocyanide and ferrocyanide. The results are summed up as follows:

The electrolysis of cyanide solutions by direct current and by means of insoluble anodes is accompanied by progressive consumption of cyanide, which is the result of oxidation of the cyanide to oxy-cyanide compounds.

Anodes of metallic iron, nickel, and lead are dissolved during the electrolysis of cyanide solutions; these metals on entering the solution are immediately precipitated, the lead as hydroxide, and the iron and nickel as a mixture of hydroxide and cyanogen compounds. The consumption of cyanide by means of these anodes was relatively greater the lower the current density, as at high current density oxygen was simultaneously liberated, and consequently less metal was dissolved. The consumption of cyanide was less in the case of the lead anodes than with either the iron or the nickel.

Peroxidized lead and "passive" iron anodes were found to be more permanent than metallic anodes of either iron, nickel or lead. There was no apparent corrosion of either the peroxidized lead or the "passive" iron anodes so long as they kept

continuously in use, but exposure to the atmosphere for several days destroyed their permanency, due to spontaneous oxidation. The "passive" iron anodes were found to be more satisfactory than peroxidized lead anodes for electrolyzing cyanide solutions in that the cyanide consumption was much less and the voltage was slightly less in most cases.

Electrolysis of cyanide solutions, which were used for leaching a special refractory gold and silver ore containing sulphide minerals, was found not to be beneficial either in reducing the cyanide consumption or increasing the percentage extraction.

Low current density at both the anode and the cathode gave less consumption of cyanide, and in several cases seemed to have produced a slight amount of "active" solvents, but not enough to compensate for the loss of cyanide which resulted by the electrolysis.

The cyanide consumption during electrolysis of cyanide solutions was greater when the current density at the cathode was run higher than at the anode, which suggested that the loss of cyanide is the result of oxidation, with the formation of oxy-cyanogen compounds. The lower the current density at the anode and the higher it is at the cathode, relatively greater is the cyanide consumption.

When cyanide solutions containing sulphocyanides or ferrocyanides are electrolyzed, the cyanide consumption was much less than that which occurred in pure cyanide solutions, which indicates that sulphocyanide and ferrocyanide act as protective agents during the electrolysis of cyanide solutions.

The increased alkalinity of cyanide solutions reduced the cyanide consumption during electrolysis, due to increasing the conductivity of the solution. The higher the voltage the greater the cyanide consumption, and *vice versa*.

The electrolysis of cyanide leach solutions had no apparent effect upon reducing the time required for treating a refractory ore nor in increasing the percentage extraction.

Peroxidized lead anodes and "passive" iron anodes are corroded in cyanide electrolytes which contain sulphocyanide, if the electrolysis is conducted at high current density. No apparent corrosion occurred when a low current density (below 3 amp. per sq. ft., or 32 amp. per sq. m.) was used.

"Passive" iron anodes are more satisfactory than peroxidized lead anodes for the electrolysis of cyanide solutions which contain either sulphocyanides or ferrocyanides in that by their use the cyanide consumption is very much less and the voltage slightly less. The lower the current density the smaller the consumption.

The regeneration of cyanide solutions which contain sulphocyanide and ferrocyanide does not occur by electrolysis with direct current, whether the conditions of electrolysis be made oxidizing or reducing by varying the relative current densities at the anode and the cathode.

Dr. Kern's paper elicited considerable discussion, in which Messrs. G. H. Clevenger, F. C. Frary, F. A. Liddy, J. W. Richards, and E. P. Schoch participated.

Professor Schoch thought well of the fundamental idea of the Clancy patents.

Professor Clevenger gave an interesting account of his experience in a Butters cyanide mill. They had tried everything in the way of "insoluble" anodes, but passive iron. They found that peroxidized lead anodes had a life of about a year. He thought that the use of passive iron was an interesting and good suggestion. In the whole Professor Clevenger agreed with the conclusions of Dr. Kern's paper, though he thought they were based in part on insufficient evidence.

Dr. Richards described a mercury cathode cell of the Castner type with which he had made some laboratory experiments. It is a double-compartment cell, the two compartments being separated from each other by a baffle in the center of the cell. This baffle dips into the layer of mercury provided on the bottom of both compartments. The anode compartment contains a solution of caustic soda, sodium being deposited onto the mercury and the sodium amalgam being transferred to the other (the cathode) compartment. The latter contains gold sodium cyanide solution. Gold is deposited

and the cyanide solution is regenerated from the amalgam.

Mr. Lidbury remarked there would be an accumulation of caustic, but this would be an asset. Mr. Clevenger spoke of a similar scheme which was to be tried on a large scale, but as far as he knew nothing had come of it. He thought the scheme was not impossible.

Session in Boulder

The second technical session, held on Wednesday morning in the University of Colorado, in Boulder, was taken up by the reading and discussion of two papers. The paper by Clevenger and Hall on electrolysis of cyanide solutions was presented by Professor Clevenger; the paper by Lyon and Keeney on possible applications of the electric furnace to Western metallurgy was presented by Mr. Lyon. Both papers elicited a very animated and extended discussion, lasting in each case for almost an hour. As matter of fact, the discussion would have been continued further, if the announcement had not to be made that in spite of the heated discussion the lunch was getting cold.

Electrolysis of Aqueous Solutions of the Simple Alkaline Cyanides

The paper on this subject by Professor G. Howell Clevenger, of Leland Stanford University, and Mr. Mortimer L. Hall, of La Touche, Alaska, gives the results of an experimental investigation in which chemically pure potassium cyanide was used throughout. For details of the various series of experiments the reader must be referred to the full paper which also contains interesting notes on methods of analysis for the electrolyzed solution and a summary of the reactions (in part hypothetical) occurring during and after electrolysis.

The authors' conclusions are as follows:

The bulk of the decomposition of cyanide occurring during electrolysis is due to oxygen liberated at the anode through the decomposition of water.

The principal final reaction in solutions of cyanide containing protective alkalinity, which in cyanide mill practice is usually due to lime, is the formation of the corresponding alkali carbonate. This accounts for the formation of large amounts of calcium carbonate during the electrolytic precipitation of gold and silver from cyanide solutions.

A consideration of the reaction previously discussed will explain the failure in many cases to obtain regeneration of cyanide when such regeneration is theoretically possible. It is apparent that the decomposition of cyanide may easily be equal to or greater than that regenerated through the precipitation of the metal in solution.

The Liebig titration for the estimation of cyanide in an electrolyzed solution gives results which are low.

The changes taking place in fresh and electrolyzed cyanide solutions, upon standing, are different.

While not wishing to discourage investigators working along the lines of aiding extraction with refractory ores by electrolysis, we think that it is of importance to call attention to the disadvantages which may arise through the indiscriminate use of electrolysis with an ore pulp. First, a large loss of cyanide may result from which there is no benefit derived, and, second, electrolysis may actually interfere with extraction through the formation of a coating of calcium carbonate upon the ore particles.

The discussion was opened by Mr. Lidbury who in commenting on Professor Clevenger's paper and on Professor Kern's paper of the day before, thought that not enough attention was paid to a fundamentally important principle, often emphasized by Professor Bancroft, namely, that all electrolytic processes should be considered first of all as chemical reactions. We should endeavor to disclose the pure chemistry of the reaction. What is electrolysis called upon to do? What are the chemical conditions and the chemical reactions which we desire to produce? What are the anode reaction and the cathode reaction, if strictly separated and investigated separately? The investigation should be conducted

with exact observance of conditions, such as steady temperature, steady potential drop, etc. The evolved gases should be investigated, etc.

Professor Clevenger replied that his investigation had been purposely carried out just in the same way as the work under discussion had been done by the various practical experimenters all over the country. Temperature cannot be maintained constant in commercial mill practice. But he agreed that Mr. Lidbury was right about the way that future investigations must be carried out.

Professor Richards said that practical cyanide men were well acquainted with many of the chief chemical problems of their work, like the action of different solvents, etc. One chief difficulty of the application of electrolysis to cyanide solutions was found in the fact that for the electrolytic deposition of the gold only 2 per cent of the current was usefully employed, and 98 per cent was wasted and great care had to be taken to make this 98 per cent of current harmless so as not to destroy the cyanide. As to the life of lead peroxide anodes in cyanide solutions he said that if lead anodes were first coated with peroxide in an alkaline solution they would be found far more resistant than if oxidized in an acid solution.

Professor Frary questioned some of the hypothetical reactions in the paper.

Professor Schoch emphasized the wonderful opportunities in bringing in a meeting like this the practical and the so-called theoretical men together. As to the necessity of separating the anodic from the cathodic reaction, this was most important in the first teaching of electrolysis, though many widely used text books were shocking in overlooking this fact. Mr. Lidbury's criticism should make professors using such text books blush.

Mr. Skirrow thought that in a careful investigation conditions of temperature, current density, etc., should be varied one at a time. It was clearly impossible to study everything at once.

Possible Applications of the Electric Furnace to Western Metallurgy

The paper on this subject, by Dorsey A. Lyon and Robert M. Keeney, of the U. S. Bureau of Mines, first gives a review of the present status of the electric furnace in *smelting iron ores*. The fact that coke is unsatisfactory as a reducing agent in the electric iron ore reduction furnace (chiefly on account of the much better electric conductivity of coke as compared with charcoal) seems to limit the electric iron reduction furnace to the use of charcoal. Hence it is necessary that such a furnace be located in close proximity to a well timbered region, where charcoal as well as electric power can be produced at a low cost. This is unfortunate, especially as regards the situation in southern California and in other of our western States, where there are quite large iron deposits of A-No. 1 ore, but where coke is too expensive to permit of ordinary blast furnace smelting, and where the use of charcoal for electric furnace work is entirely out of the question, as there are no forests to furnish the wood necessary for its production. On the other hand, crude oil is generally more or less plentiful and comparatively cheap in such districts, especially in California.

As a result of a preliminary investigation conducted by the metallurgical section of the U. S. Bureau of Mines on the possibility of using crude oil as a reducing agent, it seems as if about the only manner in which crude oil may be used for this purpose is to first convert it into a fixed gas and then introduce this into the crucible of an electric furnace, thus preheating the gas to such a temperature that it will effectively reduce the iron oxides as it passes up through the shaft of the furnace.

A review is given of the commercial development of electric iron smelting in Scandinavia and California. From the Scandinavian reports it is shown how step by step the results have been improved. The output has been increased from 3.10

to 3.20 tons of iron per h. p. year. The electrode consumption has been reduced to 3 k. g. (6.6 lb.) per ton of pig iron. The cost and time required for repairs were items which could not be estimated beforehand. Experience has shown that both the cost and the time required are less than could have been hoped for. Wherever employment for the gas can be found, this should also be taken into account, as is being done at Hagfors, where its value for firing the open-hearth furnaces is estimated to reduce the cost of the pig iron by about 66 cents.

With respect to *copper smelting* in the electric furnace, reference is made to the same authors' recent Am. Inst. Min. Eng. paper on this subject (see our September issue, p. 522).

The smelting of straight *lead ores* in the electric furnace has never been attempted either commercially or on a large experimental scale, largely because of the ease and cheapness of smelting by combustion processes. For the smelting of ordinary lead ores it has no special application, but in the treatment of complex lead-zinc-silver ores the electric furnace, might be profitably used. The roasted ore could be treated in an electric furnace operated at a temperature such that the lead reduced by carbon melts and collects the precious metal values and the zinc is volatilized and condensed.

The use of the electric furnace in the metallurgy of non-ferrous metals has had greater application for the treatment of *zinc ores* than in the metallurgy of any other of the non-ferrous metals except aluminium and ferro-alloy manufacture. Since 1885, when an electric furnace for the treatment of zinc ores was patented by the Cowles Bros., experimental work has been conducted on the subject. The commercial application of the process, however, has not resulted to any great extent because of the difficulty of condensing the zinc vapor produced under the smelting conditions of the electric furnace. The cause of this difficulty has not yet been definitely determined. With few exceptions the experimental work has not been conducted on a very large scale, and so it may be said that the electric smelting of zinc ores is still in the experimental stage.

Undoubtedly the electric furnace has a very promising field in the metallurgy of zinc, but before its wide application can be expected the condensation problem must be solved and mechanical problems in connection with the electric furnace worked out. The latter should not be difficult. The special field for electric furnace work in zinc metallurgy is due largely to the low thermal efficiency of the present zinc retort which is given by Richards as only 7 per cent.

With the efficiency of the electric furnace varying from 50 to 75 per cent, the difference in the heat necessary for carrying out the smelting operations is so great as to permit of a higher cost of electrical energy than is permissible in many electrometallurgical operations.

The authors review the present methods of zinc smelting and compare therewith the electric furnace method of zinc smelting. They summarize the results obtained in commercial operation at Trollhättan, describe the experiments made by A. Stansfield and W. R. Ingalls at McGill University, refer to the experimental work of W. McA. Johnson, discuss the electric smelting of zinc sulphide ores with iron as desulphurizing agent, and then sum up the present status of electric zinc smelting as follows:

"The process is still largely in the experimental stage. There is no plant operating on a commercial scale except the Trollhättan works, taking from 7500 to 10,000 kilowatts. The experimental work mentioned is still going on, but as yet none of it has developed into a commercial operation.

"From what has been said concerning the work at Trollhättan and the results of others, it is very evident, as previously explained, that the difficulty lies almost entirely in condensation of the zinc vapor to a metal rather than blue powder under the peculiar conditions of the electric furnace. The electric furnace, mechanically or electrically, presents no great difficulties to solve, because all the troubles formerly experienced have been solved in the construction of large pig iron, steel, carbide

and ferro-alloy furnaces. The problem, then, is one of a metallurgical nature, and is caused by the different conditions and greater speed of smelting in the electric furnace than in the combustion retort.

"The opponents of the electric smelting of zinc ores, who as a rule appear to have no knowledge of the great development of the electric furnace itself in the treatment of other ores, often state that, while the retort is the weak part of the old system, so is the electrode the weak point of the electric method. Admitting that the retort is the weak part of the retort method, and judging from large scale electrometallurgical work on other ores, we do not believe that in general the electrode is an exceptionally weak part of electric smelting. The electrode problem, no longer a serious one, has been solved in iron smelting. It is only fair to assume that the designer of an electric zinc furnace should stand upon the accumulated knowledge and take the results of others when the metallurgical problems of electric zinc smelting have been solved. One editorial writer in the *Engineering and Mining Journal* of December 14, 1912, goes so far as to say, in reference to retorts and electrodes, 'nobody can say which will be the weaker.'

"While it is not the purpose of the writers to enter into a discussion of the relative advantage of processes, it may be of interest to present a few comparative facts regarding retort and electrode consumption.

"Ingalls' states that the average life of a Silesian retort over a period of ten years is 38 days. One charge is smelted per day. An editorial in the *Engineering and Mining Journal* of December 14, 1912, states that the Silesian retort takes 200 pounds of ore per charge. Hence a retort treats 7,600 pounds or 3.8 tons of ore. Ingalls' also states that a condenser lasts from 8 to 12 days, so that about four condensers are necessary for the ore smelted by a single retort. He also states that a seasoned retort is valued at 50 cents, and a condenser at 4 cents. Thus the cost of a retort and condensers for smelting 3.8 tons of ore is 66 cents, or 17 cents per ton of ore smelted.

"At Trollhättan, Sweden, the electrode consumption has been reduced to 6 pounds (3 kg.) per ton of pig iron produced. Considering that two tons of ore were charged for each ton of pig iron produced, the consumption per ton of ore is 3 pounds. The smelting conditions in electric iron reduction would not tend to be as strongly reducing as in the zinc retort, although the ore in each case would be an oxide. Hence it may be expected that the total consumption of electrodes in a large, properly designed zinc furnace would not exceed 3 pounds per ton of ore.

"There is no longer loss of electrode because of unconsumed ends, as this loss has been entirely overcome by joining both amorphous carbon electrodes and graphitized electrodes. This reduces the loss by one-half.

"In some experiments by the writers on a mixture of sulphide and oxidized copper ores, with no carbon in the charge, and the charge proportioned so as to contain 15 per cent sulphur, the electrode consumption with air blown into the furnace for partial pyritic smelting was only 2 pounds per ton of ore. However, the furnace was designed so that the charge protected the electrode from the air.

"Amorphous carbon electrodes can be obtained at from 3 to 4 cents per pound, so that, on a basis of a consumption of 3 pounds per ton of ore, the electrode cost would be from 9 to 12 cents as compared with a retort cost of 17 cents. These figures seem to show that, rather than 'nobody can say which will be the weaker,' the retort at the present time is the weaker.

"If the failure of the electric furnace in smelting zinc ores is not due to the electric furnace or the electrodes, what then is the metallurgical condition which causes condensation difficulties and has retarded the commercial application of the process? The difficulty is, of course, due to a difference in the smelting conditions of the electric furnace as compared with the retort. Both physical and chemical conditions influence the formation of blue powder. Some powder forms under certain

*W. R. Ingalls, *The Metallurgy of Zinc*, p. 545 and p. 242.

conditions of pressure and temperature. As to just what these conditions are has not yet been solved. It is a problem for the physical chemist.

"The chemical conditions leading to the formation of blue powder are due largely to the speed of reduction of zinc oxide in the electric furnace and to the presence of volatile impurities in the charge. Boudouard found that the permissible percentage of carbon dioxide in the zinc retort was very low, and that when the limit is exceeded the carbon oxidizes the zinc, causing formation of blue powder. The vapor of the zinc retort is usually below this limit or not much above it, so that the zinc is condensed chiefly as liquid zinc. But in the electric furnace the temperature directly beneath the electrode in either the arc or resistance furnace is so high that the reduction proceeds so rapidly that the large amount of carbon dioxide formed does not have a chance to be reduced to monoxide by the hot coal of the charge before entering the condenser. Hence there is the oxidation of the zinc mentioned.

"By the use of electrodes of large cross-section it is probable that this concentrated heating could be reduced so that, rather than a very high temperature at one place, there would be a uniform comparatively low temperature all over the charge. This was found to be the case in the smelting of iron ores, both at Héroult, California, and Trollhättan. Or the current could possibly be distributed more evenly by flat electrodes of rectangular cross-section.

"The formation of blue powder is also caused by volatilization of silicon, which condenses on the zinc either as silicon or silica. For example, in some experiments at the Bureau of Mines Laboratory the condensed fumes from a ferro-silicon furnace contained 67.69 per cent SiO_2 , 9.72 per cent Al_2O_3 , 4.11 per cent FeO , 0.20 per cent MgO , 1.90 per cent CaO , with the rest carbon. This volatilization of silicon occurs in any electric furnace operated with an arc, as in ferro-silicon manufacture, when there is silica in the charge. At the lower temperature of a resistance furnace it is not so likely to occur.

"The solution of this problem is very probable, and, while it is difficult, there is no reason why it should not be worked out in time. When this is solved, and it is not necessary to resmelt a large proportion of blue powder as at Trollhättan, where two tons of blue powder are smelted for each ton of ore treated, it is probable that electric zinc smelting will proceed rapidly. The use of iron as a desulphurizing agent does not seem to have advanced as far as reduction of the oxide with carbon, and it is probable that the latter will keep its present supremacy. Some work has been done in an electric furnace operated under pressure or vacuum, but only a beginning has been made in this direction."

As to the treatment of *complex sulphide ores*, the authors speak as follows: The electric furnace has a possible field in Western metallurgy at places where there are complex sulphide ores of lead, zinc, copper, gold and silver. With the solution of the zinc condensation problem it is not unreasonable to suppose that the ores of this class can be electrically smelted so as to get a fair recovery of the metals and at least the precious metal values. Also, owing to the fact that practically as high a temperature as desired may be obtained in the electric furnace, it would be possible to smelt complex ores which have high gold and silver values, but which are too low in base metals, such as lead and zinc, to make their recovery worth the additional expense of concentration.

In the blast furnace 10 per cent of zinc in the charge causes sticky slags, and with a much higher percentage the operation of the furnace is very irregular and difficult. These slags could be fused readily in the electric furnace. In such a case, if the ores contained copper and zinc, the ore could be charged into an electric blast furnace using matte as a collecting agent for the gold and silver.

In the smelting of *gold-silver ores* carrying no lead or copper, to form either a lead bullion or a copper matte, but containing iron sulphide, the electric furnace might be used in connection with an air blast for oxidation of the iron sulphide, and an iron matte as a collecting agent for the gold and silver. The value

of iron matte as a collecting agent is still largely a matter of personal opinion, but from the pyritic smelting which has been done with iron as a collecting agent, and from some experiments of the writers, using iron matte in the electric furnace, it is evident that a fair recovery can be made in this way. As in the case of electric copper smelting, the electric furnace treatment of these ores is still in the speculative stage.

Mines in isolated districts, where fuel is usually high and where the power operating the mine or mill is often hydro-electric power, could in many cases melt their cyanide precipitates more economically in electric furnaces than in combustion furnaces (as is done in Chihuahua, Mexico). The construction of a small Siemens type of furnace is simple and inexpensive. By operating the electric furnace when one of the units of the mill was down for repairs, or on the mine shift when little hoisting or drilling was being done, no additional electric installation would be necessary. This simply means that power which under ordinary conditions is wasted would thus be utilized.

On the treatment of *chromium, tungsten, molybdenum, and vanadium concentrates*, the authors speak as follows: The treatment of these ores is of particular interest to the Western metallurgist, for all of these ores are mined in the Western States, but their treatment has been confined to concentration by ore dressing or chemical methods, or shipment of their concentrates to the ferro-alloy manufacturer or the metal refiners. If these concentrates could be treated locally they would be given a "form value" at once, and thus enable the district in which they are produced to retain money which is now expended for freight charges, and also to retain the profits of the ferro-alloy manufacturer, which, because of the specialty of the product, are rather large.

For this work relatively high temperatures are required, so that the operation could not be performed metallurgically or economically in a blast furnace, reverberatory furnace or crucible. Tungsten is reduced from the trioxide at a temperature of 1300°C ., and vanadium from the vanadate of iron or calcium at about the same temperature. The melting point of tungsten is 2800°C ., molybdenum 2000°C ., and vanadium 1750°C . All of these temperatures are high compared with that for the reactions usually performed in combustion furnaces. Of course, the melting point of the ferro-alloy would not be as high as that of the pure metal, and becomes lower with increased percentage of iron. The electric furnace is especially fitted for the smelting of rare metal ores to produce ferro-alloys because of the ease with which high temperatures were maintained.

The present status of the manufacture of *ferro-alloys* in the electric furnace is briefly considered, together with the reasons why the growth of this industry has been more slow in America than in Europe. The general construction of electric ferro-alloy furnaces is discussed as follows:

In general, an electric furnace for ferro-alloy manufacture consists of a crucible without a roof, having either a conducting or a non-conducting hearth and an upper vertical carbon electrode or electrodes. Most of the furnaces first built had a conducting hearth, and this practice is still common in furnaces up to 750 kilowatts capacity, but for larger sizes experience has shown the non-conducting hearth type to be preferable. One reason for this is because there is considerable loss of electric energy by dissipation through the lining, and hence there is a tendency to have a lower power factor. In working a small furnace intermittently, i. e., discharging completely before charging again, the conducting hearth furnace is much easier to operate and start cold. When a large furnace is to be operated for several months continuously on one alloy, this advantage of the conducting hearth is of no importance. A factor which has been little considered in the construction of ferro-alloy furnaces is the power factor, as they are generally completely enclosed in sheet steel, when hands would do as well, with much less induction resulting in the circuit.

The ferrosilicon furnaces are lined with carbon, while the furnaces to be used in producing other alloys are generally

lined with a basic lining of magnesite or dolomite brick or a mixture of tar with either material. When low carbon ferrochrome is being made the lining is sometimes chromite. Generally a furnace is designed so that the electrode or electrodes are at a sufficient distance from the sides to permit a chilling of the material charged upon the walls. This has been found to give the most satisfactory lining. Under such conditions furnaces have been operated for two or three years without being relined. The average furnace operated continuously runs about two years before being shut down for repairs. A furnace operated intermittently must, of course, be relined more often.

Owing to the greater part of the reduction of oxides in ferro-alloy manufacture being done at high temperature by solid carbon, most of the furnaces are run with an open top, into which the charge is placed around the electrode. Under these circumstances there is a considerable waste of heat which might be used in preheating the charge. In these furnaces the electrode usually dips about 12 to 15 inches beneath the top of the furnace, which is kept filled with cold material so as not to expose the molten mass beneath. There is burning of carbon monoxide around the electrode, and an escape of a large volume of gas through the top of the furnace house. It is stated that one of the Alby type of carbide furnaces is operating with a closed top at Kopporaen, Norway. The Helfenstein furnace does away with the open top also, as it is air tight, and in addition has feeding shafts where the charge is preheated by the waste gases. In the manufacture of the higher-priced alloys, such as ferrotungsten, it is impossible to use any furnace but one with an open top, because of the intermittent nature of the operation, but in making ferrosilicon and ferrochrome it seems advisable to preheat the charge as much as possible and prevent as much oxidation of the electrodes as is possible in this manner.

A large number of the first ferro-alloy furnaces were supplied with direct current when the furnace house was built adjoining the power house, so that there was only about 20 feet between the furnace and the generator. It was soon found that the electrolytic action in a direct-current furnace resulted in deposition of potassium, sodium, calcium, aluminium with the alloy so that as pure a product could not be obtained with direct as with alternating current. When alternating current was first used, single-phase current was the more common in general application, so that the arrangement remained the same as previously, the generator delivering directly to the furnace at the desired voltage without any transformation.

Later, when three-phase current became prevalent, furnaces designed to take single-phase current were connected in groups to three-phase systems by several methods. Then the single phase furnaces were replaced for large scale work by the present-day three-phase furnace.

The cost of electric power for electric furnace is finally considered and figures are given for the production of electric power with gas engines and steam turbines.

"As to what the cost of power produced by gas engines would be, a reliable engineering concern has furnished data which show that, with coal at \$3.00 per ton, which it is assumed will contain 13,000 b.t.u. per pound, electric power can be produced at about the following cost when using gas-engine-driven generators at a load factor of 75 per cent:

Size of Plant, Kilowatts	Cost of Power in Cents per Kilowatt Hour.	
	Coal at \$3.00	
2,000	0.745	
4,000	0.687	
6,000	0.649	
8,000	0.637	
10,000	0.633	

"These figures include operating costs and fixed charges

"The same engineering concern has also supplied the writers with data in regard to the cost of electric power produced by steam-turbine-driven generators. Based upon this data, the cost of electric power is shown in the following table, with coal at \$3.00 per ton. These costs include operating costs and fixed

charges, and, as before, it is assumed that the coal has a heating value of 13,000 b.t.u. per pound and that the load factor is 75 per cent."

Size of Plant, Kilowatts	Cost of Power in Cents per Kilowatt Hour,	
	Coal at \$3.00	
2,000	0.688	
4,000	0.597	
6,000	0.532	
8,000	0.526	
10,000	0.514	

A table is finally given of the approximate cost of power at various electrochemical plants in various countries. The prices range all the way from \$5 per kilowatt year (Rjukan, Norway, for nitrate manufacture) to \$110 per year (Baintree, England, for electric steel making).

The paper elicited a spirited discussion.

In a communicated discussion by Mr. F. A. J. Fitzgerald, the scope of the work of the U. S. Bureau of Mines was criticized.

Mr. Lyon, in reply, stated that the work of the Bureau was to ascertain facts and make the public acquainted with the facts. He welcomed constructive criticism of the work of the Bureau of Mines by the American Electrochemical Society. A committee specially appointed for this purpose could render useful service.

Professor Richards stated, unofficially, that Director Holmes was desirous of having such a committee appointed in order that his Bureau might work in full harmony with the national societies in investigating questions of national concern in the various fields of science coming within the purview of the Bureau. Professor Richards moved "that the Board of Directors be requested to appoint a conference committee authorized to confer with the Director of the Bureau of Mines regarding the activity of the Bureau in electrochemical and electrometallurgical matters."

Mr. F. A. Lidbury seconded this motion, which was then unanimously carried.

Dr. Richards, in discussing the paper, said that electric zinc smelting was no longer on the experimental scale. There was a new plant now in operation at Helsingfors, in Finland. At present 2500 h. p. are used, but the plant is to be enlarged to 6000 h. p. But the United States is behind Europe in this development. As to the idea of some electric zinc men that electric zinc smelting was chiefly an engineering problem, this was quite erroneous; the chemical and metallurgical problems must remain in the foreground.

Mr. C. A. Hansen discussed, at some length the possibilities of electrochemical methods in the West. He also emphasized the excellent prospect of electric steel casting plants. He gave a set of very interesting figures, favorable to the electric furnace, in comparison with the acid open-hearth, the Tropenas converter, and the crucible. As these figures, based on the experience of several years in large-scale operation, were given by Mr. Hansen from memory, we reserve their publication for a future issue, when the figures will have been verified by Mr. Hansen and given out authoritatively.

Dr. John A. Matthews, of the Halcob Steel Co., said that while in Scandinavia and California the electric iron ore reduction furnace had made rapid and important progress, yet in the Eastern States of this country the possibilities of the electric furnace in the iron and steel industry were chiefly in three directions: first, for the production of high-grade steel; second, in combination with the Bessemer converter for large tonnage products; third, for steel castings. This third field was probably very important.

Mr. E. L. Crosby of the Detroit Edison Company spoke of possibilities of electric furnace loads for central stations.

Mr. L. B. Skinner emphasized the comparatively high cost of electric power in the West. In Denver no power could be had at such a price as 1 cent per kw. hour. This fact had to be considered in connection with electrochemical schemes in the West. Further, the cost of labor is high. He referred to an extended trip of a distinguished Western metallurgist to Europe

to study the status of electric zinc smelting there not so long ago; he had not found that the problem had been solved commercially.

Owing to the lateness of the hour the discussion had to be interrupted. It was, however, taken up again the next day in connection with the two electric zinc papers.

Session in Golden

The third technical session was held in the School of Mines at Golden in the afternoon of Thursday and was probably the best attended session, there being nearly too members and guests present.

Mr. McGregor presented his paper on electrostatic concentration and Dr. Lind his paper on the transformation of radiant into chemical energy. The two electric zinc papers, one by Mr. Johnson, the other by Mr. Petersen, were next presented and discussed at considerable length. On account of the lateness of the hour all the other papers were then read by title and briefly discussed.

Progress in Electrostatic Ore Dressing

The paper by Mr. Frank S. MacGregor on this subject attributes the continued success of the electrostatic concentration process (Huff process) to three factors:

(1) Use of electricity produced by commercial electro-magnetic generators and transformers, whose output and potential is steady and capable of easy regulation in the place of the

characteristic which enables the separation just described to be made, but, if the crystal be immersed for a short time in a very weak solution of copper sulphate, it becomes coated with a film of copper sulphide. This is an excellent conductor of electricity (speaking electrostatically) and if the particle is dried at a moderate heat, it behaves as a conductor, and is repelled by the electric charge in the same manner as pyrite, chalcopyrite, graphite or other conducting minerals. Separations of zinc sulphide from gangue have been made in this way on a laboratory scale for some time, but the method has been utilized commercially for the first time in the Ouray plant.

The treatment is very simple: the zinc product is run into tanks, covered with a 0.5 per cent solution of copper sulphate, allowed to stand for a short time, the solution then drawn off for further use, the ore then dried and on passing through the electrostatic separators a 40 per cent zinc product is raised to 51 per cent with only 5 per cent zinc remaining in the gangue product, making a recovery of 97 per cent. The consumption of copper sulphate is very small, a new solution treating several lots.

Besides the zinc field other fields are now being entered by electrostatic concentration. Excellent work in the concentration of crude silver ores has been done in a 30-ton mill in Austin, Nevada.

In Australia a Huff plant concentrates molybdenite from its gangue.

Two graphite mills now employ this method for removing the mica from the graphite flake, the solution of a very vexing problem. It is not improbable that the entire concentration of some graphite will be done electrostatically.

A recent development of great importance, both to the process and to the industry it affects, if the present indications are borne out, is for the cleaning of coal, both anthracite and bituminous. With anthracite occurs slate and other impurities which in the finer sizes are very difficult of separation. Electrostatic separation has offered a solution for this problem, reducing the ash in the "culm" from 25 to 30 per cent to 10 or 12 per cent.

In bituminous coal, the problem consists in the removal of ash and sulphur. Sometimes most of the sulphur present is "organic," so associated with the carbon that it cannot be mechanically removed, but when the sulphur is present as "pyrite" or in "bone," it is readily repelled, and with its extraction comes a large reduction of the ash. Activities are just entering this field.

A typical mill flow sheet for an electrostatic plant consists of a cylindrical dryer, about 4 ft. x 20 ft., made of sheet iron; bucket elevator to the top of the mill, carrying the dried ore to the screens. In some cases only a "scrap" screen is needed if the sizing required is 20 to 50 and through 50 mesh. If the various minerals will break mechanically free above 20 mesh, one or two coarse screens are necessary, as 6 to 12, 12 to 20 mesh. Then the ore is taken to the separators, through which it falls by gravity, the conductors and non-conductors as separated passing directly to storage bins. Returning a small amount of middlings from each separator back into the system is beneficial in increasing the tonnage and making cleaner products. These middlings finally pass out of the system in the finished products.

The power required is small, a 3-hp generating outfit being sufficient to electrify a plant of from one to fifty separa-



ON TOP OF PIKE'S PEAK

very erratic frictional static machines which were formerly tried.

(2) Use of concentrated and powerful electrostatic field by means of two adjacent electrodes, giving steadiness and strength to the action.

(3) Use of a grounded machine constructed almost entirely of metal, and therefore free from electrical disturbances.

A review is given of the use of this process and the results obtained at the following plants: U. S. Smelting, Refining & Mining Co., at Midvale, Utah; Calumet and Sonora Mining Co., at Cananea, Mexico; Sunnyside Mines in Eureka, Colorado; Mary Murphy Mine at St. Elmo, Colorado, and a custom zinc plant at Ouray, Col.

Zinc smelters pay a much better price for a high-grade zinc concentrate than for a lower one, even if the impurity is only gangue, and it is not always feasible to remove all this gangue on the ordinary wet tables without entailing too great a loss of zinc. At the latter plant an interesting electrochemical action has been utilized to raise the percentage of zinc in the zinc product.

Ordinarily a crystal of zinc sulphide is a non-conductor, a

tors. The machinery of one plant is operated by 12 actual hp, which includes power for the dryer, elevator, screens and six separators. One attendant will care for a shift, exclusive of handling ore.

In conclusion the author said that while the installation of electrostatic separators has not been as rapid as the exploitation of some other metallurgical processes, it has developed a unique field of its own, to which no other process is as well adapted, and the tonnage treated electrostatically is steadily increasing.

The paper was discussed by Mr. G. E. Collins, who gave interesting details of the result obtained with electrostatic concentration at one of his mines and compared it with electromagnetic concentration.

The Transformation of Radiant Into Chemical Energy.

A paper by Dr. S. C. Lind, of the Bureau of Mines, gave a summary of a continuation of his theoretical and experimental researches in this field. The results are summed up as follows:

Much experimental evidence is now at hand in support of the principle of electrochemical equivalence in gas reactions taking place under external ionizing agencies.

The use of the term "applicability of Faraday's law to gas reactions" may lead to confusion, especially in the case of reactions produced by electrical discharge, in which the measured current does not correspond directly to the quantity of chemical change. The term ionic-chemical equivalence in gas reactions seems preferable.

The theoretical maximum amount of ozone producible per hour by one gram of radium in equilibrium is estimated to be about 0.72 gm. Warburg's premises by means of which he estimated a lower value are discussed and the sources of error pointed out.

A method is shown by which the coefficient of the transformation of (ionizing) radiant energy into chemical energy may be calculated from the N/M ratio (where N is the actual number of pairs of ions produced and M the number of molecules entering into reaction).

For several *endothermic* reactions the transformation factor is shown to have the low average value of about 2 per cent, the same as has been found for some reactions produced under other forms of radiant energy.

The ratio of chemical to radiant energy in *exothermic* reactions is of the same low order, indicating that the influence of the free chemical energy in these reactions when produced by ionizing agencies, is very subordinate in importance, if it exists at all.

The paper was discussed by Professors Frary and Schoell.

The Art of Electric Zinc Smelting

Mr. Woolsey McA. Johnson's papers on this subject begins with a frank statement that "strictly speaking, there is no art of electric zinc smelting, for the commercial trials in this country, Canada and abroad have been clever, but are not art, since none of them are pronounced commercial successes.

"Our ideas and practical embodiment thereof are radically different from previous commercial attempts. We are not, however, claiming anything more than our personal faith in our ability to operate our process on zinc-lead-copper ores, recovering spelter, base-lead bullion and copper matte in one operation at a probable handsome profit under certain conditions. It is evident that the only criterion of a successful new process is the actual production of finished product at a profit for the market under actual commercial conditions."

The author quotes from an old report of his own, made in 1904, that the weak points of the present method of treating zinc ores are high labor charge per ton of ore, high fuel cost per ton of ore, and low recovery from even high-grade ores with restrictions as to nature of ores, while the advantages of a commercial electric zinc furnace would be large units, higher thermal efficiency, ease of control over smelting and production of pure spelter from impure ores.

Mr. Johnson then points out some of the inherent difficulties

of the problem. "... This very friendship of electricity and zinc is seductive, for any experimenter with 20 kw at his disposal, a few firebrick, some zinc ore and a couple of carbon electrodes cannot help making some zinc, only a few pounds, to be sure, but enough to allure him to further pursuit. And the very contrariness of the metal that seems to have so much free-will is that if one tries to make 100 per cent of zinc vapor as commercial zinc-dust, one will fail. For zinc is a peculiar metal.

"All our attempts up to 1909 were concentrated on the condensation of zinc vapor to spelter. We were correct in the statement, I believe, that we knew seven or eight years ago that the main conditions for good condensation were as follows:

"1. A regular flow of pure gas.

"2. Space at right temperature.

"3. Sufficient time."

The author discusses the production of blue powder instead of liquid smelter and distinguishes between "chemical" and "physical" blue powder (see this journal, Vol. 10, p. 451) and says that there are a number of conditions which produce chemical blue powder and that all but three of them can be easily prevented by foresight, provided their causes are known. In one of the tables in the paper giving the results of some of his last year's work it is seen that on complex ores, high in FeO and analyzing 4 per cent S the author has exceeded the condensation factor of 50 to 70 per cent, obtained in retort practice on simple ores well roasted.

The author sums up the requirements of successful electric zinc smelting as follows:

1. A first-rate preheater, mechanically strong and machine operated. Preheating *per se* has no advantage save to reduce kw hours per ton of ore. Preheating must be done right, not too hot, not too fast, not too slow, and not too cold, with a progressive application of metallurgical means.

2. The flow of heat through the charge in the electric smelting furnace and slag must be under perfect control. The electricity must be controlled, and care must be taken that the zinc globules in eductor assume no statical charge, otherwise coalescence of the spherules is hindered.

3. A regular flow of pure zinc gas must be delivered at proper temperature to the condenser.

4. The condenser must be durable and under good metallurgical control.

When all these conditions are fulfilled, spelter will come as a result of continuous electric zinc smelting in surprising amounts.

"The present Johnson zinc smelting process depends commercially upon the by-products. Furthermore, there is no doubt that our recoveries of lead, copper, gold and silver will be much higher than those of a lead furnace or of a copper furnace.

"Our general proposal is to treat low-grade ores high in by-product values. While we have treated oxidized ores, as Joplin 70 per cent zinc or even oxide of zinc, at the plant of the Continuous Zinc Furnace Company, we prefer to use low-grade ores analyzing about 35 per cent zinc.

"It seems distinctly probable that the cost of treatment of the zinc ores in the Johnson electric zinc furnace will be roughly inversely proportional to the percentage of contained zinc. Our process, therefore, will have quite an influence on the concentration of zinc ores and utilization of crypto-crystalline ores which are now commercially unavailable. It is thought by us that the general tendency, when we are successful in our commercial trial, will be to shorter and more direct concentration with as much of the Cu, Zn and Pb in one concentration as is possible. Lead smelting and copper smelting with low rate for power will be as cheap or cheaper in the Johnson furnace than in the ordinary lead or copper furnace because of the increased recoveries, provided metals stay at average prices. The relation between concentration and smelting will be different. It looks as if what the Johnson process needs were zinc-iron middlings with considerable included

galena for maximum commercial efficiency, though, of course, existing concentration methods will also be applicable. We see that the general commercial conditions governing grade of ore are diametrically opposite in the Johnson process and in the retort process, where, with the exception of dry carbonate zinc ores, the cost of treatment increases as the grade of ore treated is lowered. Metallurgical needs of the Johnson process are for fluxing ores high in lime and iron."

"While I hesitate to disagree with Mr. Ingalls, who has studied so long on zinc problems, yet when he says that, 'if there be any dreams of an electric furnace analogous to a blast furnace in which run of mine ore may be charged roughly and from which spelter, lead and perhaps also copper matte may be obtained concurrently, I fear that there will be no realization of them.' I take issue direct with him. He, in my judgment, is not unqualifiedly wrong, but qualifiedly wrong. The success of our process will come at first because the Johnson furnace is similar and analogous to a lead blast furnace.

"The use of ZnO prepared by a fired process from low-grade sulphide ores in an electric furnace does not appear especially attractive, for we had the idea of igneous concentration of zinc ores making zinc oxide on apparatus similar to Wetherill grate in 1908 with subsequent electric smelting of ZnO so produced, but put aside the idea. However, the Truax process for manufacture of zinc oxides out of slags has, to my notion, certain brilliant possibilities and is a triumph of American originality. It certainly looks good. The Betts process, using silicon as a reducing agent, has decided possibilities. Consideration of wet or hydrometallurgical processes for zinc ores is useless. Their metallurgy is complicated, plant expensive, and cost of treatment high and uncertain. Quantitative analysis writ larger does not spell metallurgy. High recovery may some day bring them to life, but only at the time when every Italian laborer knows as much as a \$1000-a-month chemist. Their only chance, and that a remote one, is for very low-grade ores containing an insoluble gangue."

Mr. Johnson thinks that his particular problems are those of an engineer rather than those of an inventor and gives a number of tables of runs in his furnace during the last year. (Since the tables printed in the advance copies of the papers contained numerous numerical errors, the reader must be referred to the corrected tables which will appear in the *Transactions of the Society*.)

The "accounted zinc factor" in three runs (leaving aside a fourth run with a split condenser pipe) ranged from 80.5 to 81.9 per cent, the condensation factor from 77.8 to 81.5 per cent. The "accounted zinc factor" is defined as the ratio of weight of zinc accounted for in all products to weight of zinc delivered to electric furnace. The "condensation factor" is defined as the ratio of weight of zinc in spelter produced to the sum of weight of zinc in flue dust plus weight of zinc in ladle skimmings plus weight of zinc in blue powder plus weight of spelter.

In the same three runs the kilowatt hours per lb. spelter varied between 3.10 and 3.74, the kw. hours per short ton of preheated charge between 1452 and 1748, the thermal efficiency (estimated in two ways) between 33.8 and 41.4 per cent, the total electrode carbon consumption per 1000 kw. hours between 5.57 and 8.25 pounds. The spelter made was sold to a brass founder in Hartford.

"Our aim in these tests, basically considered, was to attain a condensation factor higher than retort practice. I estimate the maximum average yearly condensation factor attained in retort plants in the United States at 65 per cent. Many retort plants in this country, however, have a condensation factor of less than 60 per cent. Of course, the mixture of ores used at our Hartford experimental plant is not metallurgically or commercially possible in retorting practice.

"At our Hartford plant we have mastered condensation on an experimental scale, for some of our best condensation has been with refractory ores."

"Mr. Ingalls, in his extremely clear article, speaking of the claims as regards power consumption, states: 'While it is doubtful whether such a result has yet been actually realized by anyone, the claim is plausible with respect to certain kinds of ores, although even then it is necessary to assume a high furnace efficiency. It would seem that 1200 kw. hours per 2000 pounds of ore would be a sufficiently brilliant realization. Even such a one would put electric zinc smelting upon a sound basis in so far as power consumption is concerned. It is curious to note that 1200 kw. hours given by Mr. Ingalls as lower probable limit is upper probable maximum limit of power consumption which I have set down for 35 per cent zinc ores. Actually we have bettered this figure, and 600 kw. hours for a 35 per cent zinc ore does not seem to be impossible.

"With a glance at the future, and basing our reasoning on our own progress in the past three years, and extrapolating this progress, the prediction is hazarded that electric zinc-lead smelting will have a treatment cost of as low as \$6.00 per ton of ore, and that the losses exclusive of roasting losses will be: Zinc 40 pounds, lead 1 pound, copper 2 pounds, silver 0.30 oz., gold 0.01 oz., per ton of ore. This is for the ultimate future. For the immediate future multiply this estimate by two and we have a safe estimate of what can be done."

Mr. Johnson's paper elicited quite an extended discussion. Dr. Richards emphasized as one of the principles of the Johnson process the preheating of the charge so as to remove water vapor and reduce the reducible oxides (iron) with the carbon without reducing the zinc. He also said that Mr. Johnson had paid special attention to the treatment of complex ores, such as are not treatable with the retort process, like Bully Hill ore. But it may be questioned whether it would not be better to begin with simplest ores.

Mr. E. J. Ericson criticized briefly some special points in Mr. Johnson's paper and asked whether the claim of a condensation factor as high as 80 per cent was not a misprint, since others, like DeLaval, had not succeeded in condensing any such percentage as this as liquid zinc. Dr. Roebor said that it was not a misprint, but that according to Mr. Johnson the condensation factor in his more recent work was 80 per cent or more. This was confirmed by Dr. Richards, who said that in the DeLaval furnace the zinc vapor was overheated, which caused condensation difficulties.

Mr. C. A. Hansen took up some points of the discussion of the day before. He thought the electric furnace was not suitable for all zinc problems. He also emphasized that the development of the electric furnace had been held back for several years by superficial statements of men who knew or should know better.

Mr. F. S. MacGregor discussed electrostatic separation in zinc metallurgy, with reference to the discussion, by Mr. Collins, of his own paper.

Mr. L. B. Skinner reviewed and criticized Mr. Johnson's paper at some length. He thought that the paper was incomplete and that the tables did not give sufficient data. He also emphasized that the development of a new process should be carried out in such a way that the simplest problems were first solved, beginning with chemically pure materials. He agreed with the author that hydrometallurgical processes were not promising for zinc.

This latter remark induced Mr. John L. Malm to speak briefly on his own process. A paper on this subject is expected to be presented at the next meeting of the Society.

After a few more remarks by Professor Clevenger and Professor Richards the discussion was closed and the second electric zinc paper, by Mr. Peterson, was taken up.

The Electric Zinc Furnace

The paper by Mr. Peter E. Peterson gives a quite detailed description of the experiments which the author has made during the last years in Butte, Montana. The author describes in detail, with illustrations, the construction of the various electric furnaces and condensers which he has used.

In this last experiments the efforts at improving the con-

condensers were temporarily abandoned and arrangements were made to recover all zinc products. To accomplish this a new furnace was built and special iron condensers were made to recover the blue powder; also a bag house was used to catch the zinc oxide in fumes.

A continuous twelve-day run was made without difficulties of any kind.

The charge was fed in every hour, and slag and matte tapped every twelve hours.

"This furnace in its present improved form is about all that can be expected of any furnace. There are far less difficulties encountered than in running an ordinary blast furnace. The deterioration of the furnace is small. Such a furnace should run from six months to a year without any extensive repairs. Large units can be constructed by merely the addition of more electrodes, and there is no reason why units of 25 to 50 tons each could not be operated as easily and cheaply as the one-ton units."

As to the metallurgy of electric zinc smelting Mr. Peterson says as follows:

"In the operation of the electric zinc furnace the composition and condition of the smelting charge is quite as important as the furnace. More care must be exercised in making up furnace charges than in ordinary blast furnace practice. The charge must be such that at all times there is a perfect reducing atmosphere in the furnace; the slag must be such that deterioration of the furnace lining is reduced to a minimum, and with a sufficiently high formation temperature to volatilize the zinc. A matte must be made that is uniform in composition and is readily tapped from the furnace, and made in sufficient quantity to effect a good saving of copper, gold and silver. The proportion and nature of the reducing carbon is important. As before mentioned, a perfect reducing atmosphere must be maintained, and at the same time the reducing carbon must be completely consumed. If an excess is added, it cannot be burned by an air blast as in blast-furnace practice, and the only oxygen available for its consumption is that held in combination by the constituents of the furnace charge.

"The physical character of the charge is quite as important as the chemical for the best work. The charge constituents should be finely divided so as to allow the various reacting elements to be in intimate contact with one another, thus insuring more perfect and economical results. But the charge must not be too fine when charging in loose rather than in brickette form. The particles should be of such weight or size as to readily settle from the zinc vapors and gases passing outward into the condensers. If the particles are too fine they are apt to find their way into the condensers in the unmetled condition.

"It is necessary to have the charge absolutely dry. The presence of water even in small quantities is serious, due to the oxidizing of the zinc vapor by decomposition of the water. What is still more serious is the instantaneous generation of steam and other gases, causing a great increase in the velocity of gases, thus disturbing the condensation of the zinc in the condensers and thereby causing a loss of zinc and also a loss of silver and other metals, due to furnace charge being blown outward into the condensers.

"If it is not possible to mix the charge with hot calcines from the roasting furnace and then immediately feed it to the electric furnace, there must be a preheating furnace to heat the charge before introducing it into the electric zinc furnace.

"The making of slags is governed by the nature of the furnace linings, and these are dependent on the nature of zinc concentrate or ore to be smelted. If the concentrates are high in iron and low in silica, then a basic slag high in iron is made, and consequently a basic furnace lining (such as magnesite brick) is used; and if the reverse is the case, then high-silica slags with silicious furnace linings are used.

"The principle used in the elimination of zinc from slags was to make slags of such high formation temperature as to prac-

tically volatilize most of the zinc before slagging of the charge begins.

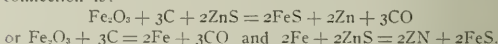
"In silicious slags the high percentage of silica is the principal controlling factor. In basic slags the percentage of reducing carbon is increased, which for practical purposes gives the same effect.

"In the work I have been identified with we have had practically no difficulty in eliminating zinc from either basic or silicious slags. The degree of elimination depended on the quantity of power used. Following are analyses of slags showing how zinc elimination can be varied:

	Zinc	Copper	Iron	Insoluble	Silver	Gold
Slag No. 1..	0.3%	0.24%	16.1%	66.2%	1.2 oz.	trace
Slag No. 2..	1.6	0.13	19.2	57.5	0.6	trace
Slag No. 3..	5.5	0.47	22.9	45.5	4.1	0.01 oz.
Slag No. 4..	8.5	0.38	23.5	44.4	2.1	trace

"In the treatment of complex ores the other metals, gold, silver and copper, are also recovered. This is accomplished by producing a collector (matte) similar to the lead button in crucible fire assaying.

"In the earlier work it was attempted to use as a collector a matte consisting principally of iron sulphide, but its use was not attended by any degree of success. The matte formed was very irregular in composition, varying from metallic iron to iron sulphide. The amount of matte formed was also irregular, and it was found necessary to add copper ores until the furnace charge would assay in the neighborhood of 1 per cent copper in order to get a matte uniform in quantity and composition. The quantity of matte was varied by varying the percentage of sulphur in roasted zinc calcines. Zinc calcines were successfully treated that varied from 2 to 15 per cent sulphur. The best results were obtained between 5 and 7 per cent sulphur. One of the important reactions in this connection is:



"Coke was found to give the best results as reducing agent. When coals were used it was noticed that the condensing of the zinc was seriously interfered with, due to deposition of fine carbon or soot in the condensers from the decomposition of the hydrocarbons in the coal."

The author discusses briefly methods of treating blue powder to change it into liquid zinc, and the possibilities of the design of a more efficient condenser.

"Electric zinc furnaces constructed on the same principles of heating as retort zinc furnaces have given as good condensing results. In these furnaces the heating of the furnace charge is accomplished by the radiation of heat from an electric arc located above the charge. Practice has proven this a very inefficient means of electric heat, yet so far as condensing is concerned it gives the best results."

Mr. Peterson finally illustrates "a proposed condenser for electric furnaces that should have a large capacity and be effective in its operation. The principle of the condensing is to pass the gases from the electric furnace through a molten bath of zinc; the bath is automatically maintained at the right condensing temperature by being covered with a molten bath of zinc chloride which boils at a temperature which will not allow the molten zinc bath to get too hot. The passage leading to the molten zinc is heated to prevent any condensing and consequent clogging of the tubes.

"For the operation of this type of condenser the zinc vapors must be distilled under pressure; for observations made with this idea in mind there should not be any difficulty in maintaining a pressure of 2 pounds per square inch (0.13 kg. per sq. cm.), which should be sufficient to work this condenser.

"The quickest and most effective method of condensing any vapor to liquid is the one mentioned above. A good illustration of the method of condensing is the passing of steam into water."

An estimate, worked out on a one-ton unit, of the costs per ton of pay charge, is given as follows:

Roasting	\$1.00
Supt. and Assay Office, etc.....	1.00
Labor based on 33-ton plant.....	.83
Powder—1401 Kw. hours, at 0.332 cent.....	4.65
Depreciation of plant.....	.40
9.5 lbs. electrodes, at 14 cents.....	1.33
0.1215 ton coke screenings, at \$5 per ton.....	.60
440 lbs. matte. Cost of marketing.....	2.86

Total cost \$12.67

"With the electric zinc furnace the following extractions can be depended upon for the above costs: Zinc extraction, 85 per cent; copper extraction with 5 per cent copper in charge is over 95 per cent, and with 1.5 per cent copper in charge is over 84 per cent; silver extraction, 65 to 72 per cent; gold extraction, from 98 to 100 per cent.

"The gold, silver and copper are recovered in the form of a copper matte. The difference in the value of the metals and that actually obtained from them is called the cost of marketing the matte.

"The zinc as obtained with the process at its present perfection of condensation with the above costs contains only 30 per cent spelter ready for the market. The remaining 70 per cent of the zinc is obtained in the form of an 80 per cent zinc product, as blue powder and some zinc oxide. This zinc as produced by the electric zinc furnace would be worth \$4.36 a hundred pounds when spelter is selling at \$5.50 per hundred, that is, when the 30 per cent spelter produced is placed on the market and the remaining 70 per cent of the zinc in the form of an 80 per cent product is shipped to zinc smelters on an ore basis treatment charge. Greater economies could be effected if the electric smelter had retorts of its own to treat this high-grade concentrate of blue powder and zinc oxide.

"If a straight zinc concentrate assaying 50 per cent zinc is shipped to a zinc smelter, the probable basis of settlement would be as follows: 85 per cent of the zinc content at market price of spelter, with \$16 deducted for treatment charge. With spelter at 5.5 cents per pound, one ton of 50 per cent concentrate is worth \$30.75.

"The electric smelter, after deducting \$12.67 for treatment, would return for the same concentrate \$23.41. But if a complex zinc concentrate containing approximately 10 per cent iron with copper, gold and silver values were to be considered, the difference would be greatly in favor of electric smelting. The penalties on iron, etc., and the poor extraction of copper, gold and silver by the retort smelters would make it impossible for them to compete with the electric zinc smelter.

"In the West complex zinc ores are abundant, and in some cases water power can be obtained near the mines, which would be an added advantage to the electric smelter by effecting a saving to the miner on freight on concentrates.

"Another probable source of increased revenue would be the sale of zinc products, as zinc oxide, for the paint industry. Statistics show that, in 1910, 16 per cent of all zinc produced in the United States was sold as zinc oxide, and in this form brought 0.1 of a cent more per pound than if sold as spelter. There is absolutely no difficulty in making zinc oxide suitable for paint by the electric furnace. The principal accessory required is a bag house and packing plant. A small percentage of the product could be sold as blue powder at prices about equal to that of spelter."

Mr. Peterson's paper was briefly discussed by Mr. Clevenger, Mr. Skinner, Mr. Ericson, Mr. Snow, and Dr. Richards. Mr. W. M. Snow, of the Illinois Zinc Co., said that in improved modern retort practice metal recoveries of 86 to 91 per cent were obtained. While the condenser of present zinc metallurgical practice had been evolved by try-and-hit-and-miss methods, it was nevertheless quite an efficient instrument.

Dr. Richards pointed out the following difference in condensation in retort practice and in the continuous electric furnace. In the former the condition of the gases is continually changing, in the latter it is constant. This difference may

necessitate a very different design of an efficient condenser in both cases.

Electric Smelting of Chromium, Tungsten, Molybdenum and Vanadium Ores

The paper by Robert M. Keeney, of the Bureau of Mines, on the above subject first takes up the electric smelting of chromite and the production of ferrochrome.

The author discusses the reduction of chromite by carbon, silicon, aluminium, calcium carbide, further the available ores and raw materials and the process of manufacture.

The electric furnace used in the manufacture of ferrochrome may be one of several types, all of the principle of the Siemens crucible with a conducting hearth, or similar to the Héroult steel furnace, with vertical electrodes in series.

"The process of manufacture of ferrochrome consists of mixing a charge of chromite and anthracite coal in proportions based upon the theoretical calculations given. This is shoveled into the open top of the furnace around the electrodes. The furnace is charged at regular intervals so as to keep the top in a solid condition. The ore is very finely powdered, but the coal is about as it comes from the mine. There is some loss by ore being carried off with the gases which escape at the top of the furnace.

"The ferrochrome is tapped from the bottom, at intervals of from two to four hours, into iron pots set on cars, from which it and any slag are dumped when solid. The slag and metal come out of the same tap hole. When cold the metal is broken up with hammers to separate it from the slag and packed into kegs or boxes. For tapping an iron rod is used to open up the tap hole, which is plugged with fire clay.

"Ferrochrome manufacture is carried on continuously, in some cases for two years, before shutting down the furnace for repairs. In one operation, directly from ore, ferrochrome containing down to 5 per cent carbon can be made economically. It has been possible to produce an alloy containing as low as 2 per cent carbon in one operation, but great care is necessary in the regulation of the charge and the operation of the furnace. The slags from the average ferrochrome furnace making a 5 per cent carbon alloy run from 0.5 to 1.0 per cent Cr_2O_3 ."

The author then discusses the refining of ferrochrome. Decarburization of high-carbon ferrochrome is accomplished by melting the alloy with a slag of lime, chromite ore and a little fluorspar. Other methods of refining are briefly noted.

The author then describes some experiments made by him in 1912 on the production of ferrochrome from chromite with carbon as a reducing agent. The conclusions drawn are: "First, that ferrochrome can be easily manufactured directly from chromite in the electric furnace; second, that the percentage of carbon in the ferrochrome cannot be kept low by regulating the carbon charged without excessive loss of chromium in the slag; third, that the percentage of carbon in the ferrochrome must be regulated by decarburization with an oxide slag of iron or chromite after tapping off the slag from reduction; fourth, that silicon and phosphorus cannot be kept low in the alloy under the strong reducing conditions necessary; fifth, that sulphur can be easily slagged; sixth, that, as the addition of lime does not seem to aid in removing phosphorus, it is advisable to use no lime, but as pure raw materials as possible; and, seventh, that the power consumption should not exceed 3.7 kilowatt-hours per pound of ferrochrome tapped, or 0.85 kilowatt-year per ton. Some further notes on power and electrode consumption are added.

The author next takes up ferro-tungsten, giving historical notes, theoretical considerations of the reduction of wolframite, ferberite or scheelite with aluminium, silicon or carbon, and describes the process of manufacture as follows:

"Ferro-tungsten is manufactured from ores in three ways: First, by direct reduction with carbon in a crucible; second, by reduction in an electric furnace by some reducing agent other than carbon; and, third, by direct reduction with carbon in an electric furnace.

"In manufacturing ferrotungsten by the crucible process, concentrates are placed in a clay-lined crucible with the proper proportions of reducing agent and flux, and heated to a high temperature in a gas-fired furnace. There is considerable wear on the crucible in this method. For a 30 per cent tungsten alloy the crucible will last about three heats, but for a 65 to 75 per cent product they last but one heat. Higher percentage tungsten alloys than this are not made in the crucible furnace.

"In the electric furnace process, ferberite, wolframite, hubnerite or scheelite are reduced with carbon in an arc furnace. As with many expensive alloys, the process is generally an intermittent one, *i. e.*, the reduced charge is tapped from the furnace before another charge is added, or the charge is let solidify and chiseled out of the furnace. Sometimes a campaign of several days is made. The reduced alloy is either decarburized with refining slags in the same furnace or melted and refined later in another furnace. The furnaces used are similar in design to the ones described for ferrochrome production, but a more complete recovery is effected by the use of a tilting rather than a stationary furnace. Wolframite, ferberite and scheelite are reduced easily in the electric furnace, most of the manganese being volatilized. Scheelite is more difficult to reduce, and very sticky basic slags result. There is a greater loss in its reduction, so that it does not command as high a price as the other ores, selling for about \$1 per unit less. The product is tapped into molds, broken up on cooling, and packed into kegs or boxes."

The author describes some experiments made by him on the reduction of ferberite with carbon to produce ferrotungsten in the electric furnace. From the experiments the following conclusions are drawn: "First, ferrotungsten can be produced directly from ferberite in the electric furnace; second, by the use of a decarburizing slag before tapping, the percentage of carbon in the alloy can be kept below 2 per cent; third, manganese, silicon, phosphorus and sulphur do not enter the ferroalloy in high percentages; fourth, the loss of tungsten in the slag need not be excessive, and, fifth, the power consumption need not exceed 3.46 kilowatt hours per pound of ferrotungsten tapped, or 0.79 kilowatt year per ton."

In discussing **ferromolybdenum** the author gives again historical notes and theoretical considerations on the reduction of molybdenum ores. Concerning the process of manufacture the author says:

"The commercial manufacture of ferromolybdenum is carried on in an electric furnace of the electrode type operated as a resistance furnace, or in a crucible furnace of the resistance type. Comparatively little ferromolybdenum is made because of difficulty in getting ores. This is due to the fact that satisfactory methods of concentration for molybdenite ores have not been fully worked out as yet. . . .

"Buyers require an ore or concentrate containing 90 to 95 per cent molybdenite, for which the price is about \$450 per short ton.

"In commercial manufacture, raw or roasted molybdenite, iron turnings, lime and coke or coal are mixed to give a ferromolybdenum of desired proportions, and treated in the electric furnace. The furnaces are operated intermittently, *i. e.*, one charge is completely discharged before another is added. Ferromolybdenum is first made as a high-carbon alloy containing 3 to 4 per cent carbon. This is then decarburized with a lime slag or a slag of lime and iron oxide. If the latter is used the percentage of iron in the alloy is increased."

From some experimental work of the author on the electric smelting of molybdenite the following conclusions are drawn: "First, that ferromolybdenum low in carbon can be made directly from molybdenite in the electric furnace, with excess lime as a desulphurizing agent and carbon as a reducing agent; second, that a product of low percentage carbon can be made, and, third, that sulphur can be readily slagged as calcium sulphide with a charge of excess lime."

The paper is concluded by some notes on **ferrovanadium**.

"Most of the ferrovanadium made is produced by the thermit

process, by reduction of the oxide V_2O_5 in a crucible with carbon or by reduction of vanadate of iron with carbon in a crucible. In Europe some ferrovanadium is made by reduction of the oxide, sulphide or vanadate of iron with carbon in the electric furnace.

"The largest source of vanadium is the Peru deposit of patronite, a sulphide of vanadium containing about 60 per cent sulphur and 20 per cent vanadium. In Colorado and Utah there are large deposits of low-grade carnotite, a uranium and vanadium oxide mineral. Another ore found there is roscoelite, which contains vanadium oxide but no uranium.

"Patronite ore is first roasted and then reduced by the thermit process. Or, in the electric furnace it may be directly reduced with lime as a desulphurizing agent and carbon as a reducing agent. The other ores are treated by chemical or ore-dressing processes to obtain the vanadium as oxide or vanadate of iron, and then reduced with carbon in a combustion or electric furnace or by the thermit method. With oxide the electric furnace method is similar to the production of ferrotungsten with carbon as a reducing agent. The sulphide reduction is similar to that of molybdenite with lime and carbon. Several methods have been patented using silicon as a reducing agent in the electric furnace. The process as carried on in the electric furnace has no novel features beyond those stated regarding chromium, tungsten and molybdenum. Decarburization is with oxide of iron ore or oxide ore of vanadium. Vanadium is rather difficult to reduce, and as a result excess carbon is used, which is absorbed up to about 1 to 4 per cent."

The Carnotite Industry

A paper on this subject by Dr. Siegfried Fischer, Golden, Colo., is interesting in its relation to the production of radium.

Mineralogical investigations have shown two sources which contain uranium in sufficient amounts and of which there are large enough deposits to justify their commercial working. These are pitchblende and carnotite. While there are considerable deposits of monazite, the thorium mineral which also carries at times small amounts of uranium, its comparatively low radio activity eliminates it from becoming a true source for radium production.

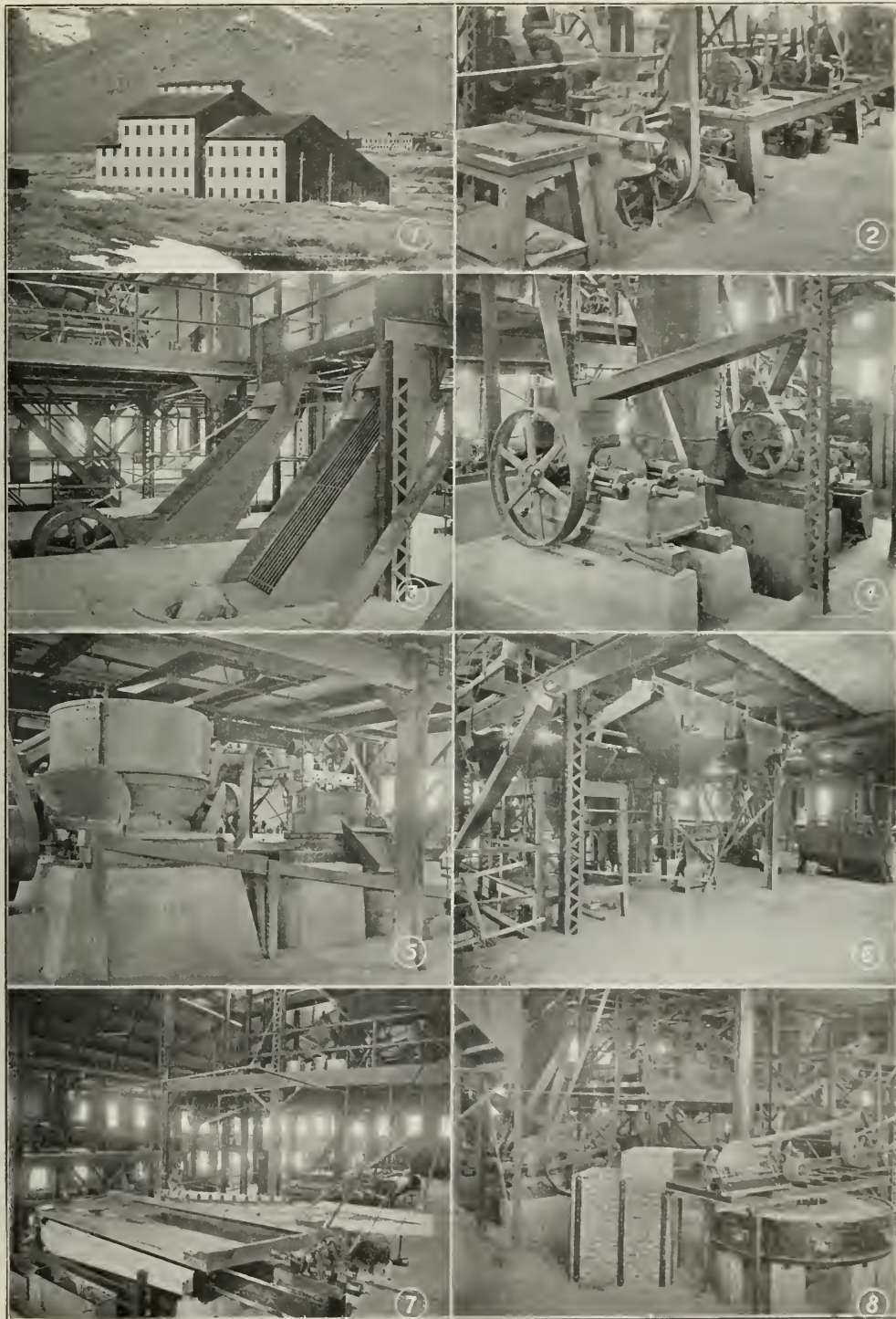
Up to recent years pitchblende was the commercial uranium ore.

The author's paper discusses the possibilities of carnotite for winning radium. Carnotite is a dark green to canary yellow, soft, powdery impregnation in some gangue material such as sandstone, limestone or clay, and on rare occasions in coal. According to the percentage of uranium in an ore, the yellow color and the radio-activity are more or less dominant.

Data are given on the analysis of carnotite ores, and occurrence. The main sources of carnotite are Colorado and Utah. Up to recent years nearly all the carnotite was shipped to Europe. All the radium made was a foreign product, and it was thought that America was incapable of utilizing its own crude material. "This idea is now a thing of the past, as only last month \$30,000 worth of radium was shipped abroad by an American firm. It will probably not belong before this country is one of the main producers of this valuable element."

"The intrinsic value of carnotite lies in its uranium and vanadium contents, and on these factors the commercial value is based. It is rather hard to give definite figures which will hold true in all cases, as the signed contracts between the dealer and the buyer are of an extremely varied nature. The only data that I can vouch for are from a Colorado firm selling carnotite, and the figures held for last year.

"The contract stated that ores must contain at least 2 per cent U_3O_8 and a maximum of 5 per cent V_2O_5 if anything at all was paid for the vanadium. On this basis the U_3O_8 sold at \$1.25 per pound, and the vanadium oxide at \$0.30 per pound. For foreign shipments the cost per pound of V_2O_5 was \$0.70 *f. o. b.* Hamburg or Liverpool. The price for the uranium to these ports is not known to me. For each



FIGS. 1 TO 10.—TESTING PLANT, COLORADO SCHOOL OF MINES. (FOR DESCRIPTION SEE PAGES 500 AND 580.)

per cent over the percentage stated in the contract the price per pound increased \$0.05.

"The same firm claims that the profit per ton is \$20, and the net cost per ton of ore is between \$50 and \$60, including sampling and analyzing.

"This crude industry is very simple. The ore is mined and sampled, analysis is made at both ends, and the product is sold on contract. But more recent researches performed in this country will undoubtedly put the carnottite industry on a more scientific and commercial foundation."

The author discusses at some length the question whether concentration is desirable or not as a first step in the treatment of carnottite. European trade seems to demand that the material be delivered without mechanical change. But the author pleads in favor of concentration and gives as his argument an abstract of his article published in this journal, June, 1912 (our Vol. 10, page 356).

As to subsequent chemical treatment the author distinguishes three steps: Extraction of values; separation of uranium and vanadium; production of radium from uranium.

As to the extraction of values (see also the author's article in our issue of August, 1912, Vol. X, page 469) there are two treatment methods in use at present—acid and basic extraction.

With respect to *acid* extraction processes information is given on the Flech, Haldane, White patents 880,645 of March 3, 1908, and 890,584 of June 9, 1908.

"The acid process has never appealed to me for the reason

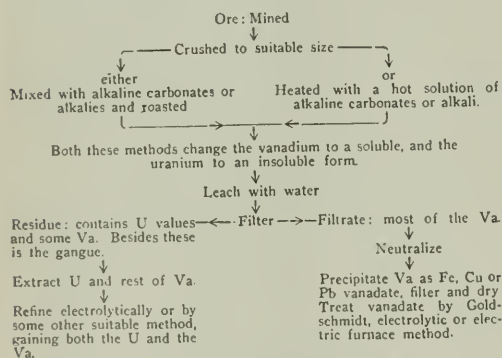


FIG. 4.—SCHEME OF BASIC TREATMENT OF CARNOTTITE

that so many impurities are brought into the extraction product, and must be gotten rid of at some stage of the operation. The patent also shows that the process is a tedious one, requiring many steps, thus increasing the time, reducing the output and increasing the cost. In spite of these facts, however, a plant is in operation based on these patents. Whether or not it is a success time alone can show, since it has been running less than a year."

A general diagram of the *basic* treatment scheme is shown in Fig. 4.

Most of the work along the line of the basic extraction of the values from carnottite was done by Warren F. Bleeker, now with the Standard Chemical Company of Pennsylvania. Bleeker's patents 1,049,330 (January 7, 1913), 1,015,469 (January 23, 1912), 1,050,796 (January 21, 1913), and Fischer's patent 1,054,102 (February 25, 1913) are outlined.

"Radium being closely allied to barium, a barium separation was performed on pitchblende ores. On this basis I have treated several uranium concentrates obtained from carnottite; and have had fairly good results. The quantity of concentrates used having been comparatively small, the barium and radium sulphate residues were not of sufficient amount to make absolute examinations on them, but the uranium residue, after treatment to eliminate all foreign matter, showed a decided decrease in radio-activity. Three methods are claimed to be applicable to carnottite for the extraction of the radium,

their commercial value depending greatly on the locality in which the manufacturing of the radium salts will take place. For further knowledge on this topic we will have to await the publications of the Bureau of Mines in Denver, which is doing extensive and most valuable work in this direction.

"As no industry can be considered a success until all the by-products are utilized, the carnottite industry cannot be classed as a fully developed commercial process. Vanadium has an established market; the demand for radium is increasing steadily, but what becomes of the uranium? When we consider 2,600,000 parts by weight of uranium will produce only one part of radium, it is evident how insignificant, commercially speaking, the amount of radium is in comparison to uranium. And yet for this important by-product of the carnottite industry there is practically no market demand. Some uranium is used in the glass and porcelain industry as a color medium, but the amount is a mere bagatelle. This subject calls decidedly for research. Some of the large steel companies have tried to use uranium in their line of work, but with little or no success. This seems to have discouraged investigators, and the work spent on this element, outside of producing radium, has nearly ceased. I would like to suggest investigating alloys of uranium with other metals besides iron. It may be that it will contribute properties to some that may be of the greatest value and open up a market for what is now practically a waste product."

Base-Metal Thermocouples

A paper by Prof. O. L. Kowalke, of the University of Wisconsin, discusses the possibility of getting reliable base-metal thermocouples.

One of the chief causes for the inaccuracy of a thermocouple is heterogeneity of the wires.

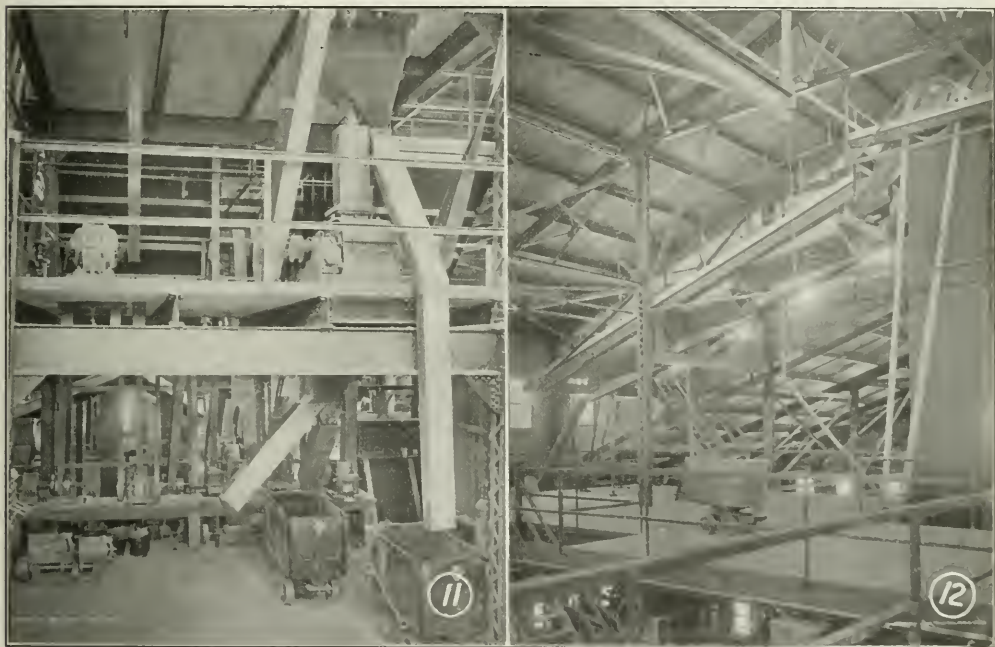
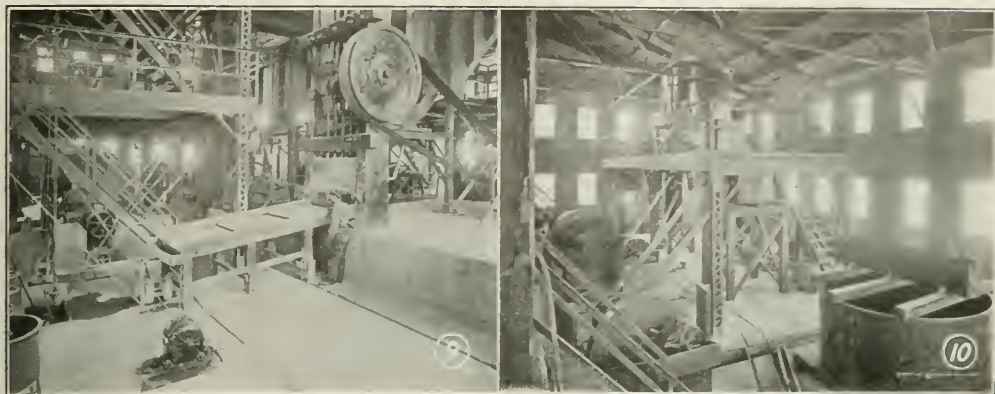
"Since the voltage generated by the couple for a given temperature is the summation of all the electromotive forces due to the contact of two dissimilar metal portions, it can readily be seen that it is imperative that the metal be as homogeneous as possible if the electromotive force indicated be that resulting from the two metals at the hot junction. If, then, the couple is composed of metal which is not homogeneous, then at each junction of two dissimilar metals there will be set up a voltage which is a function of the temperature existing at that point. Should the position of the couple be changed with respect to its depth of immersion, then the resulting voltage will be changed even though the temperature had not changed at all. A couple made of wires which are perfectly homogeneous is apt to be rather expensive, and the only combination which seems to fulfill this condition is the platinum and platinum-rhodium couple.

"It is possible to get a couple which is sufficiently homogeneous so that the indications are satisfactory for many commercial purposes where an accuracy of about 25 to 50 degrees is wanted. This condition is realized in the combinations of which base metal couples are made. Not all of the couples meet this condition with the same accuracy, and not all of the couples retain their original accuracy. Some couples appear to change or deteriorate with use more than others, due to oxidation, crystallization, segregation of the metal and other undetermined causes.

"It was the purpose of this investigation to determine how electromotive forces of couples varied with the temperature, and how constant this force remained with successive heatings and coolings, and also when exposed to various temperatures for extended periods of time.

"Couples were purchased from five of the prominent makers, and were cut to lengths of about 18 inches. To each couple about 3 feet of flexible lamp cord was soldered, and the wires at the point of soldering and all other required points were carefully insulated from each other."

These couples were subjected to the following careful tests: Calibration with a length of 4 inches heated; calibration with a length of 15 inches heated; heating to 400 deg. C. per 24 hours and renewed calibration with a length of 15 inches



Ore Dressing and Metallurgical Plant, Colorado School of Mines. Figs. 1 to 8 on page 587: Fig. 1, Outside View, Ore Dressing and Metallurgical Plant, Colo. School of Mines. Fig. 2, Preparation of Samples. Fig. 3, Classifier and Jigs. Fig. 4, Concentration Tables. Fig. 5, Wilfley Roaster. Fig. 6, Stamp Batteries and Plates. Fig. 7, Tube Mill and Cyanide Department. Fig. 8, Sampling Apparatus. Fig. 9, Elevator and Trommels.

heated; heating to 600 deg. C. for 24 hours and renewed calibration; heating to 800 deg. C. for 24 hours and renewed calibration.

The wires of the five couples had the following composition, the negative element being given first, the positive second:

- (1) Alloy of nickel and chromium, alloy of nickel and iron.
- (2) Nickel with a small amount of aluminium, iron-nickel alloy.
- (3) Nickel with a small amount of aluminium; iron with a small amount of nickel.
- (4) Nickel with a small amount of iron; iron with a small amount of nickel.
- (5) Nickel-copper alloy; iron-manganese alloy.

These couples had different protective coverings, of which one with a winding of asbestos cord, over which there was

applied a solution, presumably sodium silicate was apparently the most robust and satisfactory of the whole lot.

The results of the tests are given in form of diagrams and tables. They are thus summarized by the author:

"This series of investigations seem to point to the fact that it is possible to obtain a base-metal couple which is reasonably homogeneous and will give indications of temperature which are sufficiently constant to meet the needs of perhaps the greatest number of requirements for high-temperature measurements in the industries.

"There are some instances where the temperature is to be known to an accuracy better than 25 deg., and for such requirements it would be necessary to use a higher grade of equipment, and, furthermore, for such requirements it will be necessary to make frequent calibrations.

"It would seem reasonable that the makers of thermocouples determine from the purchaser the particular conditions under which the couple is to be used and that the calibration of this couple be made for the same length of immersion at which the purchaser is going to use it.

"A couple which will indicate an apparent difference of temperature amounting to about 130 deg. C. when immersed at different lengths in the bath or furnace is not giving the service that ought to be rendered.

"From these tests it appears that not all manufacturers are careful in sending out couples which have been thoroughly annealed to remove all strains resulting from mechanical treatment. Once the couple has attained its permanent structure there does not seem to be much change in the electromotive force."

Osmium-Platinum—A New Alloy

A brief paper by Dr. F. Zimmermann, of Baker & Co., Newark, N. J., gives an account of a research made to find a substitute for iridium-platinum on account of the growing scarcity of iridium.

Highly refined platinum and osmium have been successfully combined in widely varying proportions yielding alloys of commercial value. While the two metals may be combined in almost any proportion, alloys containing from 1 to 10 per cent of osmium and 99 to 90 per cent of platinum are chiefly used.

Great purity of the components is essential, as the presence of small percentages of other elements appears to be very detrimental to the properties of the resulting alloy. According to the chemical and physical behavior, it seems that one part of osmium in an alloy with platinum will take the place of two and one-half times its weight of iridium. The osmium-platinum alloy is very acid-resisting, and for this reason may be of great service in the electrochemical industry. Its electrical resistance is considerably higher than that of an iridium-platinum alloy of the same percentage composition. The alloy further possesses great hardness and tensile strength. Wires of the finest size are drawn with comparative ease.

Effect of Light on Decomposition Voltage

The paper on this subject by Alan Leighton, of Cornell University, reaches the following conclusions:

The decomposition voltage of a copper sulphate solution between platinum electrodes is not affected appreciably when the anode is illuminated by a mercury vapor lamp.

The decomposition voltage of a copper sulphate solution between platinum electrodes is increased when the cathode is illuminated by a mercury vapor lamp.

The light tends to make the solution and the deposited copper less stable. The latter effect is the greater under the conditions of the experiment.

It is possible to regulate the voltage so as to make copper precipitate on the shaded portions of the platinum cathode and not on the illuminated portion.

Graphite absorbs something from a copper sulphate solution, presumably a cuprous salt, which acts as an anodic depolarizer and can be removed by electrolytic oxidation. The reaction is accelerated by light.

Owing to this absorption the decomposition voltage for a copper sulphate solution with a graphite anode and a platinum cathode can be brought down temporarily to about 0.4 volt.

Rapid Refining of Copper with a Rotating Cathode

The paper on this subject, by C. W. Bennett and C. O. Brown, of Cornell University, points out that electrolytic refining of copper, as it is ordinarily done, is carried out with a current density of from 2 to 4 amperes per square decimeter of cathode surface. The amount of copper precipitated in one or two hours by a current of this density is small unless the cathode surface be very large.

For the purpose of demonstrating electrolytic refining as a laboratory experiment, the apparatus must be small, the

amount of metal deposited must be appreciable, and the time must be limited. If the cathode be rotated rapidly the current density can be increased greatly, current densities up to about 425 amperes per square decimeter having been used. With a small apparatus using current densities of this magnitude, sufficient amounts of copper for demonstration purposes can be deposited in a short time and appreciable amounts of the impurities may also be obtained as slimes.

The authors describe some tests on the rapid refining of copper to show that it could be done, and also with the idea of using it as a laboratory experiment. The experiments serve only to show that the same conditions obtain in rapid refining as in the commercial refining. They also give another instance of the rapidity of electrochemical work when the solution is stirred vigorously.

The experimental arrangement is described and illustrated.

The tests were made with a copper anode containing silver as only impurity, with an anode containing silver, arsenic and bismuth as impurities, and finally with an anode containing the usual impurities of copper anodes.

The tests show that electrolytic copper refining can be demonstrated in the course of an hour if a rotating cathode be used. The anodes may contain impurities usually found in commercial anodes, the demonstration being more striking if the impurities are present in amount approximating 1 per cent each. The anodes should be contained in a diaphragm, and the slimes may be removed partially with the diaphragm and the remainder by means of a stiff brush and water from a wash bottle. The deposited copper may be stripped off and sampled if aluminum cathodes are used.

Arsenic shows the largest tendency to contaminate the refined copper. It requires, however, the delicate Marsh test to detect this metal.

The statement that silver dissolves in neutral copper sulphate solution is probably an error.

Simultaneous Determination of Copper and Lead with the Rotating Anode

A paper by Arthur J. White, of Philadelphia, Pa., points out that, in electrolyzing nitric acid solutions of copper, lead tends to hold up some of the copper. However, an addition of sulphuric acid allows of copper being completely precipitated from a nitric acid solution while lead is being determined simultaneously on the anode, provided that the proportion of lead is not high enough to precipitate lead sulphate. This salt does not appear unless the proportion of lead to copper is greater than one to four, as the copper nitrate exerts a solvent action on smaller quantities.

When less than 0.15 gram of lead is present with 0.60 gram of copper, they are quantitatively precipitated in thirty minutes by a current of 8 amp at 6 volts.

The best proportions of acid are 4 cc. concentrated nitric acid and 10 drops of concentrated sulphuric acid in a volume of 70 to 80 cc. The anode is rotated at from 400 to 700 revolutions per minute and the copper is deposited on a 60-gram platinum dish.

The application to brass analysis is described in detail and some further observations are given which seem to warrant the application of the method to other alloys, provided they contain less than 2 per cent of arsenic or bismuth. These metals in small quantities act as reducing agents and require the addition of a little more nitric acid, whereas iron has an oxidizing effect and allows of less nitric acid being used.

Solid Thick Deposits of Lead from Lead Acetate Solution

The paper on this subject, by Prof. Frank C. Mathers, of Indiana University, emphasizes that "any soluble lead salt will give a solid, dense, coherent electrolytic deposit if the proper addition agents are used. Those solutions which do not give good deposits will do so when the proper addition agents are introduced. This theory cannot as yet be proved in all cases, but this paper gives the experimental proof in the case of lead acetate solutions."

Peptone was found to be the best addition substance for use in lead acetate solutions containing ammonium perchlorate. The effect of varying amounts of ammonium perchlorate, the effect of acetic acid, the effect of the quantity of peptone and of other factors were studied in detail.

Of other salts in place of ammonium perchlorate, sodium perchlorate and sodium naphthalene sulphate gave good deposits.

Among other addition agents the following were studied: Extract from feathers, casein boiled in acetic acid, casein modified by digestion with enzymes, tannin, meat extracts, vegetable extracts, essential and fatty oils, gums.

* * *

After a vote of thanks to all who had made this Colorado meeting so eminently successful, the meeting adjourned.

The final meeting held on top of Pike's Peak has been reported above.

The next meeting will be held in April, 1914, in New York City.

The following is a complete alphabetical list of those who registered at the meeting:

John C. Bailer, Golden, Colo.; T. F. Bailey, Alliance, Ohio; F. S. Bauer, Boulder, Colo.; E. A. Beck, New York City; F. D. Burr, Denver, Colo.; W. W. Case, Jr., Denver, Colo.; Regis Chauvenet, Denver, Colo.; W. R. Chesley, Golden, Colo.; G. B. Chorpene, Clarksburg, W. Va.; F. L. Clerc, Boulder, Colo.; G. H. Clevenger, Palo Alto, Cal.; C. L. Colburn, Butte, Mont.; A. E. Gerald Collins, Denver, Colo.; Mr. and Mrs. C. E. Collins, Denver, Colo.; C. W. Comstock, Denver, Colo.; W. D. Coolidge, Schenectady, N. Y.; E. L. Crosby, Detroit, Mich.; C. L. Devine, Clarksburg, W. Va.; D. C. Dodge, Denver, Colo.; Mr. and Mrs. J. V. N. Dorr, Denver, Colo.; C. J. Downey, Denver, Colo.; Wm. Dreyfuss, New York City; Mr. and Mrs. C. E. Eastman, Denver, Colo.; L. Ekeley, Boulder, Colo.; E. J. Ericson, St. Louis, Mo.; H. S. Evans, Boulder, Colo.; C. G. Fink, East Orange, N. J.; Herman Fleck, Golden, Colo.; Prof. and Mrs. F. C. Frary, Minneapolis, Minn.; R. H. Gaines, New York City; A. Galbraith, Chicago, Ill.; A. E. Gibbs, Philadelphia, Pa.; F. W. Gill, Arlington, N. Y.; Prof. and Mrs. S. L. Goodale, Pittsburgh, Pa.; J. R. Griffiths, Niagara Falls, N. Y.; H. M. Hall, Massena, N. Y.; C. A. Hansen, Schenectady, N. Y.; R. S. Hawley, Golden, Colo.; C. W. Henderson, Denver, Colo.; C. B. Hersey, Denver, Colo.; A. T. Hinckley, Niagara Falls, N. Y.; M. B. Holt, Denver, Colo.; A. J. Hoskins, Denver, Colo.; Mr. and Mrs. Wm. Hoskins, Chicago, Ill.; Miss Hoskins, Chicago, Ill.; J. A. Hunter, Boulder, Colo.; B. H. Jackson, Boulder, Colo.; D. R. Jenkins, Boulder, Colo.; H. W. Kellogg, Niagara Falls, N. Y.; Prof. and Mrs. Geo. Kemmerer, Snorro, N. Mex.; S. P. Kinney, Salt Lake City, Utah; K. A. Kithil, Denver, Colo.; R. Lachman, Breslau, Germany; F. B. Laney, Washington, D. C.; L. R. Leonard, Boulder, Colo.; O. C. Lester, Boulder, Colo.; F. A. Lidbury, Niagara Falls, N. Y.; S. C. Lind, Denver, Colo.; H. B. Lowden, Denver, Colo.; D. H. Lyon, Pittsburgh, Pa.; J. A. MacDonald, Denver, Colo.; F. S. MacGregor, Boston, Mass.; J. L. Malm, Denver, Colo.; B. S. Manuel, Denver, Colo.; J. A. Mathews, Syracuse, N. Y.; M. S. McCarthy, Denver, Colo.; A. McK. Gifford, Pittsfield, Mass.; R. McNeill, New York City; L. F. Miller, Golden, Colo.; R. B. Moore, Denver, Colo.; J. M. New, New York City; and Mrs. H. A. Nemelee, Denver, Colo.; O. C. Ralston, Colorado Springs, Colo.; J. W. Richards, South Bethlehem, Pa.; E. F. Roebur, New York City; J. W. Root, Denver, Colo.; C. C. Saunders, New York City; G. H. Seihman, Denver, Colo.; C. Schneiderberg, Pittsburgh, Pa.; Prof. and Mrs. Herman Schlundt, Columbia, Mo.; Prof. and Mrs. E. P. Schoch, Austin, Tex.; Harry M. Showman, Golden, Colo.; Mr. and Mrs. L. B. Skinner, Denver, Colo.; F. W. Skirrow, Westminster, Colo.; Acheson Smith, Niagara Falls, N. Y.; W. M. Snow, Peru, Ill.; H. N. Spicer, Denver, Colo.; C. D. Test, Golden, Colo.; F. W. Traegen, Golden, Colo.; L. D. Vorce, Detroit, Mich.; H. V. Welch, San Francisco, Cal.; E. S. Wiard, Denver, Colo.; H. J. Wichmann, Denver, Colo.; M. W. Wilkinson, Denver, Colo.; H. H. Willard, Ann Arbor, Mich.; Miss C. Wilson, New York City; L. H. Wilson, Newark, N. J.; Harry J. Wolf, Golden, Colo.

Old Freibergers in America

The dinner in honor of Oberbergrat Dr. Richard Beck, Rektor of the Freiberg Bergakademie, Freiberg, Saxony, Germany, was held on Tuesday, September 9, 1913, at the Engineers' Club, in New York City. Many reminiscences told of former Freiberg days and several toasts were given. Mr. Franklin B. Guiterman, executive director of the American Smelting & Refining Company, acted as toastmaster. Dr. R. W. Raymond, Secretary Emeritus of the American Institute of Mining Engineers, responded to the toast "Reminiscences." Mr. Charles L. Bryden, head of the school of Metal Mining and Metallurgy, Scranton, Pa., responded to the toast "Old Freibergers in America," and Dr. Beck responded to "Alma Mater."

Dr. Beck, who has been visiting this country, attended the International Geological Congress in Canada. While there he was honored by the degree of LL.D. from the University of Toronto, Toronto, Ontario, Canada. Dr. Beck sailed for Europe on September 12.

The organization of the Old Freibergers in America is just one year old and the membership is over 60.

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver

Slime Agitation.—A new type of slime agitator is described in the monthly *Journal of the Chamber of Mines of Western Australia*, by Mr. H. B. WRIGHT. The device, which is illustrated in Fig. 1, is in use at the Great Boulder Perseverance mine, and has given improved results in extraction as compared with the ordinary type of mechanical agitator. It is known as the Wright-Jaentsch centrifugal aerator, and consists essentially in several pipes, say four, supported at an angle of 45 deg. in the tank and attached to the central revolving shaft. The pipes act as centrifugal pumps. Four pipes of 6 in. diameter, revolving at a rate which gives the outside ends a circumferential speed of 350 ft. per minute, have given good results. The author gives figures showing

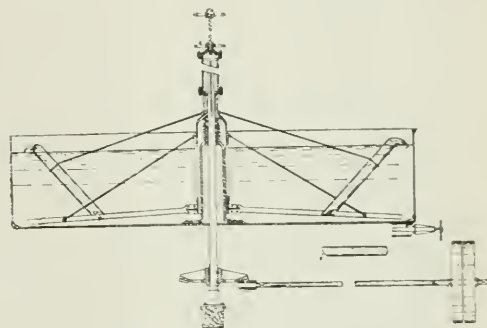


FIG. 1.—WRIGHT-JAENTSCH CENTRIFUGAL AFRATOR

comparative tests between the centrifugal aerator and the ordinary mechanical agitator, working on the same pulp. The centrifugal aerator showed an increase of 7.5 per cent dissolved gold as compared with the mechanical agitator. Twenty-three of these aerators are in operation at the Great Boulder Perseverance; each vat is 20 ft. diameter and 4 ft. 6 in. deep, and the arms are rotated at 6.7 revolutions per minute. Three horsepower is required for each vat. The centrifugal force exerted is 6.5 lb. The greatest pumping effect is obtained when the pipes are just submerged, as then there is no hydrostatic head to be overcome.

The Cam & Motor Mill, Rhodesia.—One of the newest mills to be designed for the Hartley mining district of southern Rhodesia is the Cam & Motor. The method of treatment to be adopted was determined by laboratory tests supplemented by work in an experimental unit. The details of this work and an outline of the new mill are given by Mr. GORDON F. DICKSON in the August, 1913, *Mining Magazine*.

The ore has the following composition:

Silica	44.33	per cent
Iron pyrite	2.95	" "
Mispickel	2.15	" "
Stibnite	0.15	" "
Calcium carbonate	14.10	" "
Magnesium carbonate	26.10	" "
Graphite	Trace	

Laboratory experiments proved conclusively that an attempt to cyanide the raw ore would be unsuccessful, owing chiefly to the consumption of cyanide and the loss of free alkali through the decomposition of the existing minerals. Preliminary roasting indicated a high percentage extraction. A "sweet" roast was obtained in 45 minutes, and the extraction by leaching or agitation was 85 per cent to 80 per cent. The addition of both stibnite and graphite above the usual quantity was without appreciable effect, but care was required in the roasting operation to avoid fusing the stibnite. The carbonates of calcium

and magnesium were found to have a useful effect in maintaining the protective alkalinity of the solution. The dissolution of gold was rapid, and the consumption of cyanide was low.

In the operation of the experimental unit, all the data from laboratory tests were confirmed. On one occasion the extraction dropped suddenly from 85 per cent to 78 per cent, which was found to be due to the tendency of the ore to ball and sinter at the discharge of the furnace. This was remedied by lowering the temperature of the finishing hearth and opening side and end ports, whereupon the extraction rose again. Tests were made on the furnace, which is of the Edwards type, to see whether wood or coal was best for fuel. Under the economic conditions existing at the time, wood fuel proved economical. The furnaces have been constructed to burn wood, but can be changed to the coal-burning type when cheaper coal is available. One of the furnaces has been equipped with the Parkison patent fire-box of the type used at the Golden Cycle mill, Colorado Springs, Colo. This is designed to burn the coal on the producer principle and give a long incandescent flame.

Experiment has shown that 40-mesh is the economic limit of grinding. The sand and slime will be separated and treated by leaching and agitation, respectively. Caldecott cones and sand dewatering tables will be used to prepare the sand for leaching. The slime will be filtered in a vacuum filter of the fixed submerged leaf type. The entire plant will have a capacity of 15,000 tons per month.

Economics of Milling.—In the *Mining Magazine*, August, 1913, appears an article on this subject by Mr. GELASIO CAETANI, being one of a series of lectures delivered by him recently at Harvard University. In our issue for August, 1913, page 464, we published an abstract of the first of this series of lectures, on the principles of mill design.

Mr. Caetani first calls attention to the fact that the object of ore treatment is not to attain the highest metallurgical efficiency, and that high percentage of recovery is not necessarily an index to the efficiency of a metallurgical plant. Metallurgical recovery and economic recovery of metals are not of equal importance; the latter is paramount in practice, being the sole object of the plant. Economic efficiency is the resultant of several factors, chief among which are (1) metallurgical efficiency, (2) freight and transport charges, (3) smelter schedule, and (4) operating cost.

Economic efficiency and metallurgical efficiency may become almost identical when the final mill products are metallic, as for instance in the case of the production of gold and silver bullion from cyanide mills, or the output of metallic copper from mills in the Lake Superior region. But there are exceptions even in such cases. The author cites an example of a cyanide mill treating a gold-silver ore, the gold content of which is recovered quite readily, whereas the dissolution of the silver is slower and requires days of treatment to reduce the tailing to its minimum silver content. In such a case the economic efficiency and metallurgical efficiency will not be identical, and it may prove more profitable to leave the silver in the ore rather than spend additional time and money in extracting it.

The discrepancy between economical and metallurgical recoveries is most apparent in concentrating mills. It hinges on the fact that, under normal operating conditions, metallurgical recovery can be increased only by lowering the grade of the concentrates, and the grade of the concentrates can be raised only at the expense of metallurgical recovery. In order to operate the mill to the best economic advantage, the metal market, smelter schedule, etc., must be taken into consideration, in order that the greatest net profit may be obtained from the mill products. Discrimination between concentrate and tailing may be dependent entirely on the intrinsic value of the material itself.

In another case, the addition of low-grade middling to the concentrate will lower the average grade to a point where the unit prices paid for the metals are also reduced thereby, en-

tailoring heavy loss. As an example, assume an auriferous pyrite in quartz, the average concentrate from which will assay 2 oz. gold and 5.4 per cent lead. Suppose the smelter contract offers \$19 per oz. for gold; no payment for lead under 5 per cent, and 40c. per unit for contents above 5 per cent; treatment charge, \$5 per ton. In milling the ore we may get a middling containing gold to the value of \$6.65, 2.5 per cent of lead and 16 per cent zinc. At first sight it might appear that, since the middling contains \$6.65 in gold and the treatment charge is only \$5 per ton, there should be a net profit of \$1.65. But if 1 ton of the middling is added to 9 tons of concentrate, we will have an average grade of 1.83 oz. gold and 4.96 per cent lead. Consequently the value of the lead is entirely lost, with the result of a net loss per ton of concentrate. Mill operators sometimes delude themselves in the matter of shipping low-grade ore and mill products, and resort to the device of mixing them with higher grade material, not realizing that they are losing just as much as though the two products were shipped separately.

On the other hand, conditions may exist under which low-grade material can be thrown into the concentrate. The author cites an experience in which he disposed of the last spigot-discharge of a spitzkasten by mixing it with the slime concentrate. The profit of concentrating this spigot-discharge would have been but \$1.32 per ton of dry material treated, whereas, by mixing it with the slime concentrate and shipping to the smelter it netted \$4.69 per ton. The reason for mixing the two products was that the smelter would have refused to accept the low-grade spigot-discharge if shipped alone. Other instances of mixing concentrates to advantage may occur when the effect of mixing is to reduce the average grade of a deleterious constituent, such as zinc, without impairing seriously the grade of the valuable minerals. In this way a penalty on zinc may sometimes be avoided.

The author cites the operations of the Utah Copper Company as an example of the effect of large-scale operations on low-grade material. Success with a low-grade ore will depend on three factors: capacity, full load and continuity of operations.

In the case of combined concentrating and cyaniding plants, the two departments must be operated in such a way that the highest economic efficiency can be obtained. This will depend on the nature of the ore, the effect of the sulphides on the cyanide solution, the freight and treatment charges on concentrates, etc. Under some conditions it may pay to cyanide concentrates separately, and not ship them to the smelter. As an example, the practice at the Alaska-Treadwell is cited, where formerly it cost \$12 per ton to ship and smelt concentrates, as against a total cost of about \$5 for cyaniding at the mill.

These various considerations all emphasize the fact that the maximum economic efficiency and not maximum metallurgical efficiency is the ultimate object of the reduction plant. Many factors enter into the problem and it is the duty of the metallurgical engineer to watch the ever-changing conditions and to operate his mill in accordance with good business principles.

Zinc and Lead

Selective Flotation.—One of the recognized problems in slime concentration at Broken Hill, Australia, is the separation of galena from blende. Mr. J. Malcolm Newman, writing in the *Australian Mining Standard*, June 26, 1913, states that while it has been possible by flotation to recover lead, zinc and silver from the slimes, leaving a worthless gangue, it has been practically impossible to separate the two minerals into marketable products. Hitherto the separation has been attempted on concentrating tables and vanners, but the operation was unsatisfactory, even though the products could be mixed with those of higher grade and thus sold.

Recently efforts have been made to evolve "selective" or "preferential" flotation processes, whereby the galena and blende would be acted upon differently and thus yield separate concentrates. The Zinc Corporation is experimenting with two such processes, the Horwood and the Lyster. The importance of slime concentrating and separating processes is increasing with the deeper operation of the mines, for the minerals be-

come more intimately associated, and the opportunity for jiggling and other forms of coarse concentrating is decreased. In fact there is already a diminution in the amount of jiggling now done, and finer grinding becomes necessary to liberate the mineral particles. This produces more slime and increases loss of mineral; hence the need for methods of slime separation.

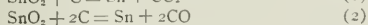
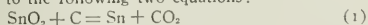
The Horwood process is better known than the Lyster. In the *Mining and Engineering Review*, Australia, July 5, 1913, some details are given of the operation of this process at the Zinc Corporation. The material treated is slime from the re-concentration section of the Minerals Separation flotation mill. It contains about 36 per cent zinc, 15 per cent lead and 22 oz. silver per ton. The slime is first thickened, then filter-pressed and dried with compressed air so that the cakes crumble when dropped from the press. The product is then roasted in an Edwards duplex furnace, at a temperature of 300 to 350 deg. The lead is sulphatized, but the zinc is unaffected, and in the subsequent flotation process only the latter floats. About 35 lb. acid and 2 to 3 lb. oleic acid are added per ton of slime. The mixture passes through seven flotation boxes, six of which produce zinc concentrate which is retreated in the seventh. The lead remains in the underflow. By this method it has been possible to produce lead concentrate containing 38 per cent lead, 8 per cent zinc and 42 oz. silver per ton; while the zinc concentrate runs as high as 50 per cent zinc, 7 per cent lead and 15 oz. silver per ton.

Less complete detail is given on the Lyster process, in the June 5 issue of the same journal. Current slime is first thickened and then fed to a centrifugal pump which is throttled to cause a violent agitation of the slime with oil and air which are admitted into the suction pipe by a small duct. The mixture is pumped to a spitzkasten with contracted overflow, in which the galena particles collect in a froth that overflows. The underflow, containing mainly zinc mineral, is then similarly treated in a second centrifugal pump with a further addition of oil, whereupon a zinc concentrate is obtained. Similar results have been obtained in Minerals Separation boxes, but the centrifugal pump is considered more efficient. The main point in the treatment seems to be successive additions of small quantities of oil followed by agitation, thus producing a fractional flotation of the different minerals. The report states that 55 per cent to 60 per cent lead concentrates have been obtained, and that a pure zinc concentrate has been made, thus demonstrating that the differential process can be applied to mixed slimes.

Tin

Electric Smelting.—Interesting details of experiments in smelting tin ore in the electric furnace are given by "A Special Correspondent" in the *South African Mining Journal* for June 7 and 14, 1913. After reviewing the present status of tin smelting in reverberatory furnaces, the author takes up theoretical considerations in electric smelting, and then cites results of experimental work. He concludes that under favorable conditions for power, the electric furnace process offers material advantages over reverberatory smelting.

The reduction of tin oxide by carbon may be assumed to occur according to the following two equations:



That part of the reaction takes place according to the second equation is indicated by the fact that CO can be seen burning at the mouth of the furnace. The author assumes that reduction takes place by both reactions, probably in equal proportions.

The combustion of 118 grams, or one gram-molecule, of tin into tin oxide will yield 145,300 gram-calories of heat, which is the amount of energy required for liberating 118 grams of tin. The combustion of 12 grams of carbon into CO₂ will yield 96,960 gram-calories; and the combustion of 24 grams of carbon into CO will give 58,000 gram-calories.

If the reduction occurs according to reaction (1) the energy requirement per ton of tin will be 145,300 = 96,960 = 48,340 kg.

calories per 118 grams tin, or 499,661 kg. calories per kg. tin, or 474 kw-hours per ton of tin.

In the same way the requirement according to equation (2) will be 855 kw-hours per ton of tin. Assuming that the reduction occurs to the extent of 50 per cent. of each reaction, the requirement would be, in round numbers, 665 kw-hours per ton of tin. In addition to this, a certain amount would be needed to heat the charge to the temperature of reaction.

Assuming that the ore contains 90 per cent tin oxide, we require 1280 kg. ore for each ton of tin. Experience has shown that the slag will amount to 16 per cent to 20 per cent per ton of ore, or say 220 kg. slag. The melting of a ton of slag requires, on an average, 500,000 gram-calories, or 130 kw-hours for each ton of tin. The heat capacity of the charge and molten tin may be set down as 65 kw-hours. Radiation losses are not definitely known, but are assumed to be as great as that consumed by the slag, or 130 kw-hours. The gases escaping from the furnace have a temperature of about 800 deg. C., and the loss from this source may be set down as 150 kw-hours per ton of tin. In summing up these items it appears that it should be possible to produce a ton of tin with the following expenditure of power: For the reaction, 665 kw-hours; slag loss, 130; specific heat of the charge, 65; radiation loss, 130; loss in gases, 150; total, 1140 kw-hours.

Current for the experiments under review was obtained from a Diesel engine plant, and was three-phase, 50 cycles, 650-675 volts. An oil-cooled transformer received current at 660 volts, and had four tapplings capable of giving 30, 40, 50 or 60 volts of 2500 ampere current.

The furnace was a shaft of special design, which is not described owing to pending patent applications. There were three carbon electrodes, 4 in. square and 4 ft. 6 in. long, cooled by water jackets, and suspended in such a manner as to permit fine adjustment. No direct arc was permitted to be formed, as this caused a great variation in the yield and in the working of the furnace. The charge formed a cone-shaped mass on the hearth, covering the reacting zone forming a small space filled with incandescent gas in which the reaction took place. Arrangement was made to prevent the escape of volatile metal, and only about 0.5 per cent of metal was thus lost.

At the start of a campaign of ten or twelve days, the furnace was preheated by a wood and coke fire. The charge contained 14 kg. culm for each 100 kg. ore. Current was first used at 60 volts, 1000 amperes per phase, and as the reaction proceeded this was changed to 40 volts and 2500 amperes, which current was maintained during the smelt. Metal was tapped half an hour after charging, and slag was drawn after the furnace had been running two hours. The proper operation of the furnace could be judged by the appearance of the slag: if it flowed freely, and was a dark brownish green color, the charge was right, but if the slag appeared sluggish and light brown a change was indicated.

Metal from the tap-hole assayed 98 per cent pure. In one case Bolivian ore of 49.9 per cent tin and 15 per cent iron was smelted for 10 days, with a net recovery of 92 per cent, running as high as 97 per cent recovery on one day. The grade of the metal from the furnace was raised to 99.75 per cent pure by simply blowing air through it.

The output, power consumption and loss in slag were found to be in close relation. It was possible to produce slag containing only 0.25 per cent metal, but this was not economical at the price for power. The output was only 6 to 8 cwt. per day, and the power consumption was 3000 kw-hours per ton of metal. On the other hand a slag containing 17 per cent to 18 per cent tin was produced, the output of metal being 1 ton and 11 cwt. per day, with a power consumption of only 1300 kw-hours per ton of metal. This slag would have to be re-treated.

It was finally found most economical under prevailing power

conditions to obtain a recovery of 96 per cent to 98 per cent, with a high power consumption.

The following figures refer to operations during the week of July 28 to Aug. 4:

Ore smelted—	
20,056 lb., 57 per cent tin.....	12,003 lb. metal
Scrap and refuse.....	2,324 " "
Tin produced	14,170 " "
Net yield	98.75%
Electrode consumption, per ton tin.....	28 lb.
Power consumption per ton tin.....	1,800 kw-hours

An Electrolytic Refinery in Wall Street

The Assay Office of New York

It has been said by a cynic that the only chemical compound with which modern financiers are thoroughly acquainted is water and that more than one triumph of modern industry has been acquired by utilizing the reaction between stock and water. Be that as it may. It is a plain fact that there is a real electrolytic refinery in Wall Street, New York City, small in size, it is true, but very big in the value of its output. This is the United States Assay Office. Next to it is the Sub-Treasury, and across the street is the office of J. P. Morgan & Company. No explanation is necessary to show that this electrolytic gold and silver refinery is at its logical place.

The relation between the Assay Office and the public is



FIG. 1.—ANODE MELTING FURNACES

twofold. First, the Assay Office buys impure gold and gold-silver alloys from the public. This may be the gold output of a copper refinery or of a cyanide mill or gold scrap from dentists, or gold imported from Europe, or old jewelry. Second, the Assay Office refines this gold and alloys for the various requirements of the public.

The metallurgical process carried out at the Assay office is, therefore, the separation of the gold from the silver and the other metal impurities and the production of the pure metals.

Formerly the sulphuric acid parting process was employed, but the electrolytic refining process has now been universally introduced into the United States Mints, and when a few years ago the New York Assay Office was damaged by fire and it became necessary to rebuild and reequip the Office thoroughly, it was decided to introduce electrolytic refining.

There is no necessity here to enter into all the intricacies of the electrolytic refining of gold and silver, since they have been dealt with in detail in numerous articles in this journal, written by the highest authorities. Reference to these articles must therefore suffice. The most important are by the inventor of the electrolytic gold refining process, Dr. E. Wohlwill, in our vol. II, p. 221 and 261, and vol. VI, p. 450; by Dr. D. K. Tuttle, who introduced the process in the Philadelphia mint and worked out a method of treating gold-silver alloys high in silver, vol I, p. 157, and vol. IV, p. 306; an article on the use of pulsating current in the Wohlwill gold refining process, vol

VIII, p. 82; further see the articles by Whitehead on the refining process in use at the San Francisco mint, vol. VI, p. 355 and 408 and by Dr. E. F. Kern, vol. IX., p. 443.

In buying gold or gold alloys from the public, the Assay Office has fixed only two restrictions. These are that the ma-



FIG. 2.—ELECTROLYTIC SILVER CELLS

terial shall contain not less than 20 per cent. of gold and silver and shall be worth at least \$100. The purchased deposits are melted down and molten dip samples are taken for assays.

Now a separation is made according to whether the per-



FIG. 3.—ELECTROLYTIC GOLD CELLS

centage of gold is less than 90% or more than 90%. In the first case the material goes to the silver refining room, in the latter case directly to the gold refining room.



FIG. 4.—GENERAL VIEW, DEPOSIT MELTING ROOM

The crucible furnaces employed for melting the material and casting into anodes for electrolytic silver refining are shown in Fig. 1, the electrolytic silver refining room in Fig. 2. The electrolyte is a solution of 3 percent nitrate of silver and 2 percent free nitric acid, this composition being carefully maintained. The tanks are of brown stoneware.

The anodes contain from 250 to 400 parts of gold, 150 or less of base metals, and the remainder silver (out of a total of

1,000 parts). The anodes are contained in muslin bags to retain the slime. The solution is stirred by a small belt-driven glass propeller in each tank.

The silver and base metals are dissolved in the electrolyte,



FIG. 5.—SWEEP HANDLING MACHINERY

but the silver alone is deposited on the cathode. The gold remains in form of slime in the anode bags. Every eight hours each cathode is lifted and the silver deposit is scraped off.

The current density is seven amperes per square foot.

The solution is frequently renewed. The gold content in the cathode silver is carefully watched and not permitted to exceed 2 parts in 100,000.

The gold refining room is shown in Fig. 3. Here the Wohlwill process is used. The cells are of Royal Berlin porcelain. The electrolyte is a gold chloride solution with some free hydrochloric acid. Pulsating current is used. The current density is 70 effective amperes per square foot.

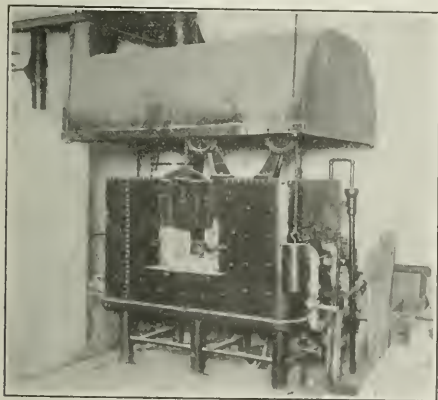


FIG. 6.—REVERBERATORY MELTING FURNACE

The platinum and palladium recovered in this process are sold to the highest bidder.

Both gas and oil are used for melting purposes in the New York Assay Office, as a matter of precaution, to make continuous operation absolutely sure. The whole furnace equipment of the Assay Office has been supplied by the W. S. Rockwell Company of New York City, who designed several special furnaces for the special requirements of the Office.

Fig. 4 shows the room for the melting of the gold in crucible furnaces, Fig. 6 a reverberatory furnace, for special uses, like

the cupelling of base bullion which contains too much base metals to permit the application of the ordinary refining process.

Fig. 7 shows the testing laboratory where the different



FIG. 7.—TESTING LABORATORY

solutions are tested; Fig. 8, the office of the melting and refining department and the make-up room.

Fig 5 shows the sweeps department which has recently been installed in the basement. The equipment consists of a jaw crusher which crushes the sweeps to one-half inch or less in



FIG. 8.—OFFICE OF THE MELTING AND REFINING DEPARTMENT AND MAKE-UP ROOM

diameter, a grinding mill of the Chilean mill type, which grinds the sweeps under water so as to pass a 60-mesh screen to two settling tanks, and a steam dryer.

Homogeneous Lead-Lining and Lead-Nickel-Lining.—

We have received an interesting illustrated circular on this subject from Mr. Bartholomew Viola, 140 Liberty Street, New York, the American representative of Kuhnle, Kopp & Kausch. The "homogeneous lead-lining" is recommended particularly for vacuum appliances, for cookers, and for all lead-lined vessels, working under pressure. The lining is claimed to be so intimately adhering that the two metals are inseparable. Another interesting novelty of this firm are wrought iron and copper vessels with a lead-nickel lining, as a substitute for pure nickel vessels. The joining layer of lead is stated to combine the metals so intimately that they cannot become loose by heat or high pressure. The nickel coating is only a few millimeters thick so that the cost is not excessive.

A New High-Resistance Thermo-Electric Pyrometer

In the deservedly widespread use of the thermo-electric pyrometer the desirability of having the resistance of the millivoltmeter comparatively high has long been recognized, but there have been difficulties in producing such an instrument at a comparatively low cost.

For this reason low-resistance thermo-electric pyrometers have been very generally used, the resistance of the millivoltmeters varying between 2 to 5 ohms. With this type of instrument it is absolutely essential that the instrument be calibrated, together with the leads or connecting wires which

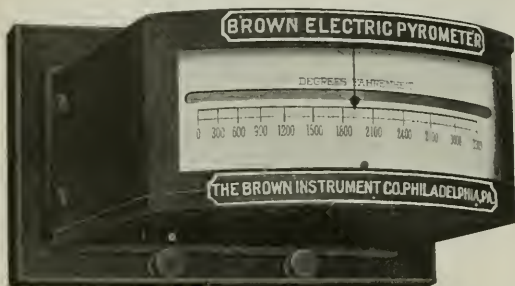


FIG. 1.—HIGH-RESISTANCE THERMO-ELECTRIC PYROMETER—INDICATING TYPE

are to be used, since any change in the length of these leads or any change in the length of the thermocouple furnished with the instrument will alter the true indication of the pyrometer so that serious errors will result. A change in the length of leads of 1 to 50 ft. is sufficient to make the average low-resistance pyrometer read 25° or more low and a change in the length of the thermocouple from 3 ft. to 4 ft. will make the same instrument read 20° more in error. For fairly accurate results, therefore, it is necessary to use the thermocouple and leads which are furnished with the instrument, or exact duplicates thereof—as long as a low-resistance millivoltmeter is employed.

It is true that for some time high-resistance galvanometers or millivoltmeters, portable in form, but not of the switchboard or wall pattern, have been produced abroad. They are, however, usually of the suspension type; that is, the moving element

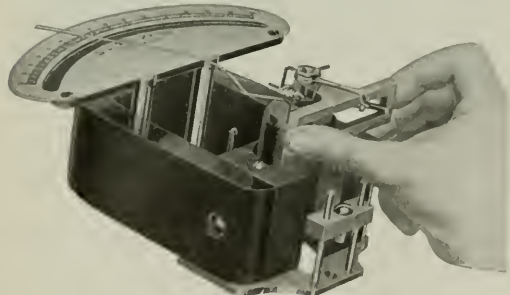


FIG. 2.—INTERIOR CONSTRUCTION

is suspended from the end of a fine wire or metal strip. Aside from supporting the moving element the suspension strip has another function, and that is to restore the moving coil or element to its zero position. It, therefore, has to be comparatively long and also small in cross-section. It is also necessary before using the instrument to carefully level it.

The advantages, however, to be gained from a properly designed and constructed voltmeter of high resistance are very marked and some of them are as follows: Any change in the

length of leads connecting the thermocouple with the millivoltmeter from 1 to 1000 ft. will not change the indications of the instrument as much as 5°; nor will any change in the temperature of these leads introduce any error in the indications of the instrument; neither will the length of thermocouple, the size of the elements forming the thermocouple nor the length of the thermocouple inserted in the furnace change the indications of the instrument. All of these advantages are due entirely to the extremely high ratio of the resistance of the millivoltmeter in relation to the resistance of the thermocouple and leads.

The Brown Instrument Company, of Philadelphia, who have done so much in the past for pyrometry in this country, have for the past two years been conducting exhaustive tests to design an accurate, high-resistance, thermo-electric pyrometer. The result has been an instrument of the pivoted type; that is, a type in which the moving element or coil is not suspended from the end of wire or metal strip, but is absolutely supported by means of small hardened steel pivots in sapphire jewel bearings. The moving coil and pole pieces of the magnet have been so changed in design as to produce a magnetic flux of great intensity in the air gap in which the coil swings. This high-resistance instrument, having a resistance of 100 ohms, can be furnished in either the indicating type or the recording type.

In Fig. 1 is shown the standard type of indicating instrument of the high-resistance type made in the horizontal type so as to afford a long scale for easy reading. This type of instrument has a mirror under the scale so that absolute accuracy in readings can be secured.

In Fig. 2 is shown the interior construction of the high-resistance system showing the redesigned pole pieces and small moving element, the moving element having been lightened to a minimum.

The Brown Instrument Company has been able to design this system in such a manner that its cost is practically no greater than the cost of manufacturing the low-resistance instrument. The instrument can be used for either base metal or platinum thermocouples as desired and is recommended by the manufacturer for either type of couple. The same type of system is also used in connection with recording pyrometers.

Personal

Mr. Max F. Abbé, who has been engaged in the pulverizing and grinding machinery business for a great many years, is in Europe. The Abbé Engineering Company, 220 Broadway, New York City, manufacturers of pulverizing and grinding machinery, inform us that Mr. Max F. Abbé is their vice-president and is looking after their foreign interests in Europe.

Mr. Maxwell W. Atwater has been retained by the Ohio Copper Company to investigate the flotation process with a view to adopting it in the company's mill at Lark, Utah.

Mr. Albert J. Bone will be superintendent of the new smelting plant of the Granby company at Anyox, B. C.

Mr. B. A. Bosqui is to have charge of the new cyanide mill of the Commonwealth Mining Company, Pearce, Ariz. The mill will be ready for operation shortly.

Mr. G. Montague Butler, formerly assistant professor of geology and mineralogy at the Colorado School of Mines, has been appointed head of the department of geology in the newly organized Oregon School of Mines at Corvallis.

Mr. A. H. Case, formerly manager for the Tennessee Copper Company, Copperhill, Tenn., has accepted a position as consulting engineer for Lewisohn Brothers, New York City.

Mr. John Dern, who has been identified with several mining enterprises in Utah, has been offered an appointment in the diplomatic service under the present administration.

Mr. George Flint, Treasurer of the Radium Company of America, Sellersville, Pa., sailed on the S.S. "Caronia" for Europe. Mr. Flint's address is care London City & Midland Bank, London, England.

Messrs. C. H. Fulton and J. Burns Read, of the Case School of Applied Science, Cleveland, O., have been making a Western tour.

Mr. Bancroft Gore has been appointed professor of metallurgy in the Montana School of Mines, Butte, Mont. Mr. Gore was previously on the engineering staff of the Washoe reduction works of the Anaconda Copper Mining Company, and later smelter superintendent at the works of the Gatico Copper Mining Company, in Chile.

Mr. Sidney J. Jennings, vice-president of the United States Smelting, Refining & Mining Company, recently visited the company's properties in California.

Mr. R. L. Lee has been appointed assistant superintendent of the copper smelting works of the Central Chile Copper Company, Panulicillo, Chile.

Mr. John L. Malm, metallurgical engineer of Denver, has returned home from an extended business trip in the South and East.

The Ray Consolidated Copper Company announces the removal of its executive offices to 25 Broad Street, New York City.

Prof. Heinrich Ries is engaged by the Mines Branch of the Canada Department of Mines to make an investigation of the clays of western Canada.

Professor Albert Sauveur, of Harvard University, has been awarded the Elliott Cresson medal of the Franklin Institute, Philadelphia, in recognition of his work in metallography.

Mr. George Schneider has been appointed director of the mining section of the Colorado Industrial Exposition which is to be held in the Grand Central Palace, New York City, during the month of November, 1913.

Mr. Frank E. Shepard, of the Denver Engineering Works Company, has been to Ishpeming, Mich., to inspect the new iron concentrating mill of the American-Boston Company. Professor Robert H. Richards also visited the mill at the same time.

Mr. Joseph T. Singewald, Jr., is making an investigation of the ores of the Silverton, Colo., district on behalf of U. S. Bureau of Mines. The results of the investigation are expected to be of assistance in devising suitable methods of ore treatment.

Professor Alexander Smith, head of the department of chemistry at Columbia University, has accepted a position as professor in Princeton University for the year 1914-1915.

Mr. Bradley Stoughton, secretary of the American Institute of Mining Engineers, was guest at a luncheon of the Colorado Section of the Institute on September 16. The local section was duly organized with the following officers: W. J. Cox, chairman; C. J. Moore, vice-chairman; C. L. Colburn, secretary-treasurer; Richard A. Parker and Thomas B. Stearns, members of the executive committee.

Mr. L. A. Test has resigned his position as assistant professor of chemistry in the School of Mines, Rolla, Mo.

Mr. Pope Yeatman has gone to Chile.

Mr. Lewis E. Young, who recently resigned the directorship of the Missonri School of Mines, will devote part of his time to teaching in the department of mining engineering, University of Illinois, and will also take up graduate work in the department of economics.

Digest of Electrochemical U. S. Patents

Prior to 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

REFINING METALS (Continued).

494,231, March 28, 1893, Charles B. Schoenmehl, of Waterbury, Conn., assignor of one-half to Alden M. Young, of same place.

Relates to refining metals using the same as secondary electrodes in a suitable electrolyte. For copper, an electrolyte of copper sulphate is used. The plates of crude metal are coated on one side with carbon or graphite, and the metal from the adjoining plate deposited upon the graphite coating. The patent refers to his copending application which became patent No. 494,232.

494,232, March 28, 1893, Charles B. Schoenmehl, of Waterbury, Conn., assignor of one-half to Alden M. Young, of same place.

Relates to apparatus for the refining of copper, etc. The crude metal is used as secondary electrodes. The vat contains supports for removable electrode holders, the holders serving also to maintain a uniform distance between all the plates, and also to support a diaphragm of a suitable fabric to protect the cathode side from floating impurities. The sides of the vat are perforated above the bottom, and communicate with parallel electrolyte vats, through which a constant circulation of electrolyte is maintained. The holes are so located as not to disturb any sediment at the bottom of the vat.

505,846, Oct. 3, 1893, Pierre DeP. Ricketts, of New York, N. Y.

Relates to a process of refining crude nickel, nickel alloys and other nickelliferous products. The product is first smelted or otherwise treated to reduce the nickel to the metallic state, and to remove most of the impurities. The metal thus obtained is rolled into suitable anodes and electrolyzed in an acid bath containing alkali salts, such as sulfuric acid containing sodium or ammonium sulfate, etc. During electrolysis the nickel forms a double sulfate with the alkali, which double sulfate is insoluble in the acid solution and precipitates to the bottom; it is subsequently collected and may be used for electroplating purposes, or it may be retorted; the nickel oxide thereby obtained is nearly pure and may be reduced by carbon. With nickel alloys as anodes, the remaining metal is either deposited during the electrolysis, as in the case of copper, or remains in solution, as in the case of iron.

514,275, Feb. 6, 1894, Lingan S. Randolph, of Baltimore, Md. Relates to the refining of copper or other metals by electrolysis in a cell in which the crude metal sheets are disposed horizontally and electrolyzed as secondary electrodes, the current entering the cell through the bottom electrode and passing out at the top. The plates are suitably separated by glass, etc., the electrolyte is circulated so as to keep the slimes in suspension and carried out of the tank where they may separate and the solution be filtered before returning to the cell. The anodes are preferably rolled or pressed, but may be cast.

514,276, Feb. 6, 1894, Pierre deP. Ricketts, of New York, N. Y.

Refers to his prior application, which became patent 505,846, of which this is a division, and relates to the method of separating nickel contained in nickelliferous bodies from the other metals and impurities combined therewith, consisting in immersing and dissolving the nickelliferous bodies in suitable acids; in adding thereto sulphates of ammonia or other similar ammoniacal salts, so as to form and retain a bath having an acid reaction of such composition as to dissolve the copper and nickel and retain the former in solution and form with the latter insoluble salts; and in causing an electric current to traverse the bath continuously from a suitable positive terminal to a suitable cathode, placed therein, whereby the copper is deposited upon the cathode and the nickel salts are precipitated to the bottom of the bath; in collecting the precipitated salts and dissolving the same in water, separating the insoluble impurities contained therein; evaporating the solution to dryness and heating the residue in a retort, whereby the ammoniacal products are collected and condensed for further use, and the nickel residue refined in any convenient manner.

515,763, March 6, 1894, Carl von Grabowski, of Eisleben, Germany.

Relates to the purification of sulfate electrolytes or other solutions containing a plurality of metals and arsenic and an-

timony. A sulfuric acid solution containing metallic sulfates is evaporated until it reaches a specific gravity of 52 deg. Baumé, the copper and other sulfates being crystallized out. The resulting solution is electrolyzed with a current density of 60 to 80 amp per square meter of anode surface using anodes and cathodes of lead or copper. Arsenic and antimony are electrodeposited and the solution entirely freed thereof. If the concentration exceeds 52 deg. Baumé, arsenic and antimony sulfides will be electrodeposited, the sulfuric acid being decomposed with liberation of hydrogen sulfid.

532,209, Jan. 8, 1895, Bernard Moebius, of New York, N. Y. Relates to a method and apparatus for refining metals, such as gold, silver, copper, bullion, etc. The apparatus consists of a horizontal tank in which is a traveling belt cathode preferably of thin metal, and above the cathode a series of removable trays having porous bottoms, the trays constituting anode compartments. The traveling cathode discharges its granular deposit upon a second traveling belt which removes it from the tank. The bullion to be separated is cast into anode plates and immersed in an electrolyte consisting of a strong solution of sodium or potassium nitrate sufficiently acidulated with nitric and sulfuric acid to keep some of the silver and all of the copper in solution. A current of small electromotive force is used, the silver being deposited upon the cathode in a loose, spongy state or in crystalline form. The copper is dissolved from the anode but remains in solution, while the gold and platinum metals are retained in the anode compartment, together with some lead peroxid, silver peroxid, and other impurities. Attention is directed to the improvement described in patent 592,097.

Book Reviews

Safety. By Wm. H. Tolman, Ph.D., and L. B. Kendall. Octavo (14 x 22 cm.), 422 pages, numerous illustrations; price 3.00 net. New York and London: Harper & Brothers.

This is an interesting and valuable book on methods for preventing occupational and other accidents and disease, the senior author being the Director of the American Museum of Safety. It discusses sources of danger and their removal in a dozen general types of works or factory, industrial hygiene in various occupations, and the social welfare of workmen in general. The information and advice are suitable and valuable to both employers and employees, while the general public will also find the book highly interesting and profitable reading.

General and Industrial Organic Chemistry. By Dr. Ettore Molinari, Professor at the Luigi Bocconi Commercial University, Milan. Translated by Thomas H. Pope of the University of Birmingham, England. Large octavo (15 x 25 cm.), 770 pages, 306 illustrations. Price, \$6 net. Philadelphia: P. Blakiston's Son & Co.

Dr. Molinari wrote also a companion volume on Inorganic Chemistry, which has been reviewed in the January, 1913, issue of this journal (Vol. XI, p. 61). The two volumes supplement each other and really constitute one work.

After a short introduction on the properties of organic compounds in general the treatise is divided into two parts; derivatives of methane and cyclic compounds. The former embrace the saturated hydrocarbons, halogen derivatives, alcohols, derivatives of alcohols, acids, derivatives of acids and aldehyde alcohols; the latter all other organic compounds. Under each individual substance there is given its history, properties, preparation in the laboratory, industrial preparation and commercial value and uses. The commercial relations, price, imports, exports and consumption, are a particular feature, intended to give the reader or student the proper idea of the commercial importance of the material.

The information is in general commendably up to date; one exception is the antiquated calculations of the temperatures produced by the burning of various explosives, which need

radical revision using modern values for specific heats of the gases produced. The calculated values given are impossible and make scientific calculations look ridiculous. The book, however, as a whole, is a most valuable and commendable work.

* * *

Iron and Steel. By O. F. Hudson, M.Sc., with a section on Corrosion by Guy D. Bengough, D.Sc. Octavo (14 x 22 cm.), 173 pages, 45 illustrations; price \$2.00 net. New York: D. Van Nostrand Co.

This short treatise is one of the series of "Outlines of Industrial Chemistry," edited by Dr. Bengough, and is intended to present to engineers, users of metals and students of metallurgy the more important principles of the metallurgy of iron and steel. The seventeen chapters of five to fifteen pages each (average length ten pages) deal with mechanical testing, production of pig iron, properties of cast iron, foundry practice, malleablized cast iron, wrought iron, crucible steel, cement steel, Bessemer process open-hearth and electric steel, mechanical treatment of steel, impurities in steel, study of iron-carbon alloys, heat treatment of steel, special steels, steel castings, welding, corrosion.

Inspection shows the various chapters to be well and correctly written, and to contain as much information as could be expected in such small space. The work, however, gives only the merest outline of a very large field, making it suitable for beginners only or for those desiring only a general review of the present status of the subject.

* * *

Das Materialprüfungswesen. Edited by Prof. Dr. F. W. Hinrichsen. 25 x 16 cm. (10 x 6½ inches), 607 pages, 215 illustrations. Price, bound in paper, 18 marks. Stuttgart: Ferdinand Enke.

Dr. Hinrichsen, who is connected with the Institute of Technology of Charlottenburg and the Testing Bureau of Berlin-Lichterfelde, has had the co-operation of Dr. Martens and twelve of his assistants of the Berlin Testing Bureau, and of Dr. Maffia of the Dresden Testing Laboratory, in producing this review in condensed form of the whole field of testing materials mechanically, physically and chemically.

This is an immense field to cover and necessitated restricting the treatment to the best methods now in use in each case. The work is taken up in the following order: Metals and Alloys, Ores, Pigments, Building Materials, Paper, Ink, Textiles, Fuels, Water, Fats and Oils, Rubber, Leather, Explosives, and each chapter or part of a chapter is handled by a specialist—in most cases by the best specialist in that line in Europe.

Under these circumstances the work is exceptionally valuable and accurate, and well worth getting by anyone interested in the general subject or in any one of the various materials discussed.

* * *

The D'Este Steam Engineers' Manual. With electrical appendix by Charles Penrose, E. E. Second Edition. 117+356 pages, with numerous illustrations. Price \$2 net. Boston, Mass.: Julian D'Este Company.

This manual which here appears in its second edition has been very considerably enlarged by the addition of an electrical appendix, prepared by Mr. Penrose. The steam section comprises 117 pages, the electrical appendix not less than 356 pages.

The book does not compete with specifically electric handbooks, but retains its character as a steam engineering handbook. In the electrical appendix the endeavor is therefore to emphasize those features of the practical side of electrical engineering which are of particular interest and importance to the steam engineer who has to do with electrical apparatus. The endeavor of Mr. Penrose not to shoot over the heads of his readers is very evident and we believe that practical steam engineers will find the book very useful. The old friends of the first edition should certainly become even better friends of the second edition.

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Some Aspects of the Half-Watt per C. P. Tungsten Lamp

The two papers by Dr. I. Langmuir published elsewhere in this issue on the new nitrogen-filled tungsten lamp mark a new milestone in the development of the electric incandescent lamp.

Oct. 21, 1879, is considered as the birthday of the electric incandescent lamp, since on that day Edison carbonized a piece of cotton sewing thread bent into a loop of horseshoe form and had it sealed in a glass globe from which the air was afterwards exhausted. Edison's genius solved the carbon-lamp problem so completely that for somewhat like twenty-five years the carbon lamp reigned supreme in incandescent lighting, until a little over ten years ago the first metallic-filament lamp appeared.

With the single exception of the Nernst lamp, which was an ingenious invention and a big thing in its day, but is now only of theoretical and historical interest, the whole development of the last ten years has been along the line of the metallic filament. The metallized carbon filament may first be mentioned which reduced the specific consumption of 3 watts per cp of the ordinary carbon lamp to 2.5 watts per cp. Then came the osmium lam, the tantalum lamp (2 watts per cp), the tungsten lamp (1.25 watts per cp), the ductile tungsten lamp (1 watt per cp), and now we have the nitrogen-filled tungsten lamp consuming but 0.5 watt per cp.

The general public is liable to look at this development as a triumph of electrical engineering. In reality it is step by step one continuous triumph of physico-chemical research. Just as in gas lighting and in electric-arc lighting the most notable advances—the incandescent gas mantle and the impregnated carbons of the flaming arc—have been the result of the work of the chemist, thus the evolution of the metallic-filament just sketched shows one pre-eminent leitmotiv of physico-chemical nature.

If we consider the light given out by an incandescent lamp as "pure temperature radiation," the radiation laws indicate that any increase in the operating temperature of the radiating filament must vastly increase the efficiency for two reasons: Firstly, because according to the Stephan-Boltzmann law the total radiation increases as the fourth power of the absolute temperature, and, secondly, because according to Wien's law of the displacement of the maximum in the energy spectrum, more of the radiation falls within the range of visible spectrum. Hence increase in temperature filament has been the leitmotiv.

Naturally the first limit to such increase is set by the melting point of the filament, and this consideration is clearly indicated in the historical development by the successive experimentation with metals of higher and higher melting points. Osmium was followed by tantalum and tungsten. Soon, however, it was recognized that before the fusion point could be reached, evaporation of

the metal, resulting in blackening of the globe, would set a limit to the operating temperature of the filament. And it is this limitation which has now been overcome by the nitrogen-filled tungsten lamp. Evaporation is held back or controlled within inoffensive channels and this makes it possible to increase again the operating temperature of the tungsten filament with such a gain of luminous efficiency that in spite of the loss by conduction of heat through the nitrogen atmosphere the specific consumption in watts per candle-power is cut in half.

The whole development has been international. In Europe Baron Auer von Welsbach and, in connection with the tantalum lamp, the chemists of Siemens & Halske have been pre-eminent among many active and able workers. But credit for the metallized-carbon lamp, for ductile tungsten, and for the nitrogen-filled tungsten lamp is due solely to the United States. Glorious work has been done by the Research Department of the General Electric Company under the genial leadership of Dr. W. R. Whitney. With his staff of able co-workers, men like Dr. W. D. Coolidge, he has directed extended researches on a strictly scientific basis, leading to the invention of ductile tungsten by Dr. C. G. Fink and the invention of the nitrogen-filled lamp by Dr. I. Langmuir.

Thus there should be glory enough for all. But that the new nitrogen-filled tungsten lamp is something great, is indicated in a rather amusing manner by the way how it has been announced in Europe. At the last convention of the German Association of Electrical Engineers Herr Salomon said: "Recently the Allgemeine Elektrizitäts-Gesellschaft has succeeded by the application of new means in making tungsten-filament lamps which have all other advantages of these lamps but consume only 0.5 watt per candle" (*Elektrotechn. Zeitschr.*, Aug. 14, 1913, page 955). Not one word about the inventor. Perhaps more diplomatic was the announcement in Great Britain "that the credit of evolving this lamp belonged to the British Thomson-Houston and associated companies, having been developed first at the laboratory of the Schenectady works by Mr. W. C. Whitney" (*London Electrical Engineering*, Aug. 28, page 487). Well, the compliment paid indirectly to physico-chemical research in the United States is genuine and sincere.

Cyanide from Beet-Sugar Waste

The removal of a $1\frac{1}{2}\%$ *ad valorem* protective tariff on imports of potassium cyanide directs attention to the status of cyanide manufacture in this country and abroad. The largest domestic manufacturer, combatting the proposal to place cyanide on the free list, stated that 5000 tons of sodium cyanide were manufactured in the United States in 1912, and urged the retention of the duty.

Chemical manufacturing in this country undoubtedly suffers by comparison with the much older and more highly developed industry in Europe, particularly in Germany. The German manufacture of alkali cyanide as a by-product from beet-sugar molasses is a case in point. It appears that beet-sugar molasses contains about 5% potash and 2% nitrogen. If the sucrose is precipitated and removed, the residue becomes a concentrate of potash and nitrogenous materials; and it is this residue, molasses *schlempe*,

which the Germans convert partly into alkali cyanide. The residue is distilled at about 400° C., yielding gaseous compounds of nitrogen which, on being heated to about 1000° C., synthesize ammonium cyanide. This is decomposed by sulphuric acid, forming ammonium sulphate and hydrocyanic acid, the latter being then absorbed in sodium hydroxide. The quantity of sodium cyanide thus produced annually in Germany approximates 5000 tons, which finds a market in South Africa.

We too have a large beet-sugar industry and might convert molasses partly into cyanides; but under our conditions the saccharine by-product finds use largely as a component of food for cattle and other stock. The case is only one of many which illustrate the difference in methods between a highly organized old industrial community on the one hand, and a new country rich in natural resources on the other. It would be a big mistake to conclude that what is industrially successful in the one would be *co ipso* economically promising in the other.

Leaching Copper Ores and Tailings

The great amount of experimental work going on in the West in the leaching of low-grade copper ores and tailings, calls attention to a marked change in the attitude of metallurgists toward such processes. Whereas, not many years ago, the hydrometallurgy of copper was regarded with suspicion, and the men who advocated it met with scornful criticism, today the tables are turned and some of the best talent in the country is engaged in this work, with reasonable prospects of success. The change is a consequence of technical progress, stimulated by the necessity of making the most of our resources; but in a broader light it is only one phase of the tendency toward a more general consideration of hydrometallurgical methods. With the realization of the limitations of current processes, attention is being directed toward chemical treatment in the wet way.

It is recognized that in the leaching of copper ores, no method of general application is likely to be evolved; and that each case presents a separate problem, varying with the material to be treated, the locality and the prevailing economic conditions. Leaching, of itself, is not a difficult feature; nor is the preparation of ore to make it amenable to leaching an insurmountable obstacle. Common salt and sulphuric acid are plentiful and cheap commodities. The latter can be produced as a by-product where ores are roasted on a large scale, so that with an increase in demand a plentiful supply can be assured.

Precipitation of the copper from solution appears to be the stumbling block and the point on which the different processes now under way are most widely divergent. Electricity, hydrogen sulphide, and iron are the agents thus far proposed and used. All three are in use in Montana: electricity at the Butte & Duluth and the Bullwhacker, hydrogen sulphide at Anaconda, and iron at the plants precipitating mine water. Electrolysis and precipitation on iron have been widely experimented with in Arizona. In the electrolytic work at one plant, the sulphur dioxide gas obtained from roasting the ore is bubbled through the electrolytic vats to prevent polarization.

Scrap iron is almost always suggested as a precipitant, but usually without thought of the difficulty of securing an

adequate supply in the isolated districts where much of the leaching is done. Further, the efficiency of precipitation and the quality of the precipitate are affected by the nature of the scrap, as Mr. Febles showed in his paper at the Butte meeting of the Institute. On the other hand, while the use of a suitable grade of pig iron is not commonly considered, it is not impracticable in certain localities. In the Southwest, for example, both domestic and imported pig iron can be bought for \$11 or \$12 per ton, and probably laid down at the plant for less than one per cent per pound. Compared with poor quality scrap at Butte for \$7 or \$8, and old rails at \$11 per ton, the use of pig iron might not be out of the question in certain localities.

Another phase of iron precipitation, however, is more interesting. Iron sponge, or metallized iron oxide, can be produced by the "step-reduction" of iron ore, as in the Jones process which was tried in Michigan. This reduction process has been applied to pyritic calcine, obtaining thereby spongy iron or iron dust, according to the fineness of the material. Experiments have shown this to be an excellent precipitant of copper. The advantageous feature of the method lies in the fact that sulphuric acid for leaching could be made from pyrite, and the resulting calcine reduced to iron sponge for precipitation of copper solutions. It might even be possible to roast chalcopryite, leaching the copper with the acid formed from the roaster gases, dry and reduce the iron oxide residue and use it as a precipitant. While this method of operation has been tried only in a small way, it is known that the investigators are hopeful of its success under suitable conditions.

The American Iron and Steel Institute.

Although it is four and a half years since the American Iron and Steel Institute was chartered, it is hardly two years since the organization began to secure papers of importance. As The Iron and Steel Institute—with the necessary accent upon the particle—is quite cosmopolitan in character and indeed numbers in its membership many American steel manufacturers and engineers, one is naturally eager to seek means for differentiating the two organizations. The Institute, which has its headquarters in England, in plain terms, can be held to "fill the bill" for the iron and steel industry of the world unless, or until, another organization shows a *raison d'être*.

The American Institute is doing well, for so young an organization, in defining a place for itself, and the papers presented at the semi-annual meeting, Oct. 24-25, in Chicago, make a fair showing when viewed in the general sense, and serve in particular to indicate that the American Institute has work of a somewhat different nature from that which the cosmopolitan Institute has done so well for many years.

Upon all subjects it is hazardous to generalize, but if the statement be not proved accurate it is at least suggestive that while the papers of the older Institute seem to be prepared largely by the men who study steel, the papers of the American Institute are prepared by those who make and sell steel. They deal less with molecular structures and more with how to get out the tonnage and how to adapt steel to new and specific uses. As to one point of differentiation a generalization is accurate, that the cosmo-

politan Institute never deals even remotely with commercial practices, its constitution forbidding, while the American Institute does. Three years ago a paper was presented, and discussed by the most prominent steel manufacturers in the country, on "Contract Obligations in the Steel Trade" and a year and a half ago a paper upon the same subject was given. The object was that contracts for the sale of finished steel products should be binding, instead of being mere options given by seller to buyer. The aim is to correct a crying evil which exists in the American trade but which is not found abroad. Unfortunately, no improvement can be traced directly to the airing given the subject before the Institute.

At the recent Chicago meeting another important commercial subject was discussed, that of "extras," several speakers insisting that the recognized extras represent an added cost of manufacture and are thus neither a bonus nor a source of extra profit, while some of the speakers specifically gave the advice that if a manufacturer desired to name a particularly attractive price he should do so by reducing his base price and not by conceding one iota of the extra. In any branch of manufacture it is recognized that the first step towards removing a difficulty must be to find and define its cause. Commerce, it appears, does not necessarily accept this principle, for one cannot find in these papers a clear statement of why extras are cut. In the case of many extras, and nearly all those which arise from the piece being of a particular cross-section, the extra cost upon which the card extra is based is a cost due to time lost in roll changing or to reduced output of the mill while actually rolling the material. When mills are running at less than full capacity this element of cost is reduced, possibly to insignificant proportions, and it is a recognized fact that it is when the market is dull and mills are not fully employed that extras are cut. This is the explanation. We do not for an instant imagine it is without the ken of those who read the papers at Chicago, our object being merely to point out that it escaped specific mention, whereas had the subject been mechanical, metallurgical or chemical, instead of commercial, the cause or the evil would have been stated as the foundation of the paper. Hence it may be questioned whether as much light can be thrown upon the problems of selling as upon the problems of manufacturing, by the reading of papers.

However, we must remember there was a time when papers upon manufacturing processes were rare because those who had knowledge preferred almost invariably to keep it to themselves. Their view broadened eventually. The reading of papers upon commercial subjects is in its infancy and in time the subjects may be handled with adequate precision and frankness.

With enterprise and directness characteristic of its locus, the American Iron and Steel Institute invites the user as well as the producer of steel to contribute to its meetings, and so at Chicago there was a paper on the steel car wheel by a producer, followed immediately by a paper from the superintendent of motive power of the Pennsylvania Lines West, giving the viewpoint of the user. Obviously this is practical, and bound to promote real progress, for after all the end of all endeavor with respect to steel is to make it render more and more service to man.

Readers' Views and Comments

"Hydrometallurgy—Joys of Its Theory, Woes of Its Practice"

To the Editor of Metallurgical & Chemical Engineering:

Sir:—In your October issue I note Mr. Warren F. Bleecker's review of my article, published under the above caption, in the September number. In Mr. Bleecker's comments, page 539, he draws the inference that I made specific allusion to a difficulty in the Malm process of dry chlorination.

The stoppage in question was not incidental to the Malm process, nor was the impurity calcium sulphate. So far as I personally am concerned the correction is of no value, but I wish to relieve Mr. Malm from the implication.

Golden, Colo.

REGIS CHAUVENET.

Slime Thickeners

To the Editor of Metallurgical & Chemical Engineering:

Sir:—I note in your issue of September, 1913, page 528, in the report of the very valuable paper of Mr. Hayden on Slime Concentration at Anaconda, that you quote the following:

"The Dorr tank has nearly the same settling capacity per square foot of settling area as the smaller unit," etc.

As compared with this information obtained from the Anaconda tests, we have advices from one of the largest gold-mining companies, which has installed a 44-ft. thickener to operate against 20-ft. settling tanks previously used, that they apparently get an increase of capacity per square foot of 5 to 10 per cent. We have also the statement of the superintendent of one of the largest copper companies in Nevada, who put in Dorr thickeners after operating small cones for a number of years, and who says:

"We are getting most excellent results from the thickener, using the overflow of two thickeners for wash-water on forty Wilfley tables and occasionally a surplus for use as wash-water on vanners. One thickener 17 ft. diameter by 9 ft. deep has a greater capacity than twelve 8-ft. cones, for the reason that there is only one unit to watch instead of twelve, without the constant result of feed to twelve cones becoming unbalanced. Also the 1¼-in. spigot used on the thickener requires no attention compared with the dozen ¾-in. spigots of the cones. We have never had a moment's trouble with the thickeners in any way since their installation."

It will be noted, of course, that the difference between the results obtained here and those at Anaconda is that the latter were compared with tests made on cones and not in regular mill operations.

JOHN V. N. DORR.

Denver, Colo.

High-Resistance Pyrometers

To the Editor of Metallurgical and Chemical Engineering:

Sir:—It is seldom that we have occasion to take exception to anything published in your journal, but the article on "a new high-resistance thermo-electric pyrometer," published on page 596 of your October issue, contains obvious errors.

Setting aside the fact that our company has been building high-resistance galvanometers of the pivoted type for more than four years, with a resistance averaging from 75 to 100 ohms, depending upon the range of temperature to be covered, the statement concerning the effect of the length of leads upon the accuracy of the instrument is obviously wrong. The resistance of 1000 ft. of No. 12 wire is 1.5 ohms, and the resistance of 1000 ft. of No. 14 wire is 2.5 ohms, hence the former would cause an error of 15 deg. at 1000° F., and the latter an error of 25° in the reading of the instruments at the same temperature, these errors being doubled at 2000° and tripled at 3000° F. If the ordinary acceptance of the wording is taken and a distance of 1000 ft. between the instrument and thermocouple is meant, the errors illustrated above would be

doubled, making a maximum error at 3000° F. of 90° for the No. 12 wire and 150° for the No. 14. For our "high-resistance" instruments we are contenting ourselves with the statement that a variation of from 50 to 100 ft. in distance makes no appreciable error in the reading.

The statement relative to the effect of a change in length of leads on the average low resistance instrument contains an even greater error. Assuming an instrument to have a total resistance of 5 ohms, including the length of the leads, the elimination of 50 ft. in distance of these leads would mean an elimination of 25 ohms, or 1/20th of the total resistance, causing an error in the reading of the instrument at 1000° of 50°, 100° at 2000° and 150° at 3000°. With an instrument having a total resistance of 100 ohms, a change of 100 ft. in length of leads would mean a variation in resistance of .5 ohm, causing an error at 1000° of 5°, 10° at 2000° and 15° at 3000°. In all these illustrations the nature of the error would be governed by the insertion or removal of the resistance in question, the insertion of resistance causing the instrument to read lower, the removal of resistance causing it to read high. For this reason the examples quoted above are not mathematically correct, as the exact percentage of error would depend on whether the leads were lengthened or shortened.

We also wish to state that we have in our laboratory a wall type pyrometer built by Siemens-Halske Co., having an internal resistance of 152 ohms and to our knowledge they have been building this type of instrument for a number of years.

D. C. DRAPER.

Thwing Instrument Company,
Philadelphia, Pa.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The article on a new high-resistance pyrometer, published on page 596 of your October issue, claims a new achievement in pyrometer apparatus.

In short, the article refers to a pyrometer indicator having an internal resistance of 100 ohms, the instrument being of the switchboard or wall pattern, and it is stated that this is a new achievement. Actually, the reader is given no opportunity to judge of the reasonableness of this claim because other factors are not mentioned. A galvanometer or millivoltmeter might have 100 ohms resistance for say, 50 millivolts, which is about the range of instruments generally used with base metal couples and the same instrument would have about 40 ohms when used with the range of only 20 millivolts, which is the approximate electromotive force of platinum-rhodium couples when used for their highest possible readings. Still further information as to what portion of the internal resistance is in copper and what in wire having negligible temperature co-efficient, would be necessary in order that one might intelligently pass on what has been achieved in an instrument of this general character.

But while that is highly important, it is not at all the point we wish to cover here and we refer to it to call your attention to the fact that the achievement represented in the instrument described in the above mentioned article is not proven by the necessary data in regard to the instrument itself but reliance for proving the point of new achievement rests entirely upon a statement in regard to what has been achieved or not achieved by others. This statement we now quote:

"It is true that for some time high-resistance galvanometers or millivoltmeters, portable in form, but not of the switchboard or wall pattern, have been produced abroad."

This statement does gross injustice, not simply to ourselves, but to many other makers of high-grade scientific apparatus. Pyrometry has been of invaluable assistance in metallurgical and other industrial work involving high temperatures.

Factors that are necessary for a high order of accuracy in temperature measurement are not known to all who wish to acquire that order of accuracy and the article we refer to is well adapted to mislead many of your readers.

We shall refute the statement on which that article mainly relies, by some data on our own instruments after here making clear that the statement does injustice to apparatus of some other makers as well as ourselves.

Our 1908 catalog illustrates and describes a wall-pattern high-resistance instrument, not of the suspended-coil type, not requiring to be set level; as well as a portable instrument of the same general type. Those instruments were furnished with 100 ohms for 20 millivolts with platinum-rhodium couples and when used with our base-metal couples they were furnished with 280 ohms internal resistance. These instruments were in every way a success and have been sold extensively for over five years in the face of much opposition by those who were not at that time making instruments anywhere near approaching such resistance, and were pointing out at that time that high resistance was not really so very necessary.

In the last five years we have been making pyrometers with base-metal couples and the resistance as high as 400 ohms in the indicator with 87 per cent of that resistance in wire of negligible temperature coefficient on the series coil and all of these instruments have been made in both portable form and the wall pattern or switchboard type. Yet according to the article above mentioned, that has not heretofore been done.

CHAS. H. WILSON.

Wilson-Macaulen Co., New York City.

Gold in Sea Water

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Professor Regis Chauvenet, in his very interesting article on Hydrometallurgy, on pages 486 to 491 of your September issue, has apparently taken someone else's word for some of his statements, without trying to substantiate them for himself. He says "for many years a paragraph had been floating around the world stating that sea water contains a definite proportion of gold. One metallurgist has taken the trouble to trace the statement to its source, with the result that the trail 'ran up a tree.' No authoritative statement of the contents of gold 'per ton of ocean' could be discovered. * * * Possibly there is some infinitesimal trace of gold there. Certainly there is not that much (1 grain per ton)."

If Professor Chauvenet will turn to page 111 of the second edition of F. W. Clarke's monograph, "The Data of Geochemistry," he will find the following: "The fact that sea water contains gold was first established by E. Sonstadt in 1872 [Chem. News, 25, 196, 231, 241, (1872); 74, 316 (1896); Am. Chemist, 3, 206, (1872)]. Its presence has since repeatedly been verified. In 1892 C. A. Munster [Jour. Soc. Chem. Ind., 11, 351, (1892) From Norck Tekn. Tidsskr.] examined water from the Kristiania Fjord, Norway, and found in it 5 to 6 milligrams of gold, with 19 to 20 of silver, per ton. In each analysis he used 100 liters of water. Liversidge [Proc. Roy. Soc. New So. Wales, 29, 335, 350 (1895)] found the gold in Australian waters to range from 0.5 to 1.0 grain per ton. At either rate, gold is present in the ocean in thousands of millions of tons. Liversidge [Jour. Chem. Soc., 71, 208 (1897)] also detected gold in kelp, rock salt, and a number of saline minerals, such as sylvine, kainite, carnallite and Chilean niter. In one sample of kelp he found 22 grains of gold per ton, and in a bittern, 5.08 grains. J. R. Don [Trans. Am. Inst. Min. Eng., 27, 615 (1897)] examined both ocean water and oceanic sediments. In the former he detected 0.071 grain of gold per metric ton, but the sediments were barren. In waters collected near the Bay of San Francisco, Wagoner [Trans. Am. Inst. Min. Eng., 31, 806 (1901)] found, also per metric ton, 11.1 milligrams of gold and 160.5 of silver. In deep sea dredgings Wagoner detected even larger quantities of both precious metals. [Trans. Am. Inst. Min. Eng., 38, 704 (1907)]. P. DeWilde [Arch. Sci. Phys. Nat., (IV), 79, 550 (1905)] and

A. Wiesler [Zeitschr. angew. Chem., p. 1795 (1906)] have published good summaries relative to the detection of gold in sea water, and also discussed the possibility of its economic recovery."

It appears then, that the trail that "ran up a tree" has run down again on the other side.

G. A. ROLSH

*Lehigh University,
South Bethlehem, Pa.*

Electric Zinc Smelting

To the Editor of the Metallurgical and Chemical Engineering:

SIR: In the August issue of METALLURGICAL AND CHEMICAL ENGINEERING a letter is published from Mr. F. L. Clerc commenting upon my article which appeared in the issue of November, 1912, entitled "Causes of the Practical Non-success of Electric Furnaces in Treating Zinc Ores."

Although my poor knowledge of English compels me to avail myself of translators who are liable to alter sometimes the meaning of what I intend to convey, I doubt whether I have put my name to an article in which it is stated that the production of carbonic acid in the actual electric furnaces is approximately equal to that of the retorts employed in the ordinary process.

On the contrary I have said (this journal, Vol. X, page 747): "This fact of the production of blue powder which does not take place in the ordinary retorts, or, if it does, in very small quantities, is due to the formation in the electric furnaces of a much higher percentage of carbon dioxide than in the usual retorts."

While I do not share all of Mr. Clerc's view, I agree with him as far as the influence of the cooling of zinc vapors, on the production of blue powder, is concerned. It is uncontested and a recognized fact that a too rapid or too intense cooling of these vapors gives rise to an excessive and sometimes total production of blue powder.

It is, in fact, this phenomenon which occurs in the electric furnaces when they are started, when care has not been taken to heat the condensers beforehand to the required temperature.

In such a case the phenomenon is even increased by the direct oxidation of the zinc vapors caused by the air contained in the condensers at the moment when it is set working.

The role played by the temperature in the condensation of the zinc vapors is well known to me and if I did not refer to it in the article on which Mr. Clerc comments it is that I considered this feature irrelevant to the subject I proposed to deal with.

In my article, in fact, I only intended to consider the present electric furnaces, their working and the manner in which they bring about the reduction of zinc ores. I left aside the condensation of the zinc which, after all, is only an accessory operation, very important no doubt, but which can and must be done easily if all necessary precautions are taken and if the reactions which release the zinc have been effected normally and without oxidation inside the furnace.

These indispensable precautions are principally the maintenance of the condensers at a temperature superior to that of the solidification of zinc (the most convenient appearing to be between 600° and 700° C.), and, as much as possible, the separation of the zinc vapors from other gases evolved by the reduction.

After this digression on the condensation problem, I regret to say that I am not at all of the opinion of Mr. Clerc as far as the reduction of zinc ores in electric furnaces is concerned.

Mr. Clerc seems to consider as a fairy tale the fact that the actual types supply a considerable proportion of blue powder. This is nevertheless so and all those who have experimented with electric furnaces agree in admitting that this notable proportion is produced, whatever thermal precautions may be taken to condense the zinc.

Mr. Clerc attributes the oxidation of zinc vapors in electric

furnaces solely to the presence of air entering the furnace by the open space around the electrodes. I also thought so previously, but having made experiments with a furnace in which this defect was eliminated I convinced myself that the oxidation was not due solely to the incriminated cause.

In the furnace I used (patent No. 989,169) the movable electrodes were fixed at the extremity of a round iron rod, which went through the furnace walls in an airtight joint in order to prevent absolutely the entrance of air. Notwithstanding said precaution, quite a notable amount of oxidation of zinc vapors took place, though of course in a lesser proportion than when ordinary electrodes are employed.

This led me to the conclusion that the proper working of the electric furnace was defective in itself and brought me to investigate the nature of the defects and their causes.

My investigations have been based on the following remarks: With arrangements as actually used, coal dust is found

Though Mr. Clerc is of the opinion that such an improvement is of difficult realization, it appears to me that the aim has been attained in the furnace hereunder described and shown by the attached drawings (Figs. 1 to 3).

This furnace, as indicated, is operated by three-phase currents, but may be adapted to the use of two-phase or single-phase current or of direct current. It is made up as follows:

A, heating and charging chamber.

B, reduction chamber separated from A by partition P, which is provided with a central opening.

C, crucible chamber.

E E, set of shunted electrodes which can be connected, one apart, by interruptors J with phase I of the electric current.

E' E', set of shunted electrodes which can be connected by interruptors J with phase 2.

The electrodes E and E' are not plunged in the charge, but simply placed in contact with it. The open space between the walls and the electrodes is filled up by an appropriate luting.

S, sole connected with phase 3.

J, interruptors to put in circuit, at will, the lateral electrodes E E'.

L, spout to draw the slag.

M, spout to draw the metals which are not volatile (lead, copper, etc.).

O, openings for the escape of gas and vapors.

R R', lateral chambers, their object being to facilitate the condensation: First, by separating by different densities the zinc vapors from the reduction gases; second, by lowering the temperature of the zinc vapors to an intermediary degree between that of production and that of condensation.

P G, partition and angle iron destined to facilitate the escape of the gases and vapors.

H, pipe leading the zinc vapors into condensers maintained at a temperature of 600-700° C.

It is easy to convince oneself that the whole charge contained in the reduction chamber B of this type of furnace is crossed in all directions by the electric current and that all of said charge is maintained practically at a uniform temperature. This is regulated as required by means of the interruptors J which modify the number or the position of the electrodes put in circuit.

The experiments carried out with a furnace of this type gave excellent results in every respect: thermic efficiency, yields of metal, etc., the production of blue powder being 2 per cent, more or less.

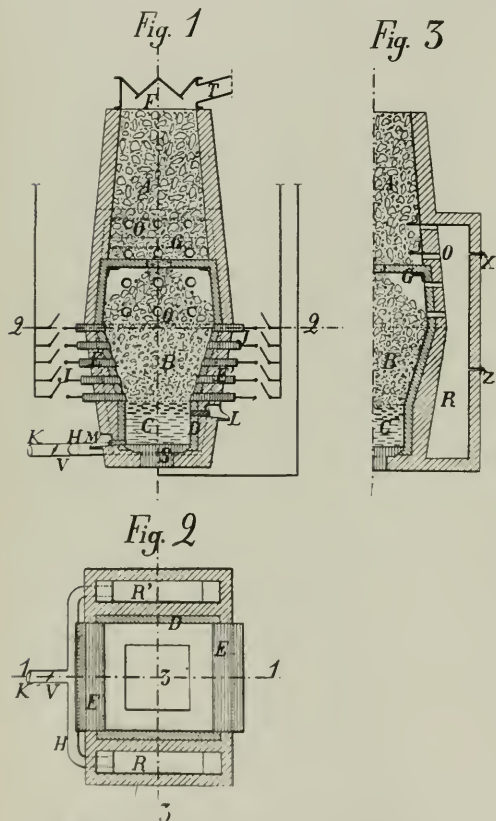
If it is taken into account that this furnace eliminates the very defects which I pointed out as the causes of the bad working of the actual electric furnaces, it remains evident that the remarkable results obtained are the best arguments in favor of my assertions.

It is besides evident, at first sight, that the reduction should not be performed as quickly as possible, and that the minimum duration is limited by the obligation to obtain absolutely complete reactions. The experimental results are unanimous in demonstrating this fact. It is sufficient, in fact, in an electric furnace working under normal conditions to increase the density of the current to notice at once an increase in the proportion of the blue powder produced.

Mr. Clerc seems to admit that this excessive speed is advantageous because it reduces the losses in radiation and facilitates the condensation.

I am at a loss to see why the condensation should be rendered easier. On the contrary, it would seem that the production of too rapid reactions necessarily gives rise to the passage into the condensers, where their presence is pernicious, of all the gases resulting from these reactions.

As regards the profit realized by the losses in radiation, this is entirely imaginary. For this gain would be more than absorbed by the losses resulting therefrom, such as defective working, bad returns in metal, etc. Besides, even taking into account the losses only in radiation, I figure that on the basis of equal production, those of the ordinary electric types would



FIGS. 1 TO 3.—SECTIONS OF ELECTRIC ZINC SMELTING FURNACE.

even in the condensers, this could only be brought about by a strong outrush of gas and vapors which is caused by a too rapid reduction, and this in turn is the result of too high a temperature in a too limited space; there is inevitably an insufficient contact of the bodies in presence of one another. Therefore the reactions that should have existed between these bodies only take place in a very imperfect manner, and I was led to the conclusion that the defect stated was due, more particularly, to the great rapidity of the reduction and that to remedy the same it was necessary to provide the furnace with some means of considerably increasing the capacity of the active zone of reduction, while maintaining a moderate but sufficient temperature and as even as possible.

be at least equal, if not superior, to those of the above described furnace.

In conclusion, therefore, and notwithstanding the apparent value of Mr. Clerc's arguments, I consider myself justified in considering as correct the opinions given in my last article, not only because I consider them based on logical arguments, but more particularly because they are supported by practical experimental results.

These results can be verified by any one interested in doing so and it will afford me pleasure to furnish parties desirous of making new experiments, with a furnace of the described type, and all information they may like to solicit from me.

Like any discussion, it is to be hoped that this one will tend to bring some light on the electric treatment of zinc ores, and if such result is obtained it will be fair to give the credit to Mr. Clerc for the large share he took in it.

Mexico City, Mexico.

F. LOUVRIER.

The Western Metallurgical Field

Minerals Separation Flotation in Colorado

The first use of the Minerals Separation flotation process in Colorado is now under way at two places in the state, on two different classes of material. Earlier attempts have been made to introduce flotation processes in Colorado, but without success; the ores were not amenable to the methods used, and the ventures seem now to have been ill-advised. It appears, however, that the Minerals Separation process is to be successful, as the first results obtained are promising, and further familiarity with the process will enable its operators to improve their work.

One of the flotation machines is at work at Leadville, on the re-treatment of an old zinc tailings dump. This material has been tested before by various concentration processes, but has resisted successful treatment. It requires grinding to liberate the zinc mineral and the difficulty of saving this fine mineral has proved a stumbling block. The Minerals Separation process, however, is adapted to the recovery of finely divided mineral and is apparently proving successful. It is possible that some coarse concentration can be done, but most of the ore has to be tube-milled before treatment.

The other application of flotation is at Ouray, where original mine ore is being treated. The material is a mixed sulphide, containing pyrite, blende, galena and silver. The minerals are brittle and entailed heavy losses in ordinary wet dressing. The first work was done on fine pulp classified from the stamp-mill product, the coarse being treated on tables as before. It is understood that the flotation machine was successful in handling this product, and that the final design of the concentrating system is now under way. The product of flotation probably will be a mixed sulphide concentrate, requiring further separation, but as there is an electrostatic separating mill at Ouray, the flotation concentrate can be handled satisfactorily.

In connection with the fine grinding required in some instances to secure liberation of the mineral, the question of smelting finely divided zinc concentrate arises. Zinc smelters do not desire an undue proportion of fine concentrate, and in some cases exact a penalty based on a screen analysis of the material. But if the use of flotation brings an increasing quantity of fine zinc concentrate into the market, the smelter may find means of handling the material satisfactorily. The lead smelter has met a similar problem in blast furnace smelting by the use of the sintering machine, and possibly the zinc smelter also will be able to adapt himself to changes in zinc concentration.

Chloridizing Base-Metal Ores in Utah

Within the past year two announcements have been made concerning the use of chloridizing processes in Utah, but no technical or operating details have been made public. Interest has attached to these ventures on account of the fact that successful operators had given them approval to the extent of building plants for their application. One of these plants is

at Park City, erected for the treatment of Ontario ore; the other is in course of construction in the Tintic district. The name of Niels C. Christensen, Jr., of Salt Lake City, has been connected with the development of the processes used at both places.

A possible hint as to the nature of the latter process is contained in specifications of a patent granted on October 7 to Mr. Christensen, in which he states that "the first 100-ton unit of a large plant is now in course of erection at Silver City, Utah, for the treatment of the ores of that district by this method." Disregarding for the time the apparatus by which the process is applied, details of which appear under Recent Metallurgical and Chemical Patents in the present issue, the steps in the proposed process are as follows:

The ore is mixed with the proper proportion of sodium chloride and pyrite, to give the chloridizing reactions, and ground to the proper fineness. It is then moistened with "mill solution," which is an approximately saturated solution of salts, NaCl, MgCl₂ and CaCl₂, and some HCl. It may be that no addition of sodium chloride to the ore will be necessary, as the saturated solution of chloride salts will carry sufficient chlorine to give the proper reaction. The per cent of solution will vary from 6 to 15 per cent. The moistened ore is then charged in a uniform layer from 3 to 18 inches thick on the revolving hearth of a furnace, where it is successively roasted, cooled and discharged. The hearth is perforated, and the ore charge is subjected to the action of hot air blown through the perforations.

The chloridized ore is then agitated with mill solution, and enough bleaching powder to dissolve the gold. The general result is that the gold is dissolved by the chlorine, the silver chloride by the other chlorides, the copper by the hydrochloric acid and the lead by the acid salt solution. This rich solution is filtered, and the residue properly washed, after which the metals are precipitated successively—the gold and silver on copper and copper and lead on iron.

The acid fumes from the roasting operations are recovered in a tower, being collected in some of the precipitated mill solution. Any volatilized metals also are thus recovered.

It is stated that numerous tests on different ores, both on a large and small scale, have given extractions as follows: gold, 80 to 95 per cent; silver, 80 to 96 per cent; copper, 75 to 98 per cent; lead, 80 to 90 per cent. The process appears to be a modification of the use of chlorine in extracting both base and precious metals from the same ore.

The Tariff on Ores and Metals

The passing of the Underwood tariff bill has finally shown the producers of ores and metals just what the import duties on their products will be for some time to come. With the new rates must come a period of readjustment to conditions, but it is not anticipated that any serious disturbances will result. The legal interpretation of some of the tariff items on ore must be awaited before we will know their exact meaning.

Following are some of the items of importance to western producers: Antimony, as regulus or metal, and matte containing antimony but not containing more than 10 per cent of lead, 10 per cent *ad valorem*; old rate on regulus and metal, 1½ cents per lb., and on matte 1 cent per lb. Antimony ore is on the free list, but only as to antimony content.

Copper in plates, sheets, rods, strips, pipes, and copper bottoms, etc., 5 per cent *ad valorem*. The old rate was 2½ cents per lb. Copper ore, regulus, cement, old copper fit only for remanufacture, and clippings from new copper, were placed on the free list.

Lead-bearing ores of all kinds, containing more than 3 per cent lead, 3½ cent per lb. on the contained lead. The old rate was 1½ cents per lb. Lead in bullion, pigs or bars, 25 per cent *ad valorem*; old rate 2½ cents per lb.

Zinc-bearing ores of all kinds, including calamine, 10 per cent *ad valorem* on the contained zinc. Under the old law these ores were free if containing less than 10 per cent zinc, and the rates were graduated up to 1 cent per lb. Zinc in

blocks, pigs or sheets, and zinc dust, 15 per cent *ad valorem*. Old rate, $1\frac{3}{4}$ to $1\frac{1}{2}$ cents per lb.

Nickel, its oxide, and alloys in which nickel is the constituent of chief value, in pigs, ingots, bars, rods and plates, 10 per cent *ad valorem*. The old rate was 6 cents per lb.

Quicksilver, 10 per cent *ad valorem*; old rate, 7 cents per lb.

The following are on the free list: Platinum, unmanufactured, or made into apparatus for chemical use; plumbago; tin ore or metal, provided that a duty of 4 cents per lb. shall be imposed when the United States is producing 1500 tons annually of ore or metal; tungsten-bearing ores of all kinds, the old rate being 10 per cent.

Smelter Items

Some delay has been encountered in the installation of the Hall sulphur recovery process at the Balaklala smelter of the First National Copper Company, in California, and it is unlikely that operation will begin much before the close of this year. The anticipated success of this process means much to California smelting, which has been seriously hampered by the farmers' injunctions against the operation of smelters which discharge noxious fumes. The Hall process will produce sulphur as a by-product, and thereby eliminate this element from the smelter gases.

The Washoe smelter of the Anaconda Copper Co. was closed for about ten days early in October, for the purpose of making general repairs and a clean-up of the furnace-flue connecting with the main attack. The concentrator continued in operation as usual, treating Butte second-class ores; first-class ores were sent to the smelter at Great Falls.

Recent improvements at the Garfield smelter, Utah, consist in an extension of slag-dumping area by the construction of a steel bridge over the railroad tracks north of the plant, and in the erection of a new 350-ft. stack to supplement the present one which is 300 ft. high. All reverberatory furnaces at Garfield are now equipped for oil-firing, and are reported to be running satisfactorily.

The Arizona Copper Company produced during the month of September, 900 tons of copper. The output was considerably curtailed on account of the unexpected delay in getting the new furnaces in working order.

Goldfield Consolidated

The final report of this company's operations in August, 1913, shows that the total production was 32,096 tons of ore, the net realization from which was \$198,784. The cost per ton of ore treated was as follows: Transportation, \$0.07; milling, \$1.85; marketing, \$0.05; general expense, \$0.03; construction, \$0.01; total, \$2.01.

The Non-Ferrous Metal Market

Dullness generally characterized the non-ferrous metal market in October. Demand has been light in spite of the tendency of producers to offer metal at lower prices. The copper producers are about the only ones to maintain asking prices in the face of no business, but the statistical position of that metal is such that buyers probably will come to sellers' terms.

Copper.—The situation in Lake copper remains unchanged and the quotations are practically nominal. Little business is being done, but producers seem content to await resumption of buying. Lake copper is quoted nominally at $16\frac{1}{4}$ to 17 cents; electrolytic at 16 to 16.10 cents.

Tin.—The domestic market has been dull and reduced prices have failed to stimulate buying. The London market also has shown a steady decline. October tin is quoted at 40 $\frac{3}{4}$ cents.

Lead.—This market has declined steadily throughout the month, following the action of the American Smelting & Refining Co. on October 1 in reducing its quotations. The last available quotations are 4.25 to 4.30 cents, St. Louis, and 4.40 to 4.45 cents, New York.

Spelter.—Toward the end of the month the concessions offered by sellers had the effect of stimulating some business

in this market, but the buying was not heavy. There is, however, a stronger undertone and a more hopeful feeling. St. Louis spelter is quoted at 5.10 to 5.15 cents, and New York at 5.25 to 5.30 cents.

Other Metals.—The Aluminium market has been dull and prices are slightly lower at 20 to 20 $\frac{1}{2}$ cents, New York. Antimony has been fairly active, with prices ranging from 6 $\frac{3}{4}$ to 7.70 cents per pound. Quicksilver prices also are slightly lower, with a fair business transacted. New York and San Francisco quotations are the same, \$38 per 75-lb. flask.

The Iron and Steel Market

October, usually a month of heavy buying, and of particularly large production through favorable weather conditions, failed entirely to live up to the precedents, as it witnessed a further decrease in buying, a more pronounced decline in prices and a diminished output of steel.

By the beginning of October a majority of the mills had reached a point at which the filling of old orders could not supplement the new orders currently received so as to maintain full production. The mills caught up with their orders and became able to ship on new orders and specifications within a few days. More strenuous competition for new business naturally resulted, with more active price cutting. A concrete evidence of the eagerness of the mills for business, even in very small orders is the fact that orders given to jobbers and to the distributing warehouses of certain mills, greatly decreased, because the buyers could purchase for direct shipment at regular mill prices.

Enough has now been developed to show clearly that the new tariff of October 4, 1913, is not directly the chief influence in the price decline. In few if any instances have mills lowered prices because the buyer put them in direct competition with foreign material. Prices are declining chiefly through the domestic mills being in competition with each other. One clear indication of this fact is that the price declines are as pronounced in the central west, which is altogether immune from foreign competition, as along the Atlantic seaboard, which is "exposed." To mention one commodity, the Pittsburgh and Philadelphia billet markets have always been independent of each other and the present experience is that billets have declined at least as much at Pittsburgh as at Philadelphia.

It is possible, however, that in a general way the tariff is responsible for decreased business activity in general, resulting in a smaller volume of demand, which in its turn causes a weakening in prices, so that indirectly the tariff is responsible for the present state of the steel market. The point to be emphasized is that at the moment there is not in progress a readjustment of prices with the cost of foreign material as the moving influence. Such buying as occurs is almost wholly for immediate shipment of small tonnages, involving orders that could not be placed abroad were the price ever so attractive. As the present readjustment proceeds a price level will probably be developed at which even large orders for extended delivery could not be placed abroad with advantage, except possibly in the case of a few commodities and in a few localities, to which foreign freights are low and freights from Pittsburgh are high.

On the basis of full rated capacity, which naturally can only be attained occasionally, under exceptionally favorable conditions, steel output has been approximately as follows: In the first half of the year, 95 to 100 per cent; in July, August and September, 90 to 95 per cent; on October 1, 90 per cent; on November 1, 80 per cent. A drop to 75 per cent by the end of November is far from improbable.

Opinions vary as to the probable duration of the present lull. The most hopeful view to be found is that a definite improvement will occur in three months, or by the beginning of February, while estimates run up to nine months. A factor to which the less hopeful probably do not give sufficient weight is that the country's steel-making capacity is relatively small considering the growth of domestic demand and the export trade. Additions to

capacity have been small in the past four years and very little new construction is now in progress. In previous depressions the common thought has been that time must be allowed for demand to catch up with capacity, but to-day no one can assert that there is an excess of capacity over normal requirements. Apart from all other considerations, the remarkable manner in which the mills were able to maintain substantially full production up to October 1, in face of extreme pessimism on the part of buyers and declining business activity generally, is evidence that the trade will pick up with unusual quickness.

Pig Iron.

Our review of a month ago noted that the fore part of September had shown a slight advancing tendency in pig iron prices, the latter part of the month showing a stagnant and stationary market. In October the condition developed into a definite declining tendency. Actual declines were confined to the Philadelphia and Valley markets, Buffalo, Cleveland and Chicago showing no change, while the southern market certainly lost tone even though it did not openly decline. During the first six months of the year pig iron declined, and by an average altogether of about \$3.00 a ton, and while the advance to October 1 did not average much more than 25 cents a ton enough occurred to show that a definite upturn had commenced. This movement has had to yield to the generally unfavorable conditions, and particularly the declining tendency in steel products. The turnover in October was very light. Buyers' yards are bare and they have relatively little iron due on contract, while the merchant production has been greatly restricted and it is practically impossible for any important decline to occur. The market stands as follows: No. 2 foundry, f.o.b. Birmingham, \$11.50; No. 2X, delivered Philadelphia, \$15.75; No. 2X, f.o.b. Buffalo furnaces, \$14; No. 2 foundry, delivered Cleveland, \$14.75; No. 2 foundry, f.o.b. Chicago furnaces, \$15; at valley furnaces (60 cents higher delivered Pittsburgh) Bessemer, \$15.00; basic, \$14; malleable, \$14; No. 2 foundry, \$13.75; forge, \$13.50. Ferromanganese is \$50. Baltimore, for prompt or forward, the price fixed by the English pool. German is sometimes offered at 50 cents less, but buyers frequently take the English material despite such a differential.

Steel.

As consumption of billets and sheet bars has decreased the ordinary contracts of consumers would easily cover their requirements, furnishing no occasion for much inquiry in the open market. As mills would sell at lower prices, however, each producer having a contract must protect it by adjusting the contract price or suffer the buyer to go elsewhere. The condition has led to wholesale readjustments, but as such readjustments can be made very quietly it is difficult to determine just what has been done, and the market can be quoted only approximately. It is quite certain that a fairly good buyer would not have to pay more than \$22.50 for billets, \$23 for sheet bars or \$26 for rods, f.o.b. maker's mill, Pittsburgh or Youngstown, for either Bessemer or open-hearth. These quotations average about \$1.50 below the nominal quotations mentioned a month ago.

Finished Steel.

In nearly all products the market has been a wide open one for several months, with prices on any given date varying according to the attractiveness of the order. During October the ordinary market has declined \$2 a ton on tin plates, structural shapes and plates, and \$1 a ton on wire products and sheets. Bars have not openly declined, but the old quotation of 1.40 cents is obviously only nominal in the circumstances. The average decline of all finished steel products from the top point is now fully \$2 a ton. Ordinary open quotations, subject to shading in most cases, are as follows, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill except Colorado.

Plates, tank quality, 1.30 cents.

Shapes, 1.35 cents.

Steel bars, 1.40 cents, base.

Iron bars, 1.55 cents, Pittsburgh; 1.25 cents, f.o.b. eastern Pennsylvania mill; 1.32½ cents, delivered Philadelphia; 1.20 cents, Chicago.

Wire nails, \$1.60, base; plain wire, 1.40 cents, base.

Sheets, blue annealed, 10 gage, 1.50 cents; black, 28 gage, 2.00 cents; galvanized, 28 gage, 3.05 cents; painted corrugated, 28 gage, 2.20 cents; galvanized corrugated, 28 gage, 3.10 cents.

Merchant steel pipe, 80 per cent off list for ¼ to 3-in. Steel boiler tubes, ¾ to 4½-in., 60 per cent off list.

Standard railroad spikes, 1.65 cents, Pittsburgh; 1.70 cents, Chicago.

Button head structural rivets, 1.85 cents.

Cone head boiler rivets, 1.95 cents.

Cold rolled shafting, 62 per cent off list, delivered.

Analyses of Anodes, Electrolyte, Wire Bar and Electrolytic Slime at the Great Falls Electrolytic Copper Refinery

In our September issue, p. 509, we gave an extended abstract of Mr. Willis T. Burns' excellent paper on the Great Falls Electrolytic Copper Refinery, presented at the Montana meeting of the American Institute of Mining Engineers.

To supplement the information given in our September issue, we give here the table of complete analyses of anodes, electrolyte, wire bar, and electrolytic slime, from Mr. Burns' paper. This table will be found exceedingly interesting, both by itself and in connection with the article in our September issue.

	Converter Anodes Per Cent	Electrolyte Per Cent	Wire Bar Per Cent	Electrolytic Slime Per Cent
Copper	99.1300	3.280	99.9500	43.3400
Arsenic	0.1183	0.500	0.0016	3.0300
Antimony	0.0534	0.041	0.0015	3.4600
Nickel	0.0420	0.377	0.0006	0.0800
Cobalt	0.0018	0.016	Trace	0.0000
Bismuth	0.0038	0.021	0.0004	0.1100
Iron	0.0110	0.600	0.0006	0.3640
Silver	0.1371	None	0.0030	17.1870
Gold	0.0008	None	Trace	0.1200
Selenium	0.0090	None		1.2000
Tellurium	0.0170	None		2.1000
Lead	0.0065	Trace	Trace	0.7600
Zinc	0.0035	0.418	0.0001	0.0900
Sulphur	0.2610		0.0025	13.2100
Oxygen			0.0350	
Silicon				0.1770
Chlorine		0.0040		0.0260
Carbon				0.5900
Platinum				0.000166
Free sulphuric acid		13.0300		
Specific gravity		1.220		

New York Section of the American Chemical Society

The first meeting of the season of the New York section of the American Chemical Society was held in the Chemists' Building on October 10th.

The chairman of the section, Dr. B. C. Hesse, in his opening address, spoke with much common sense of possibilities of changing the method of presentation of papers at section meetings so as to elicit more lively discussion. Professor F. J. Pond presented then a long paper giving a historical review of the pioneer work on the synthesis of rubber which was discussed by various speakers.

Joint Meeting on Metallurgy of Zinc

A joint meeting of the New York Sections of the American Institute of Mining Engineers and of the American Electrochemical Society will be held at the Engineering Societies' Building, 20 West 30th St., New York City, on Thursday, November 20, 1913, at 8:30 P. M. The subject of the evening is the metallurgy of zinc, and the following papers will be presented:

Progress in the Metallurgy of Zinc. By George C. Stone.

Electric Smelting of Zinc Ore. By W. R. Ingalls.

Electrolytic Refining of Zinc. By Victor Engelhardt.

"Following these papers there will be a joint discussion of the subject. As there is much conflicting opinion on the future of zinc, it is expected that there will be much disorder."

Foundrymen's Convention and American Institute of Metals

Report of Chicago Convention

The annual conventions of the American Foundrymen's Association and of the American Institute of Metals were held in Chicago from October 14 to 17.

Mr. Alfred E. Howell, Nashville, Tenn., is the vice-president of the American Foundrymen's Association; Dr. Richard Moldenke, Watchung, N. J., was reelected secretary-treasurer.

The vice-presidents elected are: Messrs. R. A. Bull, H. A. Carpenter, S. B. Chadsey, G. R. Lombard, T. L. Richmond, T. W. Sheriff, J. J. Wilson, and Walter Wood.

At the last session a silver-loving cup and salver was presented to Past President Seaman, the first president of the Association and president of the Seaman-Sleeth Company, of Pittsburgh.

The banquet was attended by more than 300 men. There was no speech-making, but music and a cabaret and much gaiety.

The exhibition of foundry equipment and machine tools again proved very attractive and was well attended.

Among the papers presented in the technical session were papers by W. S. Quigley on the use of powdered coal as fuel by F. T. Snyder on electric steel castings, by R. A. Bull on some difficulties in pouring steel castings, by E. L. Leasman on the annealing process for malleable castings by W. S. Hoyt on oxy-acetylene welding and cutting.

Notices and abstracts of some of them are reserved for a later issue.

American Institute of Metals

The Tuesday afternoon session of the American Institute of Metals was held jointly with the American Foundrymen's Association and was devoted to the presentation of a paper by Mr. M. W. Alexander on "an apprenticeship system in the metal industries," and a paper by Mr. C. E. Knoepfel on "how to make a time study."

APPRENTICESHIP SYSTEM

Mr. M. W. Alexander gave a very interesting outline of the scope and arrangement of the trade apprenticeship system of the General Electric Company and showed stereopticon views of the work done at Lynn, Mass.

The paper proved inspiring and was received with great interest. There is a general complaint of the difficulty of getting boys of any natural ability to enter the foundry trade. The need of an apprenticeship system for the metal industries is acknowledged to be very great.

HOW TO MAKE A TIME STUDY

Mr. C. E. Knoepfel, in the introduction of his paper, emphasized that the making of a time study is a most essential factor in any scheme of efficient management, and gave illustrations of the possibilities. The subject was then discussed under the following headings: Why time study is necessary; analysis of industry; the elements to be covered; variables of the man and the work; necessity for considering all factors; functions of time studies; tools needed; methods of making; studies as to classes; determining a fair standard; standardizing an operation; separation of inefficiency; detailed studies; making more than one study at a time; using regular watch with stop watch; correcting for errors; rest and fatigue; speeding the workers; practical uses of time study data; analysis sheet and schedule; organizing the work.

On account of the late hour, there was no discussion. That it was not on account of lack of interest, was proven by the Thursday session.

The attendance at this first meeting was somewhat disappointing. Both papers should have deserved a far larger audience. But many were kept away from the meeting proper by the exhibition and a desire to see men attending it.

REPORTS

The Wednesday session was opened by the presentation of the report of the official chemist, the report of the Bureau of Mines, and the report of the Bureau of Standards.

The report of the official chemists of the American Institute of Metals (Arthur D. Little, Inc.) was presented by Mr. C. F. Woods gave a concise review of the literature on metals and alloys, published during the past year. There was little discussion.

The report on the cooperative work of the Bureau of Mines with the American Institute of Metals was presented by Dr. Chas. L. Parsons. This comprises for the present the following four studies:

1. A comparative study of brass furnaces with particular reference to fuel efficiency and losses of metal in melting.
2. Study of the precautions to be taken for health and safety in the brass foundry.
3. The development of a suitable pyrometer for molten brass.
4. The development of an electric furnace for brass melting.

H. W. Gillett has completed an extensive investigation of problems 1 and 2 and his comprehensive report is to be published shortly in bulletin form. Work is going on with two promising types of electric furnaces for brass melting.

The discussion showed a vastly increased interest in the use of the electric furnace for brass and bronze. Mr. Wallace, of the National Cash Register Company, emphasized the crying need of a pyrometer for use in molten brass—that is, a works tool, not merely a laboratory instrument.

The report on the cooperative work of the Bureau of Standards with the American Institute of Metals was presented by Mr. G. H. Clamer. This work is on test bars and the report gives a discussion of the results obtained to date at the Pittsburgh branch of the Bureau of Standards on gun metal, various forms of test bars, cast in green and dry sand, flat and upright, with and without shrink heads and at different pouring temperatures. The rate of cooling has a great effect. A remarkable sag in the pouring temperature-tensile strength curve is reported at a pouring temperature of 1150 to 1160 deg. C.

INTERNAL STRAINS

A paper by Mr. J. E. Howard reported on the magnitude of internal strains set up in brass, copper, etc., by peening the surface or bending. Strains as high as 15,000 lb. per square inch were reported. Further the release of all or part of the strains by annealing was shown. All this work was done on rolled bars, as preliminary to the projected work on strains in castings.

BUREAU OF STANDARDS

A paper by Dr. G. K. Burgess on "the work in metals at the Bureau of Standards," was made quite interesting by some forty lantern slides of apparatus shown.

One of the most important functions of the metallurgical division will be the preparation of standard chemical samples, metals and ores, to secure a uniform product not only for chemical standards, but also for pyrometric and metallographic purposes.

Regarding the work on non-ferrous standard chemical samples, the chief chemist of the Bureau, Dr. W. F. Hillebrand, was quoted as follows:

"The Bureau will soon issue a rolled brass carefully analyzed like its iron, steels and ores, for the purpose of checking analysis and their analytical methods. The preparation of a cast brass is also contemplated and, in fact, a sample was prepared and nearly ready for analysis, when it was inadvertently spoiled in the process of mixing. This unfortunate accident emphasized most strongly one of the difficulties encountered in the preparation of satisfactory samples of materials of the nature of this particular brass and of all others that are subject to segregation in the casting. This brass, containing 5 per cent of lead, became very dark colored during the operation of mixing in the machine specially designed for the purpose of mixing metal chips, and it was found that the lead alone seemed to have undergone marked oxidation. The sample had to be discarded and will not be replaced until some method of casting is devised that will insure the requisite homogeneity without mixing. Possibly the method of comminuting directly from the molten mass, as is successfully employed for solders and

other similar alloys, can be applied to brasses also, but thus far success does not seem to have been attained."

"The experience already gained at the Bureau shows clearly how extremely difficult it is to secure perfectly homogeneous samples of several hundred pounds weight and also how difficult it is for co-operating analysts to obtain agreeing analyses of samples that are homogeneous. I would emphasize in this connection the futility of sending out for co-operative analysis test samples which are not known positively to be entirely homogeneous, or at least sufficiently so to justify the expectation that the results obtained by different analysts will really furnish a true criterion either of the skill of the several analysts or of the reliability of the methods used by them. Before they are sent out for such comparative test by different analysts, the precaution should be taken to have each sample carefully analyzed by one competent man and by the same method, in order to ascertain if they all represent material that is really of the same composition."

"Given a material of satisfactory character there remains the fact, shown clearly by the experience gained at the Bureau during the past few years, that the art of analysis is yet far from being established on a satisfactory basis. Analyses by supposedly very competent men often vary widely. It has become evident that this disagreement is often due to lack of knowledge of the limitations of the method used or to disregard of minor, yet important, details in the analytical procedures that apparently can be overcome only by a closer study of some of the analysts of the errors that are often involved in their methods, errors of which they are seemingly unaware, or which they are inclined to disregard as unimportant. Often they are unimportant from the point of view of ordinary industrial needs, but the fact remains that when it is necessary to determine the composition with a high degree of accuracy, the analysts are too often unable to or do not apply the refinements that are necessary for the attainment of really precise results. It seems hopeless to expect great improvement in this respect by leaving the methods of procedure to the judgment of each analyst, hence arises the need for a very careful study of the comparative value of different analytical methods for a given material and of the most precise specification of the entire procedure to be followed for the method or methods that shall be found most suitable."

THE NOMENCLATURE OF NON-FERROUS ALLOYS

A second paper, by Dr. **G. K. Burgess**, of the Bureau of Standards, emphasized the need of a systematic nomenclature of non-ferrous alloys. Tentative suggestions were made as to the system. Nothing final, however, is to be done until the matter has been thoroughly gone into by the Bureau of Standards in cooperation with committees from various societies.

The discussion showed not only the willingness but the desire of the membership at large to reform the nomenclature in a rational way.

BRASS FOUNDRY OF THE FUTURE

This was the subject of the last paper on the program for the Wednesday session, by Mr. **C. P. Kerr**, of the Bureau of Standards.

The trend of his paper is indicated by the concluding remarks:

"The success of the brass foundry of the future will depend more upon the training of the man in physical and metallurgical science and his ability to apply his knowledge, than upon the machinery employed, desirable though it may be as an accessory to facilitate the production of his castings. In other words, the greatest possible factor for success lies in the development of the man himself."

SCIENTIFIC MANAGEMENT IN THE FOUNDRY

While the attendance had been better at the Wednesday meeting than on Tuesday, it was best of all on Thursday, showing the broad interest of the members in the subject of scientific management.

Four papers on this subject were presented and elicited much discussion proving that the brass founders present are making much more rapid advance in scientific management than is gen-

erally thought. Not a single word of opposition to scientific management was heard. All seemed agreed as to the desirability of standardizing conditions, planning and dispatching, of time and motion study and of some form of efficiency rewards by which the worker is allowed to make as high wages as he can without fear of a cut in wages, the price having been scientifically set by time study.

The discussion centered on the best ways of making workmen realize that the management would not cut the wages, and on the best wage system to award efficiency.

The discussion brought out in this session was considered to be so important that it was voted that it should be edited into coherent form and printed as a pamphlet in a sufficient number of copies to allow sending it to all interested, whether new members of the American Institute of Metals or not, on application to the secretary. Mr. W. M. Corse, Lumen Bearing Company, Buffalo, N. Y.

The four papers which elicited this spirited discussion were the following:

The Efficiency Engineer in the Foundry, by **E. A. Barnes**.

Preparation for Scientific Management in our Plant, by **W. M. Corse**.

How Scientific Management Works in our Plant, by **C. B. Bohn**.

Core Room Efficiencies, by **O. F. Flumerfelt**.

The discussion of the last paper mentioned showed increased interest in handling this generally neglected branch of foundry work on a scientific basis.

BOILING OF METALS

A paper on this subject by Prof. **Joseph W. Richards**, of Lehigh University, was the first on the program of the last session, held on Friday.

The author compared the evaporation of metals with the evaporation of water. He gave a very simple relation between the vapor tension curves of water and mercury. This is that mercury always boils at 1.7 times as high an absolute temperature as water, when they are boiling at the same pressure, whatever that pressure may be.

A general application of the same principle to other metals is suggested. This is possible if one definite point on the vapor tension curve of another metal is known, that is, one temperature for which we know accurately the metal's vapor tension.

"When we are told, for instance, that zinc boils at 930 deg. C. (1203 deg. A.), at normal atmospheric pressure, and we know that mercury boils at the same pressure at 357 deg. C. (630 deg. A.), then we say that the ratio of their absolute boiling points for equal pressures is $1203 \div 630 = 1.91$, and we use this ratio to construct the whole vapor tension curve of zinc from the known vapor tension curve of mercury. Or, we could equally construct it from the vapor tension curve of water, using the constant temperature ratio $1203 \div 373 = 3.2$; it seems preferable to build on the mercury tension curve, because mercury is a metal, like zinc, and we feel a little more certain of our analogy when we compare the two metals."

The author also describes how to calculate the weight of metal vapor occupying a given volume at a given temperature.

PRODUCER GAS FOR BRASS MELTING

A long paper on "the application of producer gas to brass foundries," by Mr. **E. F. Bulmahn**, was next presented.

The following description of the equipment of the Langsenkamp-Wheeler brass plant, given at the end of the paper, is interesting.

Mr. Bulmahn's paper elicited considerable discussion. Five experiments now running in a yellow brass rolling mill indicate that producer-gas-fired furnaces give lower melting cost than coal or coke, but the metal losses are not reduced.

Rolling mill men present stated that they went through the producer gas game without success and are now looking to the electric furnace, since the mill conditions require an electric power plant anyhow and the added load comes cheaper. The reduction of the metal losses is the main point in favor of the electric furnace.

COMMERCIAL OPERATION OF THE HERING ELECTRIC FURNACE

Considerable discussion was also elicited by Mr. G. H. Clamer's paper on the Hering electric furnace (pinch effect furnace) in use in non-ferrous metallurgy.

The experience with a 20 to 30-kw. experimental furnace showed that about nine pounds of yellow brass are melted with one kilowatt hour. A large furnace is being erected which is expected to increase this figure to ten.

The power factor of the small furnace is 95 per cent, that of the large one 85 per cent or better.

During the development period much transformer trouble was experienced.

A new design of water-cooled electrodes is expected to minimize electrode troubles.

No difficulty is experienced in melting copper as well as brass, even though the resistance of copper is so much lower.

Three large Hering furnaces for brass melting are in course of construction.

SILVER PLATING AND SPOTTING-OUT

A paper by Professor C. F. Burgess and Mr. L. T. Richardson, of the Northern Chemical Engineering Laboratories, of Madison, Wis., deals with the trouble of "spotting-out" in silver plating. Attention is called to the fact that while potassium cyanide was formerly used almost universally by platers, sodium cyanide has been very generally substituted for it during recent years.

The authors produce evidence which is significant in that it suggests "a difference in properties as between potassium and sodium cyanides. The statements appearing frequently in the literature that sodium salts are less deliquescent than the potassium, and will for this reason cause less spotting-out, are evidently misleading.

"There is no doubt that the spotting-out problem is of sufficient importance to warrant even more careful investigation than has thus far been devoted to it and a part of this investigation should include a study of the relative influence of the sodium and potassium cyanides. We cannot base a definite conclusion upon the mere fact that pure sodium cyanide attracts moisture more readily than does the similar potassium salt.

"It is well known that during the operation of a plating bath the cyanides are gradually converted to carbonates. These carbonates may steadily accumulate in the bath, unless removed by the use of barium cyanide as a precipitating agent. Sodium carbonate is less deliquescent than is the potassium carbonate and we thus have a factor which may argue for the superiority of the sodium cyanide."

FLUXES FOR SOLDERS

A paper by Mr. W. Arthur dealt with "soldering fluxes for soft solder" and discussed the uses, advantages, and disadvantages of zinc chloride, an ammonium phosphate solution in water, lactic acid and ammonium lactate, resin either as powder or in alcoholic solution, citric acid solution in water, and ammonium chloride for fluxing purposes.

MELTING POINTS OF COPPER-ALLOYS

"The approximate melting points of some commercial copper alloys" was the subject of the last paper on the program, the authors being Messrs. H. W. Gillett and A. B. Norton. This is an investigation of the Bureau of Mines.

The chief results are given in Table I

There was some further discussion of the Bureau of Standards report (presented on Wednesday) on test bars by Mr. Skillman.

A committee was appointed to confer with crucible makers with a view of standardizing sizes and shapes so that all makes work in all furnaces and fit all tongs. The difference in crucible sizes changes the fuel space in coke or coal furnaces and affects fuel efficiency.

The need of a practical pyrometer for brass melting became anew the subject of discussion.

NEW OFFICERS

Mr. G. H. Clamer, Ajax Metal Company, Philadelphia, Pa., is the new president of the American Institute of Metals. Mr.

TABLE I

	Composition by Analysis				Composition by Analysis				Melting Point (Liquidus) C F
	Cu	Zn	Sn	Pb	Cu	Zn	Sn	Pb	
Gunmetal	88	2	10	...	88.4	1.9	9.7	3.0	995 1825
Leaded gunmetal	85½	2	9½	3	...	...	...	...	980 1795
Red brass	85	5	...	...	...	...	...	...	970 1780
Low-grade red brass	82	10	3	5	81.5	10.4	3.1	5.0	980 1795
Leaded bronze	80	...	10	10	...	...	...	...	945 1735
Bronze with zinc	85	5	10	...	84.6	5.0	10.4	...	980 1795
Half yellow, half red	75	20	2	3	75.0	20.0	2.0	3.0	920 1690
Cast yellow brass	67	31	...	2	66.9	30.8	...	2.3	895 1645
Naval brass	61½	37	1½	...	61.7	36.9	1.4	...	855 1570
Manganese bronze	...	...	...	...	...	...	...	...	870 1600

W. M. Corse, Lumen Bearing Co., Buffalo, N. Y., was re-elected secretary-treasurer.

The following gentlemen were elected vice-presidents: W. H. Bassett, F. O. Clements, C. H. Ivey, Robert Job, J. L. Jones, E. Weintraub, F. Moerl, P. Mueller, C. A. Finnegan, E. S. Fretz.

Mr. H. W. Gillett, who had been in charge of arranging the very interesting program of papers for this meeting, was re-appointed chairman of the Papers Committee.

Iron and Steel Meeting of the American Institute of Mining Engineers

A very successful meeting of the American Institute of Mining Engineers was held in New York City on October 16 and 17, 1913, under the auspices of the Iron and Steel Committee. It was opened by the president of the Institute, Mr. Rand. He introduced the new chairman of the Iron and Steel Committee, Dr. Sauveur, who presided at the morning sessions of Thursday and Friday, while Dr. Richards presided on Thursday afternoon, and Dr. Raymond on Friday afternoon.

The meeting was exceedingly well attended, about 200 members being registered, and the luncheons between the morning and afternoon sessions were much enjoyed as they gave an opportunity for social intercourse.

On Friday night there was a dinner at the Engineers' Club while the Iron and Steel Committee held its dinner on Thursday night.

The morning session on Thursday was devoted to the reading and discussion of papers by W. A. Forbes on blast furnace gas cleaning; by W. H. Blauvelt on the slagging producer; by F. Peter on the generation of steam by waste heat from furnaces; by G. C. Stone on the generation of steam by waste heat from regenerative furnaces and by W. R. Shimer on over-oxidation of steel.

In the afternoon session of Thursday there was first an interesting discussion on the use of powdered coal as fuel introduced by three papers by Richard K. Meade, H. R. Barnhurst and E. W. Shinn respectively.

Then followed papers on briquetting, one by E. Stütz on the scoria process and the other one by F. A. Vogel and A. M. Tweedy on the Schumacher process, while a paper by R. H. Lee deals with the use of nodulized ore in the blast furnace. The latter paper was discussed at some length by J. E. Johnson, Jr.

The morning session of Friday was devoted to a very interesting discussion on metallographic subjects, the authors of the two main papers being Professor H. M. Howe and Dr. G. K. Burgess respectively.

The afternoon session of Friday was devoted to the discussion and reading of papers by R. R. Abbott, on the influence of various elements on the absorption of carbon by steel; by J. H. Hall on shock tests of cast steel; by G. H. Clevenger and R. Ray on the influence of copper upon the physical properties of steel; by J. H. Hall on the life of crucible steel furnaces; by H. F. Miller, Jr., on a new design of regenerators for open-hearth furnaces, and by W. E. Ruder on grain growth in silicon steel. On the whole the papers elicited an excellent discussion.

On account of limitations of space it is impossible to give a detailed report of the proceedings, but a summary of some of the more important results brought out in the various papers and discussions is reserved for our next issue.

Tests Upon the Transmission of Heat in Vacuum Evaporators.

Further Tests by Prof. E. W. Kerr

The issue of October, 1913, of the *Journal of the American Society of Mechanical Engineers*, contains a paper by Prof. E. W. Kerr, of the Louisiana State University of Baton Rouge, La., entitled "Tests Upon the Transmission of Heat in Vacuum Evaporators."

The general arrangement and the scope of the tests described in this paper are the same as described in Professor Kerr's former extensive researches, an illustrated account of which was given in our June, 1913, issue, pages 333 to 338. However, Professor Kerr's paper contains much that is new.

Tests were made with five different types of calandria. The first three types are those marked *B*, *C*, *D*, in Figs. 2 to 5 of page 334 of our June issue. The fourth and fifth types are new.

The fourth type (double-tube calandria) was built by Mr. T. F. Sanborn, of New York City, and is shown in Fig. 1. It is an evaporator of the steam-tube type with special arrangements for removing the incondensable gases.

There are two thick tube plates, one above the other, both of cast iron. The heating tubes, which are 2 in. in diameter and 54 in. long, are expanded into the upper tube plate. These tubes are closed at the top and are open at the bottom. Into the bottom tube-plate $\frac{1}{8}$ in. tubes are screwed. These small tubes are open at both ends and are placed inside of the heating tubes, reaching nearly to the top of the latter.

The space below the lower tube plate is connected to vapor space of the succeeding body in an evaporator of commercial size. Steam passes up into the heating tubes in the annular spaces surrounding the gas tubes, the incondensable gases being driven toward the top, from whence they are removed by the small gas tubes, each heating tube having its own individual incondensable gas remover. The lower tube plate is made saucer-shaped so that the condensed steam drains to the center, from where it is removed in the usual manner.

The fifth type (baffle-plate calandria) was built by Mr. A. L. Weber, of New Orleans, La., and is similar to types *B* and *C* (page 334 of June issue), with tubes $1\frac{1}{4}$ in. in diameter and 24 in. long, but differs from them in the manner of distributing

the steam to the heating tubes, and in the manner of removing the incondensable gases, the steam being supplied through four openings, one above the other, thus giving better distribution vertically. (Fig. 2 on page 612.)

Between the two tube plates a vertical baffle-plate is placed so as to guide the steam along a circuitous path to the 3-in. downtake at the center. This baffle-plate is so placed that the passage for the steam is gradually reduced in cross-section in order to keep the steam velocity as high as possible overcoming to an extent the tendency to decrease the velocity due to condensation.

The incondensable gases are drawn off by means of a small perforated pipe through the top tube plate and reaching nearly to the bottom tube plate. The object of this design is to obtain high steam velocity among the heating tubes and effective separation of incondensable gases from steam.

That part of the evaporator above the top tube plate, which is called the vapor space, was 10 ft. high in all of the calandrias tested except that of Fig. 1, in which it was about 8 ft. This liberal height was provided in order to prevent, as far as possible, the carrying over of liquid in the vapors leaving the boiling surface.

As in Professor Kerr's former tests the object was to determine the effect of the following elements on evaporation: Effect of hydrostatic head, of temperature level, of incondensable gases, of density of liquid, and of type of calandria. These will now be considered consecutively.

Effect of Hydrostatic Head

Tests were made on four different calandrias in order to secure data on the effect of hydrostatic head. The results were plotted diagrammatically and for comparison the theoretical curves were also plotted according to calculations based upon the theoretical loss of temperature due to hydrostatic head. (The method of these theoretical calculations was given on page 334 of our June issue.)

Theoretically the coefficient of heat transmission decreases with increasing height of boiling liquid above lower tube plate. The results of the tests confirm this, but in all cases the loss in heat transmission due to hydrostatic head was considerably in excess of the theoretical, and the excess was greater for short tubes than for long tubes. The loss with varying heads, other conditions being equal, varies according to a straight-line formula.

The test curves seem to indicate that with tubes having the same length the rate of decrease in evaporation due to hydrostatic head is practically constant. But the curves for the 24-in. tubes are much steeper than those for the longer tubes (48 and 54 in.) showing that the effect of hydrostatic head is greater with short tubes than with longer ones. This is difficult to explain, although circulation doubtless has something to do with it.

Effect of Temperature Level

The total temperature fall in a multiple effect may be increased by increasing the temperature, or what is the same thing, increasing the pressure of the steam supplied to the first body, or by decreasing the pressure and temperature in the vapor space of the last body. It is evident that increasing the temperature fall, and therefore the capacity, by the first method results in increasing the average temperature in the heating compartments and that the second method results in decreasing it.

The relative advantages of high steam pressure in the first body as compared with high vacuum in the last body in obtaining increased heat transmission has been a matter of some question by many. Then, too, the reason for the inequality of temperature fall in the different bodies of a multiple effect, the greatest fall being always in the last body where the temperature level is lowest, is not definitely known, although it has been thought that the lowest steam density in the last body might be partly responsible.

In order to get data that would aid in settling these points a large number of tests were made. The results are given in tables and diagrams. The chief conclusion is as follows:

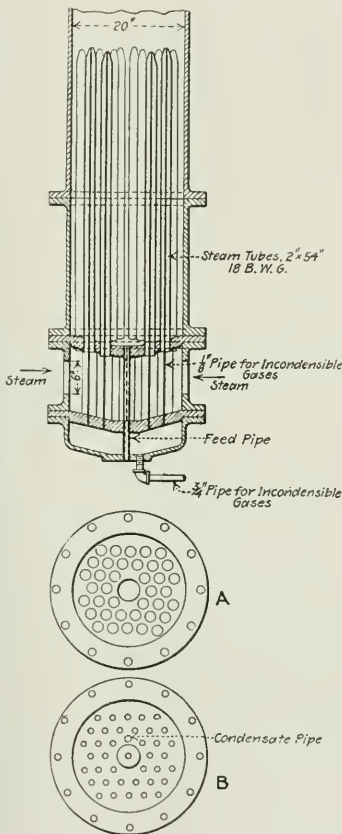


FIG. 1.—TYPE E DOUBLE-TUBE CALANDRIA

Other conditions being equal, the lower "the temperature level" the smaller the coefficient of heat transmission; in other words, the lower the temperature and density of the heating steam the smaller the coefficient. The coefficient U of heat transmission varies according to the equation

$$U = 2.25 + 17,500 D,$$

where D density of heating steam in pounds per cubic foot.

Effect of Incondensable Gases on Heat Transmission

The tests on this subject gave the following result:

Air or other incondensable gases in the heating steam greatly

reduce the heat transmission, even with relatively low vacua. The coefficient of heat transmission U varies according to the equation

$$U = C (P_s/P_t)^3$$

in which C is a constant, P_s the partial pressure of the steam and P_t the total pressure. The presence of "air-pockets" may be conveniently determined by means of thermometers in the steam compartment; good circulation and distribution of steam are important in preventing them.

In the equation for the curve

$$U' = (P_s/P_t)^n$$

the exponent n has a value of nearly 3, showing only a slightly greater decrease of the coefficient due to air than in the tests by George A. Orrok (*Transact. Am. Soc. M. E.*, Vol. 34, p. 713) on the subject of air in surface condensation in which the value of 2 was given to n . The absolute pressure in the evaporation tests was about 16.5 in. of mercury, whereas in the Orrok tests it was only about 2 in. These evaporator tests differed also from the condenser tests referred to as regards hydrostatic head and the range of the values of the coefficient of heat transmission. In the Orrok tests the maximum coefficient was only slightly above 300, while in the evaporator tests it was above 500. This latter was probably due to the greater steam density in the evaporator tests.

Attention has already been called to the fact that the evaporators shown in Figs. 1 and 2 were designed for the special purpose of overcoming the bad effects of incondensable gases. While it was practically impossible to determine the amount of air present, for the reason that the temperature existing adjacent to the heating surface could not be measured, the results obtained with these two arrangements show considerable improvement in heat transmission over those obtained from the others. The increase is doubtless due mainly to the more effective separation of air from the steam and to its prompt removal from the heating compartment.

Density of the Liquid

Increasing the density of the boiling liquid causes a loss in heat transmission due to the decrease in temperature fall. As the density is increased the boiling temperature increases according to the equation $y = C D^p$, in which C = a constant and D = density in deg. Frix. The total loss due to the density of the boiling liquid seems to be in excess of that due to loss of temperature fall. This is due to lower velocity of circulation.

Last Body of a Multiple Effect

The great temperature fall required in the last body of a multiple evaporator is due to the combined influence of greater amounts of air, steam of lower density, liquid of higher density, also, in many cases, more foul heating surfaces than in preceding bodies.

Type of Calandria

The five calandrias tested differ from each other in several fundamental features. Most important among these differences

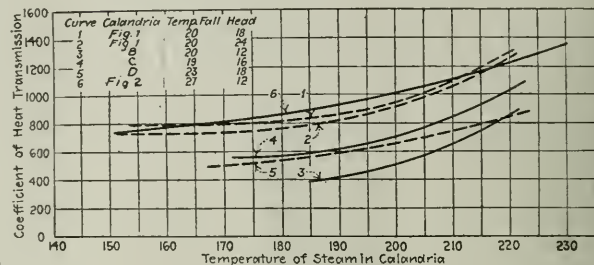


FIG. 3.—CURVES COMPARING HEAT TRANSMISSION IN DIFFERENT CALANDRIAS TESTED. CURVES 1 AND 2, CALANDRIA FIG. 1; CURVE 6, CALANDRIA FIG. 2; CURVES 3, 4, 5, CALANDRIAS B, C, D

of design may be mentioned proportions of tubes; that is, ratio of length to diameter, the use or non-use of downtakes to aid the circulation of the juice, and the methods employed for removing incondensable gases.

Comparative curves for the different calandrias are shown in Fig. 3. Curves 1 and 2 refer to the double-tube calandria shown in Fig. 1 in this article. Curves 3, 4, 5, refer to the standard types B, C, D, of calandria respectively, illustrated in our June issue, p. 334. Type C (curve 4 in Fig. 3) has a downtake, types B and D have not. Curve 6 refers to the baffle-plate calandria shown in Fig. 2 in this article.

Curves 3 and 5 show something as to the effect of varying the proportions of the tubes, the former for tubes 1½ in. by 24 in. and the latter for tubes 2 in. by 48 in., some advantage being shown for the long tubes, at least for low temperature levels. The greater heat transmission in the long tubes is probably due to better circulation. The greater the ratio of the length of a tube to its cross-sectional or carrying area, the greater will be the heat transfer per unit of carrying area, and this should increase the velocity of circulation.

A comparison of curves 3 and 4 shows a decided advantage for the downtake calandria over that without a downtake. The tests on the downtake calandria were made with a hydrostatic head of 16 in., whereas those on the other were made with a 12-in. head. Correction for this difference would slightly increase the advantage shown for the downtake calandria.

Curves 1 and 6 show the results of the tests on the two types of evaporators designed especially for efficient removal of incondensable gases. Both show coefficients of heat transmission considerably in excess of those obtained with the standard types represented by the curves 3, 4 and 5. Curve 6 for calandria of Fig. 2, the baffle-plate calandria with a head of 12 in., is only slightly higher than curve 1, which represents Fig. 1, the double-tube calandria with an 18-in. head. While the good results shown for these two calandrias may undoubtedly be attributed mainly to the more efficient separation of incondensable gases from steam and its removal, it is probable that the increased velocity of steam is also partly responsible.

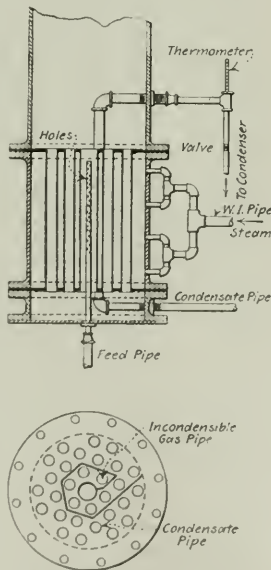


FIG. 2.—BAFFLE-PLATE CALANDRIA

The Chile Copper Co. has an immense deposit of copper ore at Chuquicamata which consists mainly of brochantite, $\text{CuSO}_4 \cdot 3\text{Cu}(\text{OH})_2$. The plan for treating this ore includes crushing to ½-in. size, leaching in vats with sulphuric acid, and electrolyzing the solution. The initial unit will have a capacity of 9000 tons per day.

0.5 Watt Per Candle Tungsten Lamps

Another Triumph of Physico-Chemical Research

At the meeting of the American Institute of Electrical Engineers, held in New York City, on October 10, 1913, two papers were presented on tungsten lamps of high efficiency. The first, by **Irving Langmuir**, dealt with "blackening of tungsten lamps and methods of preventing it." The second, by **Irving Langmuir** and **J. A. Orange**, with "nitrogen-filled lamps."

Both papers were presented by Dr. Langmuir. The presentation was accompanied by a series of most interesting experiments and demonstrations. These papers are printed in full in the October number of the *Proceedings* of the American Institute of Electrical Engineers, pages 1895 and 1915 respectively. We herewith give abstracts of both papers and comment on them on the editorial pages.

Blackening of Tungsten Lamps

The efficiency at which tungsten lamps may be profitably run is limited principally by the blackening of the bulb. It has usually been considered, especially among those most experienced in lamp manufacture, that the blackening of ordinary lamps was due very largely, if not entirely, to the presence of residual gases. The evidence which has led to this belief is discussed by Dr. Langmuir.

The sources of gases with the lamp were studied in detail. The principal gases are found to be water vapor, carbon dioxide, carbon monoxide, hydrogen, nitrogen and vapors of hydrocarbons. The specific effects produced by these and other gases were carefully determined by Dr. Langmuir. It was found that water vapor is the only one that produces perceptible blackening of the bulbs. The blackening by water vapor is due to a cyclic process in which the water oxidizes the tungsten and is itself reduced to atomic hydrogen. The tungsten oxide volatilizes and deposits on the bulb, where it is reduced by the atomic hydrogen to metallic tungsten and water vapor is again formed.

Attempts to materially improve the life of well exhausted lamps by the more complete removal of water vapor resulted, however, in failure. It is therefore concluded that, although water vapor is usually the cause of the short life of poorly exhausted lamps, yet it is not the cause of blackening in well exhausted lamps.

The real cause of blackening in well made lamps was proved to be evaporation of the filament, due to its temperature alone.

In order to test evaporation of the filament as the cause of the blackening, many experiments were undertaken to determine the rate of loss of weight of tungsten filaments when run at various temperatures in lamps. It was found that in lamps with filaments run at the same temperature the loss in weight was proportional to the surface of the filament and independent of the size of the bulb. The temperature coefficient of the rate of loss of weight was extremely high, as would be expected if it were proportional to the vapor pressure of the metal.

Furthermore, the actual measurements at various temperatures agreed remarkably well with the rational formula for vapor pressure

$$\log P = A - \frac{B}{T} - C \log T$$

From some simple considerations of the kinetic theory of gases, it became possible to calculate the actual vapor pressure of tungsten at various temperatures. It is of interest to give here simply the results at a few temperatures:

Efficiency, watts per candle	Temperature, absolute	Vapor Pressure, mm.
1.0	2400 deg. K.	0.000,000,05
0.4	2800	0.000,03
0.2 (melting-point)	3540	0.080
(boiling-point)	5200	760.

After it had been definitely settled that the blackening of tungsten lamps is caused by the evaporation of the filament a method of preventing the blackening is at once indicated.

This is the provision of a chemically inert gas in the bulb, as it will tend to reduce the evaporation of the filament. Most gases react chemically with tungsten at high temperature, but hydrogen, nitrogen, argon, and mercury vapor seem to be chemically inert towards it.

Experiments showed that the evaporation of the tungsten in an atmosphere of hydrogen, nitrogen, or mercury vapor was indeed much reduced. But further experiments were necessary to determine whether the decrease in evaporation was sufficient to offset the loss of heat by convection from the filament through the inert gas.

With hydrogen the loss of heat offsets the benefit of reduced evaporation. The reason is that the heat of conductivity of hydrogen at very high temperatures is abnormally great, due to the dissociation of the hydrogen into atoms.

With nitrogen and mercury vapor it is different. The experiments indicated that the heat loss by convection increases at high temperature rather slowly with increasing temperature in the case of nitrogen and mercury vapors, though very rapidly in the case of hydrogen as stated. Further, the heat loss from very small wires, say 0.001 in. in diameter, is not very greatly different from that from wires several times this diameter. In other words, it is much more nearly correct to say that the heat loss by convection from small wires is independent of the diameter, than to say that it varies proportionately with the diameter.

Tables were worked out which showed that the loss of efficiency (at constant temperature) due to the introduction of a gas at high (atmospheric) pressure is very much greater for filaments of small size, than for the larger ones, so that with wires of the sizes ordinarily used in lamps the temperature would have to be raised excessively in order to obtain an efficiency of even one watt per candle. Thus in nitrogen a filament of 0.0021 in. diameter (the size ordinarily used in a 20-watt, 110-volt lamp) would have to run at 3000 deg. to give one watt per candle. At this temperature of filament, the life of the ordinary lamp would be about 20 minutes, or about one fifteen-hundredth as long as that when running normally at one watt per candle in vacuum.

With filaments of larger diameter (0.005 in. and more), the loss of heat by convection is not nearly so serious, so that, if the rate of evaporation of the metal is very largely reduced by the presence of the gas, it is possible to raise the efficiency considerably without shortening the life.

The advantages of a large diameter filament can be practically obtained by coiling a smaller wire into a tightly wound helix or otherwise concentrating it into a small space.

Further experiments showed that the presence of the inert gas can be utilized in another manner. Not only does the gas decrease the rate of evaporation, but by proper design the course of convection gas currents can be directed along such channels that those parts of the bulb that are to transmit the light remain perfectly clear and any blackening deposit occurs on portions of the bulb where the blackening effect is of no account.

Nitrogen-Filled Lamps

In the second paper of which Irving Langmuir and J. A. Orange are joint authors, the authors state that by making use of the above principles they have been able to construct practical tungsten lamps which, starting at an efficiency of about 0.40 watt per candle, have run over two thousand hours, the average efficiency during life being better than 0.5 watt per candle. But such a degree of improvement has been reached only in lamps taking large currents.

Early experiments indicated the desirability of using a filament of large diameter. The larger filaments gave not only a better efficiency at any definite temperature, but also a much longer life. Thus doubling the diameter increased the efficiency from 0.05 to 0.56 and increased the life from ∞ to 300 hours.

The improvement in the efficiency is due to the relatively smaller heat loss by convection from thicker wires.

The life of the filament is determined largely by the loss of tungsten from the filament by evaporation and depends on the *relative* decrease in diameter caused by this evaporation. If the rate of evaporation per unit area from large and small wires were the same, the lives of various filaments run at a given temperature would be roughly proportional to their diameters. However, as the evaporation of tungsten in nitrogen is largely a diffusion process, it probably obeys laws similar to those of conduction or convection of heat from a wire: that is, for wires of small diameter, the actual amount of tungsten evaporated would be nearly independent of the size of the wire. The rate of evaporation *per unit area* would thus be approximately inversely proportional to the diameter. The relative lives of very small wires in nitrogen are therefore nearly proportional to the squares of their diameters.

It is, however, not desirable to use filaments of very large diameter if similar results can be obtained with smaller ones.

Unless very low voltages are used, the power consumed with the larger wires is so great that only very high candle-power lamps can be made.

Therefore, it was of vital importance to increase the effective diameter of the filament without decreasing its resistance, and

increased heat loss both by convection and radiation, and thus prevent local overheating or spotting.

The use of helically wound filaments increases the life of the lamp many times beyond the life that would be obtained with a straight filament running at the same efficiency. This is especially true of the smaller sizes of wire.

The authors then discuss the principles of the design of bulbs and locations of filaments.

In ordinary lamps about 20 per cent of the energy radiated from the filament is intercepted by the glass and causes heating of the bulb. In the nitrogen lamp, beside this radiated heat, there is an additional amount of heat carried to the bulb by convection—an amount varying with the type of lamp and ranging from 6 to 40 per cent of the total input. The convection currents carrying this relatively large amount of heat travel vertically upwards from the filament and strike a relatively small area of the bulb, which thus tends to become greatly overheated. Unless special precautions are taken, this overheating will cause the liberation of enough water vapor to cause attack of the filament and consequent blackening of the bulb. It is thus highly desirable, in ordinary cases, if small bulbs are to be used, that the filament should be placed in the lower part of the bulb. This has the further advantage that it allows sufficient surface of glass in the upper part for the deposition of the tungsten nitride.

For a similar reason it is generally desirable, although not necessary, to make the bulbs with their height considerably greater than their horizontal diameter.

By special design of the bulb, satisfactory lamps have been made with bulbs of only one-half to one-third as large a volume as that of evacuated lamps of the same wattage. This means that for bulbs of the same volume the nitrogen lamps give roughly from five to ten times the candle power of evacuated lamps. The bulbs of such lamps naturally run much hotter than those of ordinary lamps. The upper parts of the bulbs are often 100 to 200 deg. C. or more, while the lower parts are sometimes much cooler than this, although closer to the filament.

A few notes are given on lead-in wires and supports and it is mentioned that platinum has been discarded entirely, even in the smaller sizes. Mostly special alloys are used which have the same coefficient of expansion as the glass. Bulbs of special glasses into which tungsten or molybdenum wire can be sealed directly have also been used.

The principal limitations of the new type of lamp is that of current. There is no practical upper limit to the current.

For the particular type of nitrogen-filled lamp which has at present been furthest developed, it may be said that a life of over 1500 hours is obtained at efficiencies better than 0.50 watt per candle only in large units taking over ten amperes. Lamps running at 0.6 to 0.7 watt per candle have been made in units taking at least 5 amp.

A number of special types of nitrogen-filled lamps have been made as follows:

(1) *Large units of very high efficiency*, consuming 0.4 to 0.5 watt per candle with a life of 1500 hours or more. These take currents of 20 to 30 amp and (except in units over 4000 cp) are, therefore, best run from alternating-current circuits by means of small transformers or auto-transformers giving a voltage depending on the size of unit desired. Thus with 30 volts and 25 amp, the power would be 750 watts and this in a lamp of 0.45 watt per candle would give 1670 candle-power. Typical lamps are shown in Figs. 1 and 2.

(2) *Small units of low voltage* for street series lighting on 6.6-amp circuits (at 0.6 to 0.7 watt per candle), for stereopticon lamps and automobile headlights. These take currents of 10 amp or less at voltages as low as 4 or 5 volts. The specific power consumption with 1000-hour life ranges from 0.6 to 1.0, or even 1.25 watt per candle, according to the current used.

(3) *Lamps to run on standard lighting circuits (110 volts)*. Large units of this type (of several thousand candle-power) consume 0.5 watt per candle or less.

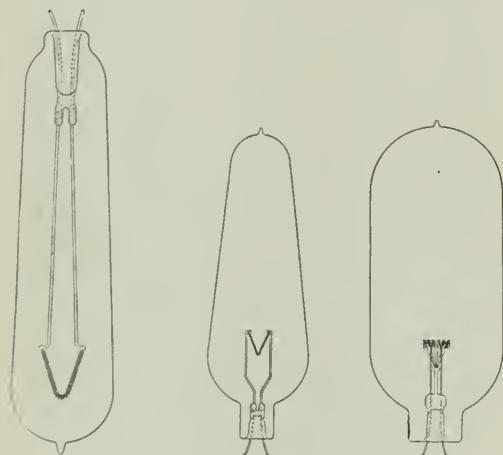


FIG. 1.—HIGH-EFFICIENCY NITROGEN-FILLED LAMP FOR LOW VOLTAGE CIRCUIT

FIG. 2.—HIGH-EFFICIENCY NITROGEN-FILLED LAMP FOR LOW VOLTAGE CIRCUIT

FIG. 3.—NITROGEN-FILLED LAMP FOR 110-VOLT CIRCUIT

various methods of doing this were tried. The method which has thus far proved most satisfactory is to wind the filament into the form of a tightly coiled helix.

The use of a helically wound filament presents several very interesting features. The life of ordinary single loop filaments is limited by the irregularities in diameter which develop after a considerable amount of tungsten has evaporated. These irregularities, after they first appear, tend to magnify themselves very rapidly, on account of the tendency for the current to overheat any spot which becomes thinner than the rest of the filament. The overheating increases the rate of evaporation and rapidly causes failure.

In the gas-filled lamps, however, when helically wound filaments are employed, a new factor is introduced which entirely counteracts this tendency to overheat in spots. If, during the life of the lamp, any part of the filament should, for any reason, evaporate more rapidly than the rest, so that the filament becomes somewhat thinner, this portion will have less mechanical strength than the rest and will therefore sag more rapidly. The helix will therefore open out wherever the filament becomes thin or becomes overheated. This will cause

Special advantages of the nitrogen-filled lamps are:

1. Much whiter color of the light, due to the fact that the temperature of the filament is 400 to 600 deg. higher than that of ordinary lamps.
2. High intrinsic brilliancy of the filament.
3. Practical constancy of ampere-volt-candle-power characteristics during life.

The figures for specific power consumption given above are watts per horizontal (international) candles measured in the direction perpendicular to the plane of the filament if this is in the form of a single loop. Careful measurements have shown that with helically wound filaments the distribution of light in a horizontal plane is almost perfectly uniform, so that the figures for specific power consumption may be considered to represent also watts per mean horizontal candle.

The ratio of mean spherical to maximum horizontal (practically mean horizontal also) candle-power averages about 84 per cent for the lamps made with single loops of helically wound iron.

An appendix is added on methods of photometry of nitrogen-filled lamps.

Notes on the Standard Apparatus and Method for Measuring the Amount and Character of Atmospheric Pollution.

By John B. C. Kershaw

Observations with the standard apparatus and method of measuring the soot and dust fall of towns and cities (which were fully described in an article by the present writer in the issue of METALLURGICAL AND CHEMICAL ENGINEERING for June, 1913) commenced in October last in the following sixteen towns and cities: London, Hamburg, Birmingham, Glasgow, Liverpool, Manchester, Leeds, Bradford, Crewe, Exeter, Hull, Malvern, Leicester, Newcastle, Plymouth and York. As Hamburg has agreed to adopt the standard form of apparatus and method of collection, it is hoped that other German towns and cities will follow its lead, and that all the European records of atmospheric pollution, wherever obtained, may in time be based upon one uniform method of collection and examination.

The Department of Industrial Research in the University of Pittsburgh has been engaged also for some time past in an investigation of the smoke problem in America, and it would be a fitting close to the labors of the staff engaged in this research, if they could initiate a similar system of observation of the soot- and dust-fall in the more important cities of the U. S. A. The need for the adoption of a uniform system of collecting and examining the character of atmospheric pollution is proved clearly by the fact that all the observations yet made in different towns and countries are worthless for comparative purposes, since each observer has used a different apparatus or method and has applied it under different conditions.

In order that all interested in the subject of atmospheric pollution may be kept acquainted with the work of the London Committee, notes will be published from time to time in this journal, recording the progress made. The regulations which have been issued by the Central Committee for exposure of the gauge and for the chemical examination of the collected water and solid matter are given below. The standard soot- and dust-gauge was fully described and illustrated in the June issue of METALLURGICAL AND CHEMICAL ENGINEERING, and it is only necessary to state here for the benefit of those who did not read the earlier article, that the chief feature of the gauge is an enameled iron funnel with a superficial area of 4 sq. feet.

The gauges are to be placed, if possible, on the ground level, in open spaces free from abnormal dust. The bottles containing the water and deposit are to be removed on the last day of each month, and are to be replaced by thoroughly cleaned empty bottles. Before removing the bottles the gauge vessel is to be washed down with some of the collected water, a brush and rubber squeeze of standard pattern being used to remove any

adherent matter. The chemical analysis of the water and deposit is to be made by a standard method (details of which are given below) and a report of the results obtained is to be sent monthly to the offices of the central committee.

The bottles properly labelled with the name and number of the station, and the period of collection, are to be kept standing in the laboratory for several days, until the insoluble matter has completely settled. The approximate volume of the water in litres should be ascertained at the time of the receipt of the bottles.

The undissolved matter is separated by filtration, the whole volume of collected water being used for this purpose.

The filtration is best effected in the following manner.—

A syphon-tube provided with a glass-tap or pinch-cock on the longer limb, is inserted into the bottle to within about one inch of the bottom. The bottle should stand at a height above the vessel in which the filtrate is to be collected, sufficient for an easy and adjustable discharge of the liquid. The filtering medium which has given the best results, consists of a layer of pulped filter-paper or of asbestos pulp suitably disposed in a Gooch-crucible. The Gooch-crucible should be fitted, by means of an adapter, in the rubber stopper of a graduated bottle of the same capacity as the station bottle. The rubber stopper of the second bottle is also provided with a tube leading to a vacuum pump. The filter is prepared with the pulp in the usual manner, and the filtration takes place under moderate suction. Should the filter become clogged, the pulp, together with any solids collected upon it, may be removed to a porcelain basin and kept for further treatment, while a new filter is made from fresh pulp.

After the clear portion of the liquid has been filtered, the bottom turbid portion may be shaken up and syphoned over into the Gooch-crucible,—the bottle is then washed out with some of the filtrate. Any pulp left over or previously removed, is finally put on top of the solid matter collected in the crucible. Should there be more than one station bottle, it is best to filter first the clear liquid of all, and then the portions containing the solids which have settled.

The Gooch-crucible, after its contents have been washed with distilled water, and dried at 105°C. is weighed; the gain in weight equals the *total insoluble matter*.

The crucible is then placed in an extraction apparatus, and the contents are extracted with carbon bi-sulphide. The loss, after drying, represents the *tarry matters*.

The residue in the crucible is then ignited in a current of air until all the carbonaceous matter is burned away. The residue equals the *ash or mineral constituents* of the insoluble matter of atmosphere.

In order to determine the amount of dissolved matter 250 cc. of the filtrate are evaporated to dryness on a water-bath; the residue is then dried and weighed—this gives the *total dissolved solids*.

The loss on gentle ignition gives the *organic matter soluble in water*; the residue represents the *soluble inorganic matter*.

For the determination of *sulphates* one liter of the filtrate is concentrated, and acidified with hydrochloric acid. The liquid is then boiled, Barium chloride is added, and the precipitated barium sulphate at the end of twelve hours, is filtered off—washed, dried and weighed.

The *chlorides* are estimated by titration of 500 cc. of the filtrate with N/10 silver nitrate solution, taking the usual necessary precautions to ensure the absence of carbon-dioxide (or carbonates) in solution.

The *ammonia* is determined by acidifying 1000 cc. of the filtrate with dilute sulphuric acid, and after concentrating the solution to 100 cc. and making strongly alkaline, distilling the residue into a measured amount of N/10 acid. The excess of acid is then determined by titration with N/10 alkali, and the amount of ammonia present is calculated from the acid which has been neutralized and has thus disappeared.

The *lime* is precipitated as calcium oxalate in 1000 cc. of the filtered water, and is weighed as calcium oxide (CaO).

The *alkalinity* (or *acidity*) of the filtered water is determined

on the usual quantity by means of N/100. hydrochloric acid (or N/100. sodium-hydrate solution) using methyl orange as indicator. The acidity is to be calculated as sulphuric acid,—the alkalinity as ammonia.

Eleven results for each monthly sample of the rain-fall will in this way be determined,—two of these representing the totals of the Insoluble and Dissolved matter, and the remaining nine being the chief constituents of the solid matter brought down with the rain.

As an example of the results obtained with this method of observation, the figures published on page 340 of our issue of

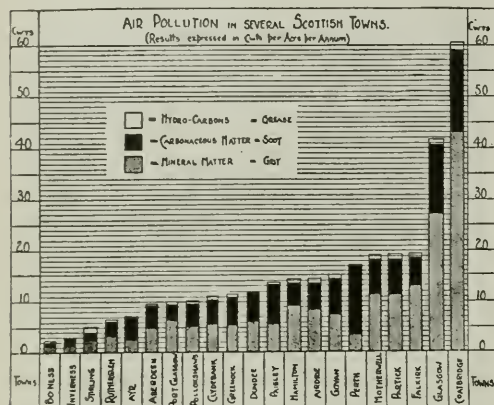


FIG. 1.—DIAGRAM SHOWING THE CONJUNCTIVE SOOT AND DUST FALL OF SCOTCH TOWNS IN TEN WINTER MONTHS OF 1910-1911

June 1913 may be referred to. These showed a total fall of soot-and-dust in the London area ranging from 195 tons per sq. mile per annum at Sutton in Surrey to 650 tons per sq. mile per annum in the center of the City of London.

As a second example, the figures obtained by Chief Inspector Fyfe in Scotland may now be quoted. These are given in Tabular form in Table I. and in diagrammatic form in Fig. 1. The method of collection used by Inspector Fyfe, namely, shallow wooden boxes—one foot square in superficial area and one foot deep, was however less exact than that used for the London observations, and it is quite possible that when the Standard apparatus and method is adopted in Scotland by other towns and cities than Glasgow, somewhat different results will be obtained.

It must be pointed out also that Inspector Fyfe's observations extended only over the two mid-winter months of the years 1910 and 1911, and that owing to the greater amount of coal burned in Winter, and to the greater carelessness of stokers and furnacemen during the long hours of darkness, both the smoke from domestic and industrial chimneys is enormously increased at this period of the year. An average for the whole year calculated on the soot and dust-fall of the two winter months, is therefore misleading and inaccurate.

The continuous records which will be obtained by the standard apparatus and method will, of course, be free from this defect, and it will be of interest to see how the soot-and-dust-fall of Glasgow compares with that of Manchester, Leeds, and other great English centers of manufacturing industry, when the first twelve months period of observations has been completed, and the returns are all in.

In concluding this account of the work that has already been achieved, some extracts may be given from the appeal for funds which has been issued recently by the Committee.

"The co-ordination of observations is the legitimate work of a Committee. The design of new apparatus and methods for carrying out any particular inquiry may properly be left to individual effort, but the selection of a method for the special work of comparison,—the specifications of common conditions,—and

"the co-ordination of the results,—are essentially matters for co-operative action and can only be carried out by a representative body.

"The Committee is now faced with the necessity for finding funds to carry out the work which will devolve upon it, when the scheme of co-operation commences. It is proposed that the several localities shall provide their own apparatus, and also make the analyses of the collected products. But when that has been done, it will still be necessary for the Central Committee to collect and arrange the records and distribute the information acquired to those who are interested in the subject, and this involves a considerable amount of secretarial work with its incidental expenses. Further, a number of questions of detail will certainly arise, with regard to the apparatus used and the methods of analysis, and these can only be answered by means of experimental inquiry.

"The Committee have no claim upon the funds of any existing Society, and that they should personally bear the costs of the inquiry, goes beyond the terms of the Resolution which ap-

TABLE I. AVERAGE AIR POLLUTION IN SEVERAL TOWNS IN SCOTLAND FOR THE WINTER MONTHS OF 1910-1911 (Fyfe)

Town	Mean population, 1901-11	Cwts. per acre per annum			Tons per sq. mile
		Mineral matter	Carbonaceous matter	Hydro-carbons, matter	
Bonness	10,086	1.35	.31	.09	2.27
Inverness	22,645	1.38	1.31	.27	2.96
Stirling	19,801	2.52	—2.50—		5.02
Rutherglen	21,502	3.30	2.84	.56	6.71
Ayr	30,841	2.68	4.35	.25	7.28
Aberdeen	158,293	4.78	4.85	.24	9.87
Port Glasgow	17,303	6.55	2.90	.58	10.03
Pollokshaws	12,037	5.52	4.01	.62	10.15
Clydebank	29,224	5.76	4.49	.97	11.22
Greenock	72,025	5.80	5.08	.82	11.70
Dundee	163,994	6.17	5.53	.20	11.90
Paisley	81,920	5.91	7.26	.62	13.79
Hamilton	35,709	9.25	4.37	.60	14.22
Airdrie	23,338	8.61	4.81	.93	14.35
Govan	85,960	7.65	6.69	.65	14.99
Perth	34,923	3.58	13.34	.39	17.31
Motherwell	35,761	11.47	6.83	.97	19.27
Partick	60,573	11.37	6.98	1.19	19.54
Falkirk	31,424	12.99	5.77	.86	19.62
Glasgow	780,028	27.03	13.54	1.04	41.61
Coatbridge	40,139	42.96	15.97	1.66	60.59

"pointed them. They therefore appeal to the public for funds "and especially for annual subscriptions in order that the work "now set on foot may be continued, and are confident that the "importance of trustworthy knowledge of the state of the atmosphere as regards impurity justifies them in making this "appeal."

The author's thanks are due to Chief Inspector Fyfe of Glasgow for permission to make use of the diagram and table giving the results of the interesting observations made in Scotland. A full account of these observations will be found in a paper contributed by Chief Inspector Fyfe to the Manchester Smoke Abatement Conference of 1911.

Liverpool, England.

The Murex magnetic concentration process has been applied successfully to both sulphide and carbonate ores. The crushed ore is mixed and agitated with an emulsion of oil and magnetic oxide of iron, after which it is passed under an electromagnet which lifts the particles which have been coated with the emulsion. The process has been used at the Cordoba copper mine in Spain, and the Whim Well copper mine in Australia. At the latter place, carbonate ore is treated, producing a concentrate of 20% to 25% copper from 6% ore, and yielding a recovery of about 80%.

The Growth of the World's Copper Mining Industry in the Years 1898-1912

By John B. C. Kershaw

There is probably no industry that can show such a remarkably rapid rate of expansion during the past quarter of a century as that of copper mining. Taking the average price of the metal produced during this period as £60 per ton, the aggregate annual value of the output of copper by all the mines of the world, has increased from £15,480,000 in 1898, to £60,024,000 in 1912, or by nearly 400 per cent.

In fact, if the aggregate value of the copper produced in 1912, be calculated on the basis of the average price obtained for bar copper in that year (namely £73 per ton) the astonishing total of £73,290,000 is obtained as the present value of the annual output of the copper mining industry. These totals prove that the industry is only second to that of gold-mining, in the value and importance of the metal produced.

The figures used in preparing the tables and diagrams which illustrate this article, are taken from the valuable monthly and yearly statistical circulars prepared by Messrs. H. R. Merton & Co., of London, to whom the writer's thanks are due, for permitting their carefully compiled returns to be used in this way.

AGGREGATE PRODUCTION OF COPPER BY ALL THE COUNTRIES OF THE WORLD IN THE PERIOD 1898-1912

Table I and Fig. 1 give the statistics showing the rapid growth of the world's copper mining industry in the fifteen years ending in 1912.

TABLE I—Total World's Output of Copper—1898-1912

Year	Output in tons	Increase or decrease
1898	429,262	29,896
1899	472,244	42,618
1900	479,514	7,270
1901	516,628	37,114
1902	541,295	24,667
1903	574,775	33,480
1904	644,000	69,225
1905	682,125	38,125
1906	714,100	31,975
1907	713,965	(—,135)
1908	754,180	40,215
1909	839,425	85,245
1910	864,275	24,850
1911	871,920	7,645
1912	1,004,485	132,565

The variations in output from year to year are seen to have been very irregular, ranging from a decrease of 135 tons in

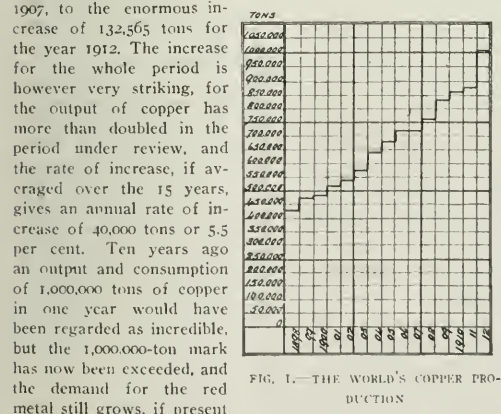


FIG. 1.—THE WORLD'S COPPER PRODUCTION

prices can be accepted as any criterion of the real demand, the relationship between the world's total output of copper and the production of the U. S. A. mines

Table II and Fig. 2 illustrate the relationship between the American copper production and that of the remaining countries of the world, and are useful as showing clearly the pre-dominating position occupied by the U. S. A. mines. The out-

put of these mines in 1912 amounted 554,835 tons, and the percentage of the total output provided from this source, worked out to 55.4 per cent. This figure agrees closely with 55.5 per cent in 1911, and proves that in spite of the large aggregate increase in production during the past year, the advantageous position occupied for so many years by the U. S. A. mines, is still maintained.

TABLE II—RELATIVE PRODUCTION OF COPPER BY THE MINES OF THE UNITED STATES AND THOSE OF OTHER COUNTRIES—1898-1912

Year	U. S. A. Mines		Other Countries	
	Tons	Percentage	Tons	Percentage
1898	234,271	55	195,355	46
1899	262,206	55	210,038	45
1900	263,502	55	216,012	45
1901	265,250	51	251,378	49
1902	292,870	54	248,425	46
1903	307,570	53	268,203	47
1904	365,050	56	278,950	44
1905	389,120	57	293,005	43
1906	409,650	57	304,450	43
1907	392,520	55	321,445	45
1908	423,300	56	330,880	44
1909	490,280	57	349,145	43
1910	484,935	56	370,750	44
1911	485,865	55	385,064	45
1912	554,835	55	449,650	45

This advantage is clearly indicated in Fig. 2, and it can be seen that the margin between the aggregate output of the U. S. A. mines and that of all other countries, is widening instead of diminishing. This paramount position of the U. S. A. mines in the copper mining industry undoubtedly enables the American financiers to exercise a control over the supplies of the metal and to regulate prices, but this is outside the scope of this article to discuss.

THE COPPER OUTPUT OF THE TWELVE MORE IMPORTANT COUNTRIES IN THE PERIOD 1898-1912

Tables III and IV and diagrams III and IV give the production figures from Messrs. H. R. Merton's returns, for the twelve more important countries (excluding U. S. A.) for the fifteen years ending 1912. Although these twelve countries have greatly increased their aggregate output of the metal during the period under review, their proportion of the total output has remained stationary, at slightly under 45 per cent.

These twelve leading countries rank in the following order of importance as copper-producing countries, when judged by their output of the metal in 1912: Mexico, Japan, Spain and

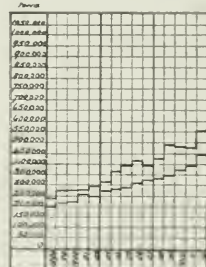


FIG. 2.—AMERICAN COPPER PRODUCTION

TABLE III—THE COPPER OUTPUT OF THE LEADING PRODUCING COUNTRIES (EXCLUDING UNITED STATES OF AMERICA) FOR THE PERIOD 1898-1912

Year	Mexico	Japan	Spain and Portugal	Australasia	Chili	Canada
1898	16,435	25,175	52,375	18,000	24,850	8,040
1899	19,335	28,310	52,168	20,750	25,000	6,730
1900	22,050	27,840	52,872	23,020	25,700	8,500
1901	30,430	27,475	55,621	30,875	30,780	15,800
1902	35,785	29,775	49,790	28,640	28,930	17,485
1903	45,315	31,360	49,790	29,000	30,390	19,320
1904	50,945	34,850	47,035	34,160	30,100	19,185
1905	64,440	35,910	44,810	33,940	29,165	20,535
1906	60,625	42,740	49,320	36,250	25,745	25,460
1907	56,565	48,935	49,675	41,250	26,685	25,615
1908	39,990	43,000	52,585	39,500	38,315	28,570
1909	56,325	47,000	52,185	34,400	35,785	24,105
1910	61,515	46,755	50,255	40,315	35,235	25,715
1911	60,905	55,000	50,930	41,840	29,595	24,930
1912	70,845	65,500	58,930	47,200	37,305	34,710

Portugal, Australasia, Chili, Canada, Russia, Peru, Germany, Africa, Norway, Servia. The figures for many of these countries exhibit the striking progress of the copper mining industry during the period under review Japan, Mexico, Russia, Australasia, Canada, and Peru showing extraordinary developments of this industry.

Mexico which in 1898 produced only 16,435 tons of the red metal, produced 70,845 tons in 1912, and has permanently displaced the Spanish Peninsula, from its place of importance as

a copper-producing country. The Mexican mines are controlled chiefly by American financial interests, and the output varies considerably according to the state of the copper market. In recent years a large number of old mines have been reopened and worked again, but these are closed down when the price of copper falls below a remunerative level.

Japan has shown steady progress as a copper-producing country during the past fifteen years, and has increased its output from 25,175 tons in 1898 to 65,500 tons in 1912, thus ranking third in the list of producing countries.

Australasia and Canada.—The copper mining and smelting industries in these two countries have also shown notable expansion in the period covered by the table—the Australasian output having increased from 18,000 tons in 1898 to 47,020 tons in 1912, and the Canadian figures showing an increase from 8104 tons to 34,710 tons.

Russia is another country in which the copper mining industry has experienced great developments in recent years, the figures in Table IV showing an advance in production from 6260 tons in 1898, to 33,010 tons in 1912.

Peru, however, is the country which can show the most remarkable expansion of the industry during the past fifteen years, the increase in this case being from 3040 tons in 1898, to 27,165 tons in 1912. The output has, however, remained stationary since 1910, and the inference is, that the copper mines of Peru have about reached the upper limit of their productive capacity. Otherwise the great rise in the price of copper that occurred during the years 1911-1912, should certainly have brought about an increased output in the latter year.

Chili, which so long ago as 1897 produced nearly 50,000 tons of copper, is once again increasing her production, and has advanced from 24,850 tons to 37,305 tons in the period covered by Table III.

Africa has advanced her output from 7110 tons in 1898, to 16,370 tons in 1912, and when the ores from the much talked-of Katanga mines, north of the Zambesi river, are being smelted on a large scale, a notable addition to the copper exports of this part of the world may be expected. At present the smelters erected in the Zambesi district are not working satisfactorily, the restricted fuel supply having created greater difficulties than was anticipated.

Serbia is a new addition to the ranks of the producing countries, and although the Serbian mines only commenced to pro-

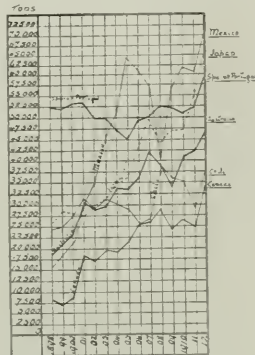


FIG. 3.—MAJOR COPPER PRODUCING COUNTRIES OTHER THAN THE U. S. A.

pansion similar to those given above. The aim of those governing the production of the copper mines situated in these countries, is well known to be that of maintaining a steady output, rather than to force production and exhaust the ore reserves, during the successive short periods of high prices.

Figs. 3 and 4 show clearly the rapid expansion of the copper mining industry in ten out of the twelve countries, during the period 1898-1912, and summarize in graphic form, the information as to progress given above.

VARIATIONS IN VALUE FOR THE PERIOD 1898 TO 1912, AND THE RELATIONSHIP BETWEEN OUTPUT AND PRICE

The figures showing the average price of standard bar copper are given in Table V., which also contains the figures for the world's total output of copper in each of the years named. Fig. 5 gives these figures in graphic form, and it will be noticed that the "curve" for the price variations, shows three periods of high

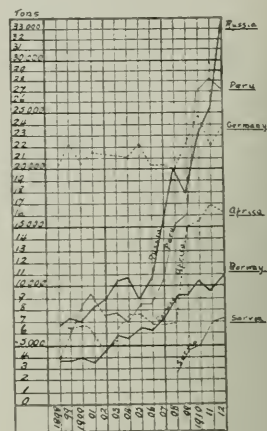


FIG. 4.—MINOR COPPER PRODUCING COUNTRIES

TABLE V.—THE WORLD'S OUTPUT OF COPPER COMPARED WITH THE AVERAGE PRICE OF "STANDARD" METAL, FOR THE PERIOD 1898-1912

Year	Total Output	Average Price, Per ton			
		£	s	d	
1898.....	429,626	51	7	10	
1899.....	472,244	72	10	6	
1900.....	479,514	73	19	7	
1901.....	516,628	67	17	3	
1902.....	541,295	52	13	5	
1903.....	574,775	57	18	8	
1904.....	644,000	58	14	8	
1905.....	682,125	69	2	6	
1906.....	714,100	86	5	2	
1907.....	713,965	87	1	8	
1908.....	754,180	60	0	6	
1909.....	839,428	58	17	3	
1910.....	869,275	57	3	2	
1911.....	871,920	55	16	2	
1912.....	1,004,485	73	1	3	

prices with three depressions. The price boom of 1898 to 1899 was due to the formation of the Amalgamated Copper Com-

pany in the States, the second and third booms have been due to more natural causes, namely the extremely active state of trade in the shipbuilding and electrical engineering and allied industries, at a period when there was some slackening off in the rate of production in the copper mining industry, and in consequence some depletion of stocks.

As pointed out by a writer in the London Times recently, activity in the shipbuilding industry creates a great demand for copper, and "if the man-in-the 'street or the speculator in 'copper-warrants were to 'make an exhaustive examination of a first-class high-speed ocean liner, he would be 'amazed at the quantity of copper, plain or as brass required 'about her engines, her auxiliary plant, her sanitary arrange-

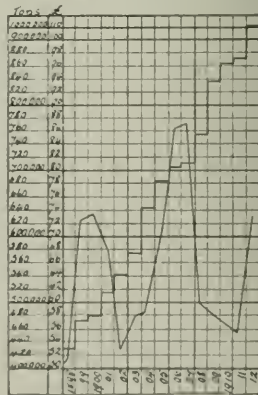


FIG. 5.—COPPER PRODUCTION AND PRICES

TABLE IV.—THE COPPER OUTPUT OF THE MINOR PRODUCING COUNTRIES EXCLUDING UNITED STATES OF AMERICA FOR THE PERIOD 1898-1912

Year	Russia	Peru	Germany	Africa	Norway	Serbia
1898	6,260	3,040	20,085	7,110	3,616	...
1899	7,210	5,165	23,460	6,490	3,610	...
1900	6,740	8,220	20,410	6,720	3,935	...
1901	8,000	9,520	21,720	6,900	3,373	...
1902	8,675	7,580	21,605	4,450	4,565	...
1903	10,320	7,800	21,205	5,230	5,915	...
1904	10,700	8,755	21,045	5,275	5,415	...
1905	8,700	8,625	22,160	7,740	6,305	...
1906	10,400	8,705	20,340	6,980	6,120	...
1907	5,000	10,575	20,490	6,800	7,010	...
1908	30,085	15,000	20,200	6,880	9,150	2,140
1909	17,750	16,000	22,455	14,945	9,080	4,480
1910	22,310	26,945	24,710	15,205	10,425	4,845
1911	25,310	28,050	22,010	16,980	9,420	6,885
1912	10,010	17,165	23,920	16,370	10,980	7,240

duce copper in 1908, their output in 1912 amounted to 7,240 tons. The Balkan war will, however have stopped all development here for the present year.

The old established copper mining industries in Spain and Portugal and in Germany do not show any figures for its ex-

"ments, her kitchens, her saloons, staterooms, and staircases, "even her portholes, rails and scuppers."

The estimated consumption of copper in 1912 was 1,024,000 tons. The stocks were reduced by 20,000 tons between January and December 31, in that year, and the output amounted, as already stated, to the huge total of 1,004,000 tons. How long the mines of the world can continue to supply copper at this rate is a question, the answer to which only the future can disclose. There will undoubtedly come a time, however, when the supplies of ore of the red metal will become exhausted, and when copper will appreciate in value owing to this cause. It will then be the duty of the chemist and metallurgist to provide a satisfactory substitute for the metal which has so long occupied a foremost place in the metal industry of the world, and it will no doubt be found that *aluminium* or some of the *aluminium alloys*, can be used for many of the hundred and one purposes, for which *copper* is now universally employed.

Liverpool, England.

Applications of the Microscope to Industrial Work

The season's first meeting of the New York section of the American Electrochemical Society was held on October 17 at the Chemists' Building and was very enjoyable and successful. The attendance was large, the big lecture hall being almost completely filled. Although the meeting lasted well beyond 11 o'clock, the attention of the audience never lacked.

Great credit is due to the chairman of the section, Mr. **Lawrence Addicks**, superintendent of the U. S. Metal Refining Company, of Chrome, New Jersey, for having arranged an exceedingly interesting program. The subject of the evening was the application of the microscope to industrial work. All the papers were profusely illustrated with most instructive and in part brilliant lantern slides. As it is a physical impossibility to reproduce here the many microphotographs shown, it is rather difficult to do justice to the meeting in this report.

The Metallography of Carbon

The first paper was presented by Professor **G. A. Roush**, of Lehigh University. In an investigation of the internal construction of carbon products and of the kinds of raw materials suitable in their manufacture, success was obtained by the application of the ordinary principles of preparing samples of metals for metallographic examination. The ordinary metallographic methods proved satisfactory also for carbon products, with the exception that some special means of etching were required to bring out the structure of the sample, since the acids used for this purpose with metal samples do not attack carbon. Relief polishing, used as with metals, proves satisfactory in some cases; this is secured by polishing the sample on a soft, yielding surface, such as broadcloth, so that the soft parts of the sample are worn away, leaving the harder parts in relief.

For ordinary work, etching with heat gave good results. A sample, after being polished, was heated to a low red heat until the oxygen of the air had attacked the polished surface just enough to dull it a little, and then the sample was cooled in a blast of air. This oxidized the softer parts of the sample faster than it did the harder parts, and thus developed the structure. Samples of carbon treated, in this way showed the individual grains of which the body of the material was composed, the thin films of binder holding these grains together,

and the pores between the grains. A sample can then be judged in regard to its porosity, and in regard to the size and shape of the grains from which it is made up. The size and shape of the grains will be found to vary widely with the method of crushing, and the porosity varies with the grain size and the pressure under which the material has been formed into shape.

By carrying matters a step further, and examining samples of the various raw materials used it was found that the more important of these showed a characteristic structure, so that the individual grains of sample from an unknown source could be identified. When this was found to be possible, the methods of measurement of microscopic particles used in petrography were made use of, and in this way, approximate quantitative analyses of materials could be made. A known sample, containing 33 per cent of petroleum coke by volume, showed by analysis 32 per cent.

Carbon electrodes which had been converted into Acheson graphite showed the outlines of the individual grains of the raw material, and frequently showed the characteristic structure of the raw material. Whether this retention of the original structure was due to incomplete conversion into graphite, or whether it was analogous to a pseudomorphous crystal, was not determined.

The examination of fine powders under the microscope sometimes can be greatly facilitated by moistening with a very dilute solution of gum arabic, pressing together firmly, smoothing off a surface with a spatula, drying, and then examining the grains making up this smooth surface.

Professor Roush's paper was illustrated by numerous lantern slides showing photomicrographs of characteristic structures.

Micrometry as Applied to Alloys

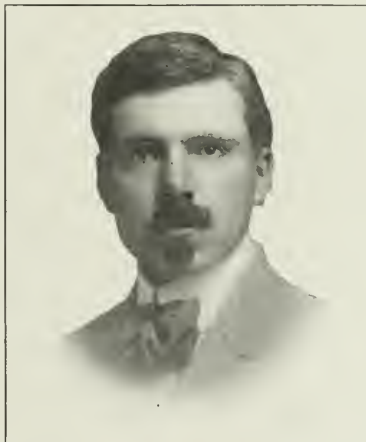
A paper on this subject was then presented by Dr. **C. H. Mathewson**, assistant professor of metallurgy at Sheffield Scientific School, Yale University.

In a general sense, methods in which the microscope is used are termed micrographic methods; when applied in a rigidly quantitative sense they are termed micrometric methods. The discussion in the first part of the paper is based on the Al-Zn system, as recently revised by Rosenhain and Archbutt.

The conditions represented in an earlier diagram of Shepherd were very simple. According to it no intermediate phases are formed, but both aluminium and zinc crystallize as solid solutions, aluminium dissolving some 45 per cent of zinc and zinc dissolving several per cent of aluminium at the eutectic temperature. The eutectic mixture contains 5 per cent of aluminium.

Rosenhain and Archbutt's later diagram, based on very accurate and delicate measurements, is of considerable complexity. This is mainly associated with the presence of an inter-metallic compound Al_2Zn_3 which escaped the attention of earlier observers as it is formed in most of the alloys by reaction between a primary crystalline phase, rich in aluminium and residual solution containing about 85 per cent Zn, with little evolution of heat.

In this investigation the errors due to incomplete reactions became evident. Thus the primary crystals may be enveloped by a layer of compound as it first forms, and the cores of the crystals are thus protected from further change. Rosenhain and Archbutt emphasize the caution which must be used



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in applying the method of thermal analysis, owing to the possibility of incomplete adjustment of equilibrium during the brief period of time in which all observations are made. This possibility must be considered in all cases where it is desired to interpret micrographic evidence quantitatively, since any change in solubility, any chemical reaction or other variety of transformation involving crystalline phases, is likely to proceed incompletely, particularly if it occurs at a fairly low temperature.

By annealing their alloys at a temperature slightly below the peritectic horizontal, Rothenhain and Archbutt were able to prevent envelopment and conduct the reaction quantitatively.

In addition to the qualitative interpretation of photomicrographs, as authors commonly apply theme in a semi-quantitative sense. In the present case, having established the fact that the reaction may be rendered complete by annealing at a suitable temperature, it is obviously in order to seek the concentration at which both reacting phases are completely exhausted in forming the compound by examining a series of micrographs. When this result is attained there will be no residual eutectic to be seen in the alloy. Rosenhain and Archbutt show a photomicrograph in which a much smaller quantity of the white constituent is to be seen. This alloy contains 80 per cent zinc and is evidently very close to the composition of the pure compound. In a third photomicrograph, corresponding to 77 per cent zinc, no eutectic is to be seen. We cannot conclude that this is the exact composition of the compound, since it forms solid solutions with an excess of Al, but, it is clear that the pure compound lies between 77 and 80 per cent zinc, and, according to the thermo-analytical results, the eutectic disappears at 78.8 per cent. The formula Al_2Zn_3 requires 78.35 per cent. The different results are quite sufficiently in accord. There appears to be no reason why we may not make use of these photo-micrographs in a rigidly quantitative sense. The structure-elements appearing in them are zinc, containing about 1 per cent Al, according to the authors, and Al_2Zn_3 , both, probably of invariable composition under the present thermal treatment. It is true however that the composition of the zinc-rich phase should be more thoroughly investigated for these purposes. When this point is settled and when we have ascertained the relative densities of the two constituents, we are in a position to calculate the actual percentages of each constituent by weight from area measurements on the photo-micrograph.

Assuming that the equilibrium diagram is available in correct form, we are thus able to calculate the percentage of Al or Zn present in any alloy within this part of the diagram by properly annealing the alloy, preparing a representative section, examining it under the microscope, and photographing a typical portion of the surface, or tracing it directly from the screen to save time. It is, of course, highly important to sample the metal carefully in the first place, as in ordinary analysis. This may be done in the usual way, by taking drillings or chips from different parts of the specimen if the may be melted down easily without change in composition. It is better to take a few coarse pieces from different parts of the specimen and melt them down under a flux. With low melting metals, this may be done as rapidly as the ordinary process of sampling and solution. In this method of analysis, a second important process of sampling or selection occurs under the microscope. At this point, the whole surface should be examined carefully and a thoroughly representative spot chosen for the graphical treatment.

While, in most investigations, the formulas of intermetallic compounds and other fundamental concentration values have been located by thermal analysis, it is quite as feasible to use the micrometric results in a similar manner, i. e., instead of represented eutectic periods of constant temperature as ordinates at the various concentration points and extrapolating concentrations at which a zero or maximum period occurs, we may represent the percentage area of eutectic in the different concentrations as ordinates and extrapolate in the same manner. This possesses an advantage over the thermo-analytical

method in that it is not necessary to use the same volume or weight of metal in the different determinations (cf. Tamman, *Z. anorg. Chem.*, 45, 292, 1905).

Dr. Mathewson has used Rosenhain and Archbutt's photomicrographs in this way with the following results: The area of the white constituent in the 80 per cent Zn alloy, determined with the polar planimeter, is 25.6 per cent of the whole; in the 77 per cent Zn alloy, it is 8.3 per cent. Assuming a linear decrease in the volume percentage of the white constituent, with the concentration (expressed in terms of Zn and Al_2Zn_3), the zero value obtains at 78.1 per cent Zn. That this result agrees almost exactly with the true composition of the compound Al_2Zn_3 , 78.35 per cent zinc, must be regarded as fortuitous, since the photo-micrographs were not selected by the authors with a view to micrographic work and, the planimetric measurements were not made on specially large scale drawings.

In using the planimeter on the 77 per cent Zn alloy the isolated white areas were joined to one another in the manner suggested by Sauveur (¹). The tracing pin was then moved over portions of the area boundaries in one direction and returned over the remaining boundaries following the connecting lines in the reverse direction, so as to nullify their original effect on the reading.

In practically all metallographic papers, we find semi-quantitative deductions based upon some of the photomicrographs. But Dr. Mathewson points out that it is dangerous to rely upon general appearances instead of actual measurements of area, even though approximate. In this case, a calculation based upon the equilibrium diagram shows that the eutectic should contain 19.4 per cent of the compound by weight. When treated so as to reach equilibrium below 236 deg., there is a possibility that the areas corresponding originally to compound will be still less, owing to decomposition of the compound and its replacement by the Zn-rich phase and another Al-rich phase, some of the former probably coalescing with the original light constituent. We do not know the densities of the two constituents, and cannot calculate the area which should correspond to the above percentage by weight, but it is certainly true that not more than one-third of the whole area is occupied by the dark constituent in photomicrograph. Rosenhain and Archbutt's photomicrographs of the slowly cooled eutectic. This conclusion was reached by tracing the outlines of the dark constituent upon finely ruled co-ordinate paper, estimating the number of squares included within these outlines, and calculating the total number as a percentage of the whole. As a result of two separate measurements, 28.6 and 29.2 per cent of the area is found to represent the compound.

It is clear that the relation between the squares and the individual areas to be estimated may be chosen so that a relatively large number of squares are entirely included within an individual area, leaving a comparatively small portion of the area open to errors of measurement. In many cases, this method of measurement will be found more convenient and probably more accurate than the planimetric method. By comparison, such methods are cutting out the separate images and weighing the paper, or measuring the portions of a straight line intercepted by the different constituents, are laborious or inferior in point of accuracy. This method may be applied very rapidly by tracing the outlines of the constituent in question upon a thin sheet of co-ordinate paper upon which the image is focused through a clear glass plate at the back of the camera. The author has made some determinations of iron in spelter in this manner at the Hammond Laboratory of the Sheffield Scientific School.

Previous investigation has shown that iron crystallizes from zinc in the form of beautifully developed individuals containing from 7 to 11 atoms of zinc per atom of iron. This represents a case very favorable to the development of a micrometric method of analysis, since some 13 times the weight of

¹Trans. Am. Inst. Min. Eng., 26, 880 (1896).

the added element, iron, is clearly visible as a distinct structure element. This case was discussed in some detail by Dr. Mathewson.

Probably more attention has been devoted to the determination of various elements entering into eutectic relations with copper than to any other micrometric problem. Heyn (*Zeit. L. anorg. Chem.*, vol. 39, p. 1, 1904) first demonstrated the structural relations brought about by the solidification of copper containing oxide and introduced a method of determining the oxygen content by area measurements. It is now recognized that this method is sufficiently reliable to be of practical value and far more convenient than any chemical method, although area measurements of eutectic structures in the presence of a primary constituent are likely to give variable results, and the exact quantitative effect of impurities, such as As, Sb, etc., has not been determined. According to this method, the relative areas of eutectic and primary copper crystals are determined by measurement, planimetric or otherwise, on the micrograph. The eutectic is composed of 3.45 per cent cuprous oxide and 96.55 per cent copper, according to Heyn. Since one unit of oxygen by weight corresponds to 8.95 units of cuprous oxide, which in turn correspond to 28.99 units of eutectic, the oxygen is represented in the micrograph by a distinct structure element 261.46 times its own weight. We may make errors of several per cent in estimating the weight of the eutectic by area measurements without seriously affecting the results as expressed in oxygen.

Hofman, Green, and Yerxa have determined the oxide in samples of refined copper by using the planimeter on enlarged micrographs (magnified 1150 diameters) and by analysis, according to Hampe's method. Bardwell has described the use of a similar method at the Great Falls smelter and gives comparisons with ordinary analytical data which clearly indicate the value of the method in technical practice. Dr. Mathewson has used the micrometric method of determining oxide in cast copper with students in the Department of Mining and Metallurgy at the Sheffield Scientific School for several years.

Huntington and Desch have tested the micrometric method of analysis very fully in the case of copper-phosphorus alloys which also contain the non-metallic element in the eutectic condition. In this case, the eutectic is composed of copper crystals containing 0.175 per cent phosphorus in solid solution and copper phosphide, Cu_3P , the total composition of the mixture being 8.27 P + 91.73 per cent Cu. The amount of phosphorus held in solid solution by the copper appears to be constant in the cast and annealed alloys. A very different factor, however, renders the ordinary planimetric determination of the phosphorus content by measuring the eutectic areas very inaccurate. This lies in the variable composition of the metal surrounding the primary copper crystals. Normally this should be of definite (eutectic) composition, but during the cooling of the alloy subsequent to eutectic crystallization, or during the annealing, the small copper crystals of the eutectic have coalesced with the large primary crystals in the immediate vicinity and the phosphide has also coalesced around the enlarged copper crystals. It is evident that the eutectic no longer contains 91.73 per cent Cu if we measure it from the boundaries of the black crystals, but somewhat less than this. Huntington and Desch were able to correct the measured areas of the black crystals with surprising accuracy by measuring the area of the enveloping belt of phosphide and deducting the amount of eutectic copper normally corresponding to this phosphide from the black crystals. When the phosphorus content is low the net work of eutectic is originally fine and, as a result, all of the copper coalesces with the large crystals leaving a pure phosphide network.

In the micrometric treatment of eutectic structures great care must be taken to recognize and allow for coalescence. In some cases, it is practically absent as in copper-phosphorus alloys containing copper phosphide as a primary constituent, i. e., those high in phosphorus. There also appears to be relatively little interference of this sort in the copper-copper oxide alloys previously discussed.

In forecasting the future possibilities of micrometric methods, Dr. Mathewson pointed out that we must be mindful of the fact that the microscope offers primarily a means of studying the proximate composition of metallic material and that the direct results obtained in this manner can be translated into ordinary analytical results only in favorable cases and under closely regulated conditions. It is, therefore, more logical to expect future development to lie principally in the direction of interpreting and applying the proximate results themselves in the estimation and specifications of quality, without regard to chemical compositions, except within prescribed limits. It is conceivable that, in many cases, it may be far more important for the manufacturer to know that his product conforms to a certain standard in respect to size of crystals, percentage of area occupied by inherent crystalline phases, or micrographic character in general, than to possess elaborate analytical detail concerning the ultimate composition.

It appears more desirable to record the micrometric results directly in terms of the structure elements measured, and seek to make full application of them in this form rather than to attempt the calculation of ultimate composition from them. For example, the character of the copper-cuprous oxide eutectic is influenced by the presence of arsenic. In the absence of precise information on this point, it does not appear desirable to calculate the oxygen in an arsenical copper under the assumption that the eutectic contains 3.45 per cent Cu_2O . It seems better to record the percentage area of primary copper crystals along with qualitative data regarding the appearance and distribution of the eutectic.

Obviously, it is impossible to draw any deduction concerning the ultimate composition of an alloy composed of a homogeneous solid solution from micrometric results. It is a matter of common practice, however, to control the heat treatment of such alloys by measuring the size of the crystals or counting the number present in a given area under given magnification.

Dr. Mathewson concluded that we have in micrometric analysis a valuable aid in the scientific or technical examination of alloys, which is often superior to ordinary analysis as regards the character and extent of the information returned, but which ordinarily constitutes a source of distinctive results rather than a substitute for ordinary analysis. A satisfying total of analytical information is obtained by judiciously combining the two methods.

Microstructure of Raw and Manufactured Copper

A really remarkable series of microphotographs was shown and discussed in a paper of Mr. W. H. Bassett, Technical Superintendent and Metallurgist of the American Brass Company of Waterbury, Conn.

The microphotographs shown indicated the changes in structure due to working copper, that is, they showed the differences between copper in the cast and wrought condition.

The pictures also showed clearly the influence of the oxygen which is always present in copper, upon its structure, both in the cast and wrought condition. The presence of the copper-copper suboxide eutectic is readily distinguished microscopically, and several quantitative methods have been devised for determining the amount of oxygen in the copper by the aid of the microscope.

Certain impurities in the copper, such as antimony, modify the character of the eutectic, and their presence can be readily distinguished through a microscopic examination.

A photograph was shown indicating the presence of a considerable quantity of the copper-copper suboxide eutectic in over-poled copper where such copper contained a comparatively large percentage of sulphur. The picture exhibited was that of an over-poled specimen carrying 0.012 per cent sulphur.

A number of pictures were exhibited showing the relation between the set and microstructure of wire bar copper. Also, the influence of the amount of oxygen present upon the soundness of the bars.

Photomicrographs of native copper samples from Lake Superior were shown. These have the same type of structure as wrought copper. The crystals, however, are very much enlarged probably due to the very slow formation. The pressure exerted upon this copper in the rock produces the same form of slip bands as those shown in the crystals of cold rolled copper. A number of photomicrographs were shown indicating the various types of structures of hot rolled, cold rolled, drawn, and annealed specimens. Attention was called to the oxygen spots in this material and their absence in wrought specimens of copper which had been deoxidized at the time of casting. Several diagrams indicating the influence of cold working upon the physical and electrical properties of copper were discussed, and the influence of annealing at various temperatures on these properties, and, also, upon the microstructure noted.

The hope may be expressed here that Mr. Basset's able paper will be published in full with the full series of microphotographs.

The Microscope in Mineralogical Analysis

A paper on this subject was presented by Mr. Gilbert Rigg, research engineer of the New Jersey Zinc Company.

Mr. Rigg has probably been the first to apply color photography to metallography. He exhibited a striking series of Lumiere plates showing photomicrographs of zinc ores, with resolution of the minerals present. This beautiful set of colored photomicrographs fittingly closed the official program.

In view of the late hour, no time was left for extended discussion. But Professor William Campbell also showed some interesting slides of photomicrographs.

Microscope Exhibit.

A very interesting exhibit of microscopes of the firm of Ernst Leitz, New York, was inspected by many of those present and explained by Mr. A. Traeger after the close of the meeting. As this exhibit showed the remarkable advances made in recent years in the design of microscopes especially for metallographic work, some brief notes on the principal types exhibited should be interesting.

The Leitz micro-metallograph is constructed after the le Chatelier principle and embodies some suggestions of Prof. Guertler, of the Technische Hochschule in Charlottenburg, and Dr. Wm. Campbell, of Columbia University. It is being used by the leading metal industries in their laboratories distributed over various departments. The special advantage of the Leitz micrometallograph is its extremely easy manipulation, the image, once observed by the eye, can be immediately transferred to the ground glass or plate respectively, after the ocular tube has been removed out of the optical axis. The instrument itself is mounted on a heavy triangular optical bench, thus eliminating any vibration which is most essential, considering the fact that practically in every industrial laboratory, vibration to a certain extent is experienced, this due to the factory being at a close range of same.

This instrument was shown with three different optical outfits. First, the achromatic, or ordinary microscope objectives, are used mostly for routine investigations and especially in cases where the instrument is not desired for the photo-micrographic study of metals, opaque ores or alloys.

Second, the semi-apochromatic objective series is advanced in its constructions over the achromatic objectives, this due to the increase of flatness of field, color correction and resolving power. This series of lenses is recommended, in preference to the achromats, for the visual observation and photo-micrographic study.

Third, the apochromatic objectives, however, represent a series of lenses of highest efficiency. The color correction, resolving power and flatness of field, have been increased to the highest state that optical science provides for at present. The apochromats come especially recommended for the photo-micrography of metals, alloys and opaque ores, and the results with these lenses are superior to those which may be obtained with either one of the optical equipments mentioned above.

Another interesting instrument exhibited was a small metallurgical microscope (Leitz Model B), constructed after the suggestions of Prof. Wm. Campbell. This metallurgical microscope is of a smaller type and recommended for use in those laboratories where the metallographic study is only attended to when occasion arises. This instrument, combined with a small upright photo-micrographic camera, serves advantageously for the photo-micrographic study of metals, alloys and opaque ores. A decided advantage of this apparatus is its detachable model. While small metal pieces are placed on the stage of the microscope (the latter is supplied with rack and pinion movement for cross focus) this method of investigation would not be possible for large metal pieces. In this instance, the upper part with microscope tube is removed, the microscope table, mounted on column, eliminated and the upper part attached direct to the horseshoe base. The microscope stand, if assembled in this way, can be placed direct on large metal surfaces.

The Leitz-Greenough binocular microscope, which was also shown, is supplied with a prism arrangement, allowing the use of both eyes for the stereoscopic vision of cross minerals, metal pieces, etc. The stereoscopic image is erected, this contrary to the results obtained with the regular compound microscope.

Another interesting exhibit was the Leitz petrographic microscope, constructed after the suggestions of Prof. Chas. Berkey, of Columbia University, especially for work in laboratories of industrial concerns and educational institutions. The optical equipment which is used in connection with this instrument covers an essential outfit needed for the routine observation of minerals, etc.

The exhibit also included a special microscope stand for the advanced study of mineralogy, and another microscope stand specially designed for chemical laboratories where frequently the observation of specimens under polarized light and then again the regular biological observation is necessary. The polarizing adjuncts, therefore, are mounted in such a way that they are easily removed in order to render the microscope available for biological observation.

The next meeting of the New York section of the American Electrochemical Society will be held jointly with the New York section of the American Institute of Mining Engineers on Thursday, November 20, at the Engineering Societies Building, 29 West Thirty-ninth Street, New York City. The subject will be the Metallurgy of Zinc, and figures will be presented by Messrs. G. C. Stone, W. R. Ingalls, and Victor Engelhardt.

Calculating Sulphuric Acid in Vats.—The following tables, given by W. Shellshear in the *Mining and Engineering Review*, Australia, facilitate the calculation of the quantity of acid in a vat. The vat is first measured and its water content obtained and tabulated per inch of depth. From this the amount of acid in the vat is read as tons of water. This reading can be converted into tons of mono-acid by multiplying by the sp. gr. of the acid, and by the percentage strength of acid corresponding to this sp. gr. That is: Tons of mono-acid = tons of water X (sp. gr. of acid $\times$ per cent strength of acid), or tons of mono-acid = tons of water X. The value of X can be calculated for different specific gravities.

Thus, if the sp. gr. of acid is taken by the t waddell hydrometer at 15° C., the table would be:

Tw.	X	Tw.	X	Tw.	X	Tw.	X	Tw.	X
80	763	92	819	104	876	116	1,054	128	1,181
81	712	93	820	105	946	117	1,064	129	1,191
82	722	94	839	106	956	118	1,075	130	1,201
83	731	95	848	107	967	119	1,085	131	1,212
84	741	96	858	108	977	120	1,096	132	1,224
85	751	97	869	109	987	121	1,107	133	1,233
86	761	98	879	110	996	122	1,118	134	1,244
87	770	99	889	111	1,006	123	1,125	135	1,256
88	780	100	895	112	1,015	124	1,139	136	1,267
89	791	101	905	113	1,025	125	1,150	137	1,278
90	800	102	916	114	1,035	126	1,160	138	1,289
91	810	103	926	115	1,044	127	1,170	139	1,301

Example: A vat holds 18 in. sulphuric acid of 105° Tw. The vat when full to the same depth of water holds 5 tons. Then the mono-acid in the vat = 5 X, or 5 $\times$.946 = 4.73 tons.

The Brush Discharge

By A. Vosmaer, Ph.D.

The literature on the brush discharge is very scarce. Most scientists work on too small a scale, while practical men have neither the time nor the opportunity to study the brush discharge as such.

For this reason I hope that the following account of my personal experience of fifteen years with brush discharge phenomena will be found useful.

Part I.—The Brush Discharge through Air without the Use of Solid Dielectrics

There is perhaps no essential difference between the passage of electricity in a solid, a liquid or a gas, although it is customary to call the first conduction, the second electrolytic (ionic) conduction and the third convection. It would perhaps be better to consider all of them to be convective in character.

In any case a potential difference between the two points considered is essential for a current; if there is no P. D. there will be no current. It is immaterial whether we think of P. D. as the first cause or whether we look upon resistance as the essential for a P. D. again.

In the whole of physics the rule holds that there can be no force exerted unless there is a resistance against which said force can be applied. Sometimes action creates its own reaction (static back pressure). There is a general tendency in nature to reduce all "free energy" to a minimum, thereby increasing the entropy to a maximum; all water at a high level will run downward; all differences in temperature will be equalized at the cost of the higher temperature; never will any amount of heat as measured by heat units be capable of raising the temperature of something that has a higher temperature. In one word, in nature's free state all energy change is in the direction toward decrease of intensity and increase of capacity.

Now if said change is not checked in some way or another it will exhibit itself; the usual way of checking is by means of the unknown entity called resistance.

Ohm's famous law is usually expressed by saying that the current is the quotient of tension and resistance, but it would be more logical to say that tension is the product of current into resistance. The first way, however, is much more convenient when reasoning in a popular way upon some common thing.

A potential difference, however small, even when infinitesimal, will always cause a current of electricity to flow through a solid conductor (if we may use the term of flow). But this is not true for an electrolytic conductor, in which case the impressed e.m.f. has to be greater than the counter e.m.f. of polarization in order to produce a current.

In gases the problem is still much more complicated as we shall learn.

We may neglect the counter e.m.f., but there is a lower limit of e.m.f. below which no current will pass through a gas. The value of this lower limit of e.m.f. varies with such factors as the temperature, the degree of ionization, the shape and state of electrodes, etc. Therefore, the value of this lower limit is not a constant at all and cannot be given even approximately. It may be in the neighborhood of a hundred volts, or much higher.

The discharge of electricity in a gaseous medium depends on quite a number of factors, such as the pressure of the gas, its temperature, its nature. Further, it depends on the shape, on the size and spacing of the electrodes, and on the finish of each of them. Moreover, there are some secondary influences such as radiation (ultraviolet light) and the effects of a magnetic field or an electric field. It also depends on the kind of current supply, its tension, frequency, wave form and the circuit as a whole.

We shall discuss these influences in succession. Evidently the problem is rather a complicated one so that we shall leave out of consideration many important details.

PRESSURE

In the first place we shall have to confine ourselves to the study of the discharge in a space filled with gas and will not consider the phenomena in vacua, as these would form a treatise many times the length of this one.

It may be mentioned that a pressure lower than the atmospheric, greatly facilitates the passage of a current down to a certain point where there is so little gas left that there are not enough ions to convey the electricity.

The brush discharge is affected in such a way that even a small diminution of pressure changes its character altogether. In view of this fact, one may feel inclined to try higher pressures. But the same holds true there, though in a less degree. We have tried up to three atmospheres, but every increase in pressure seemed to make it more difficult for the brush to keep up its being.

Watson, when constructing his high-tension voltmeter, has taken advantage of the fact that at high pressures gases become insulators. They are very poor conductors when liquefied; the conductivity of liquefied gases is of the same order as that of pure water.

TEMPERATURE

I have had no occasion to experiment either with very high or with very low temperatures, but it is probable that the first will be detrimental to a true brush discharge and the second will not have much effect until extreme temperatures are reached.

MOISTURE

It is several years ago that Steinmetz, experimenting on high-tension sparks, published the result that moisture has little effect on the dielectric strength of air, and what influence it has points to increase rather than decrease as one would suspect at first thought. The brush discharge does not seem to be sensitive to moisture; although there is some evidence pointing in the direction of decrease of P. D. for the same brush, I would not dare to claim this as a fact.

KIND OF GAS

Since the object of my researches has been the manufacture of ozone, I have always been experimenting with air or oxygen and have not tried other gases except hydrogen and carbonic acid. No marked change in the character of the brush could be detected when going but superficially into this matter. It would have been very interesting to know just what happens when other gases are subjected to the very strong influence of the brush discharge, but the scope of my work was already too big so that I could not go too far beyond it.

IONIZATION

It is a very strange fact that though the brush discharge is powerful it does not seem to have much force of ionization. When a discharge of as much as two watts was confined in a small tube of glass and the air blown to a charged electroscope, after having been filtered through cottonwool, it did not change the position of the gold leaves, no matter whether they were negatively or positively charged.

This result surprised and annoyed me because I had expected to be able to prove that the brush discharge is essentially a positive one. I have every evidence that it is a discharge from the positive wave and not from the negative, but there are many investigators who hold the reverse to be true.

KINDS OF DISCHARGES

Before proceeding I wish to call particular attention to the fact that for the present moment we are considering discharges in free air, no solid dielectric being between the electrodes; the latter case will be considered afterward.

In free air, then, at atmospheric pressure and normal temperature, there are four distinctly different kinds of discharges, viz., the glow, the brush, the spark and the arc.

The glow is the first stage. It is merely a bright spot on the electrode and causes no appreciable loss of energy unless very prominent, in which case it is very liable to become a brush

¹The corona loss, which is caused by the glow discharge, is but small compared to what it would be were it a brush discharge.

The interest in the glow discharge has greatly increased since in high-tension power transmissions the voltage has been varied so much as to bring about the corona effect.

There is much information now to be had on this subject of the corona effect which a few years back had no interest for the practical men. The corona loss starts at the point



FIG. 1.—BRUSH DISCHARGE

when the glow is about changing to the brush. There is no sharp line of demarkation between the glow and the brush discharges, though in their most characteristic forms they show no resemblance to each other.

The true *brush* (a photograph of which is reproduced in Fig. 1) has very distinct features. Its color is a dark bluish



FIG. 2.—BRUSH DISCHARGE

violet, its hissing sound and "wind" are well known,² its shape is a straight stem detached from the electrode from which it starts or rather separated from it by a dark space, and at its end there is a widely spread out cone. This cone or conical



FIG. 3.—SPARK

end may be flattened into a fan-shaped end, but not in all directions.

It is very important to remember this special form of the brush, as it gives a clue to many phenomena which are apt to appear when one is building up an ozone apparatus.

The ignorance of this particular behavior of the brush ac-

²The names, Silent Discharge and Dark Discharge should be dropped.

counts for many a failure in actual practice of ozonators, as we shall presently see.

The brush is a three-dimensional discharge; at the best it may be reduced apparently to two dimensions, that is, it may be flattened in one direction so that it looks as though it had no thickness. Still one should never lose sight of the fact that it has to have some extension in space in all directions.

For practical purposes this means that one cannot possibly have a continuous line of discharge. There must be free

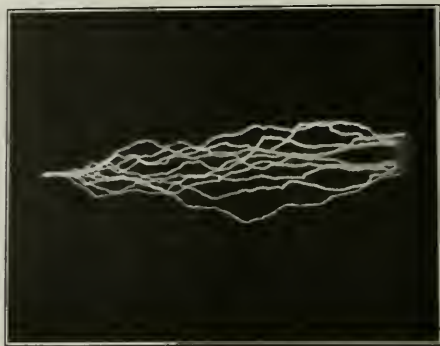


FIG. 3A.—SERIES OF TEN CONSECUTIVE SPARKS

space left between the different discharges, or in other words, it is necessary not to use a perfect smooth line, like a razor blade, but a dented one, for the discharger.

Fig. 2 explains this sufficiently. The latter has been exaggerated purposely. In actual praxis, using 1.3 millions of points for a 20-kilowatt apparatus, there were 36 points used per inch. Sharper and longer points allow to use 50 points per inch.

All this refers only to spacing in one direction. We shall see later on what the spacing has to be in the other direction, that one being more evident on account of the visible spread out at the base.

The brush discharge passes over again into the *spark* without a sharp boundary between them, although here again the brush and the spark, in their most characteristic forms, do not have any resemblance.

The spark is characterized by its white yellow color, its peculiar zig-zag path, which under certain circumstances can

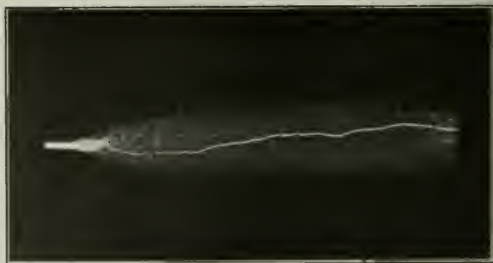


FIG. 4.—SPARK MIXED WITH BRUSH DISCHARGE

be made straight, its loud report and its oscillatory motion, in contrast to the brush which is unidirectional and not intermittent.

Fig. 3 shows a photograph of a spark of 10 in. in length given off by a powerful 12-plate Whinshurst machine. Fig. 4 shows the mixing of a spark and some faint brush discharge from the same machine, run without its condensers. Fig. 3a represents a series of ten consecutive sparks.

If circumstances are such as to allow of it, a spark will

change to an arc. This is most dangerous of all forms of discharges on account of the tremendous amount of energy transformed into heat and concentrated in a small space; it generally means ruin of the apparatus unless checked in due time or taken care of and controlled by some method such as those now used in the systems of making nitrous compounds out of the air.

We shall not discuss in detail all these discharges, but had to mention them because we shall have to refer to them.



FIG. 5.—ARC

The arc (Fig. 5) is very hard to photograph because it is so restlessly moving about. Its characteristic features are its faint blue color, its intense heat and its continuity.

POLARITY

When we speak of the brush we always mean the positive brush because there is no real negative brush, although there is a negative discharge under the same circumstances.

In Fig. 5a we show side to side the positive and the negative discharge. The difference in appearance is striking, but could have been shown even much more strikingly had I taken other prints.

If matters are arranged in such a way that the pointed discharger is made the negative pole, all we obtain instead of the beautiful brush is a tiny little violet spot about one-tenth the size of the positive brush or sometimes even less than that. The demonstration of this very marked difference is quite easily made. It used to strike most scientists who watched it. A powerful Rhumkorff induction coil is the best source of supply for this experiment, as it gives practically unidirectional high-tension current, the intermissions are too short to be noticed, and if the output is taken up by a good number of points, say several thousand, then there is no interference from undesired sparking.



FIG. 5A.—POSITIVE BRUSH (LEFT) AND NEGATIVE DISCHARGE (RIGHT)

It is known that a Rhumkorff gives a secondary current of very peculiar wave form, but that is no objection.

ELECTRODES

Shape and Size of Electrodes.—There are many variations to be tried. First let us take two flat surfaces, the discharge invariably will take place at the edge because here the situation differs from that at other points, the electric charge being greatest at the edge. In the center of said flat surfaces there can be no discharge whatever, while from the edges an irregu-

lar and rather poorly developed brush may occasionally start.

The next experiment is to take two spheres. There can be no brush discharge under ordinary circumstances of operation. If the spheres are of equal size, the equipotential surfaces have the form of spherical shells or rather ellipsoidal shells, quite unfit for our purpose.

Two parallel wires are unsuitable for the purpose of the brush discharge for the same reason, namely, the fact that the equipotential surfaces are coaxial with the wires.

Two crossing wires may give rise to a brush, but will not do so unless very carefully arranged. As a rule a wire is a very bad shape for a brush and this is fortunate, since this is the reason why after all the corona loss does not amount to very much even on very high tension.

Two points cannot give off a brush discharge, unless under very special circumstances; it is very difficult to obtain a brush and it is impossible to work with it. All these facts are easily told, yet, until we understand the exact conditions of the true brush, they seem puzzling.

If we refer to Fig. 1 it is easily understood that the arrangements of electrodes just summed up cannot give a brush discharge. Since the origin and end of the discharge are different, the two electrodes should also be made different; that is,

we must not take two electrodes of equal but of different shape.

Let us now consider these cases and also take into consideration another circumstance, viz., that of sign or polarity.

A great many combinations could be considered, but as none of them except one will answer our purpose we can be very brief about the others. It is of no use summing up all of the combinations; just a few may be enough to bring out the evidence clearly.

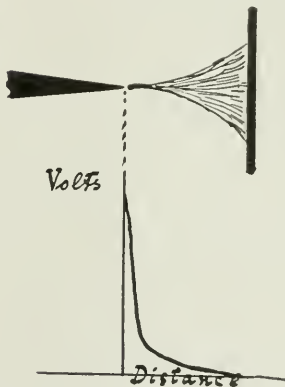


FIG. 6.—BRUSH DISCHARGE AND POTENTIAL GRADIENT

A positive plate and a negative sphere give no brush.

A positive plate and a negative wire give no brush.

A positive plate and a negative point give no brush.

A negative plate and a positive sphere give a brush.

A negative plate and a positive wire give a brush.

A negative plate and a positive point give a beautiful brush.

(The reader will permit us for the sake of brevity not to dwell upon the fact that, of course, such terms as sphere and point require a more precise definition, but as the boundaries of discharges are just as vague as those of shapes, which after all differ only in degree of curvature, there is no need of further definitions and we trust on the reader's good will for understanding.)

So much on the question of shape. Evidently there is no choice left. The best arrangement is to have the discharge from a pointed toward a flat surface, and if we make a drawing of the potential gradient that is likely to occur it will be clearly like the curve shown in Fig. 6.

It might be said here that the photograph of the true brush in Fig. 1 has been taken from one in free air from a small sphere against free air, no opposite pole being provided so as not to influence beforehand the free development of the brush.

In actual work one uses alternating current because direct current of very high tension (say 10,000 volts) is not so easily available as is alternating current. As it is only in the half-period in which the point is positive that it discharges a

brush, the negative half-period is lost, not with respect to energy, but with respect to time.

There is a simple way to make up for this, shown in Fig. 7. Here each discharger proper is intended to emit the electric discharge from its sharp edged side when positive and to receive it on its flat back when negative (if we may express ourselves in a popular way). There evidently is no gain at

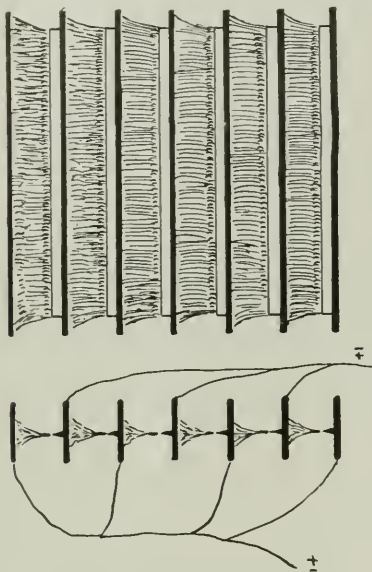


FIG. 7.—ARRANGEMENT OF DISCHARGE

all except in the reduction of the size of apparatus for a given capacity.

The sharper the point the better it is; that means the less voltage it takes to have a discharge. In actual practice I used at first steel needles made especially to suit the purpose. But when I had to increase the capacity of the apparatus from two to twenty kilowatts it could not be done in that way on account of the high cost, as there were 100,000 needles per kilowatt. For this size of ozonator I used No. 35 pure nickel strips toothed like a saw (36 teeth per inch) and for the other electrode a nickel strip of suitable width, the width depending on the polar distance. In my case the width was $\frac{3}{4}$ in., the polar distance being $\frac{1}{2}$ in.

The size of the dischargers proper is of much greater influence than one might imagine at first thought. The flat surface should be $\frac{1}{4}$ more than the polar distance.

The size of the positive sharp electrode matters little, but

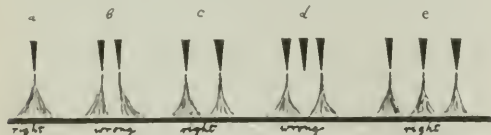


FIG. 8.—SPACING OF DISCHARGES

it should not be extremely short as it should be free from the influence of the electric field caused by its back.

There is a very convenient check to find out whether or not an ozonator has been designed properly. If it is properly designed the energy it takes is directly proportional to the total length of dischargers (or number of points).

This statement will seem self-evident. However, those who have been working in this field know that at the beginning

when the experimenter does not yet know all these details he is every now and then surprised and very often puzzled as to the reason why after an apparatus has been built for say 100 watts, on making it twice as large it does not take twice as much energy.

That will happen when the rules governing the brush discharge are overlooked, due to ignorance. I may say here that it has taken me over 12 years of experimental work and practice to bring the whole affair to a satisfactory end.

We will now describe an instance in which multiplication of discharging points does not correspond to an equal multiplication of wattage.

Spacing of Electrodes.—The spacing apart of the different parallel dischargers proper is an interesting point of great importance. It bears on the principle of electrical shadow.

An example will make clear what I mean.

In Fig. 8 *a, b, c, d, e* are the dischargers proper, mounted in such a way as to allow of easy displacement sideways when under test.

One strip (*a*) will give a perfect series of brush discharges when connected up to the source of electricity, a 10,000-volt transformer.

Now move a second strip near to its side. At a distance from the first strip of say anything above an inch it will work just as good as the first. The energy will be twice what it was before (case *c*).

But when the second strip is moved nearer and nearer to the first one, the discharge is losing in brightness (*b*). Now let us move it away again till all is as bright as possible (*c*). Suppose this occurs at a distance of $\frac{1}{4}$ in.

Put in a third strip, make the distance again $\frac{1}{4}$ in. Then the second discharger will be extinguished and only Nos. 1 and 3 will work (case *d*).

Repeat this with a fourth discharger and so on and it will be seen that any but those at the end are extinguished. The inside dischargers will not discharge at all.

The energy consumed is therefore not proportional to the number of strips, unless these are spaced apart so far that they do not longer interfere with each other's action (case *e*), that is until they are beyond each other's shadow.

Since this spacing depends on the polar distance, at greater polar distance the brush spreads out wider and requires more spacing.

This experiment, which is very fascinating to watch, explains why multiplication of dischargers not necessarily means equivalent increase in capacity. It also makes clear the reason why from a flat surface one does not obtain any discharge from the central part, but only from the edges.

I may once more draw attention to the fact that we are speaking of discharges in free air without any solid dielectric being placed between the electrodes. If this is done, as we shall see later on, the conditions of working are altogether altered.

(3) *Material and Finish of Electrodes.*—It is quite immaterial of what material the electrodes are made. I tried nearly all the more easily available metals and never found the slightest difference.

For the purpose of ozone making platinum would be the best, or gold, but, of course, these are out of question on account of excessive cost. Silver is rapidly corroded by ozone, and so are aluminium, brass, zinc, iron, but copper stands well and so does nickel.

But besides resistance to corrosion there is another property to be considered, viz., that of hardness. It is a very curious phenomenon that the perpetual bombardment of the negative electrode should result in an actual wear. After a year's use the flat nickel strip shows flaws, it has the appearance of a skidded rail (in miniature). But one would not expect wear on such a hard metal as nickel by bombardment of air molecules.

There is no wear whatever on the positive strip so evidently the wear is not through metallic particles of the positive strip. This is important to note for a wear of the positive electrode

would indicate a change in the condition of working. Fortunately this does not happen.

The discharging positive electrode should be sharp-pointed, not sharp-edged. A razor blade gives very poor brushes. There must be real points because the brush must have a support for its stem. Perhaps it is better to say that the stem of the brush must have an inducement to build itself up, this inducement being provided by discontinuities in the line made either like teeth of a saw or as true points (needles).

The finish of the negative electrode should be smooth, without protruding parts or sharp edges or corners. These ought to be rounded off.

ELECTRIC AND MAGNETIC FIELD

The influence of an electric field is very much disturbing and in large apparatus it may be very difficult to avoid it.

We have already spoken of the disturbance caused by the own electric shadow of the discharges, but here we have in view the stray fields caused by the metallic parts of the apparatus.

We have experienced great trouble from this cause and the most convenient way to avoid it was to have part of the apparatus built of wood instead of metal. These stray fields are uncontrollable and cannot be neutralized by proper grounding. By the way, it may be mentioned here that grounding a high-tension circuit is not such an easy thing as it looks.

A magnetic field affects the brush discharge just as it would affect an ordinary alternating current. A strong field tends to "blow it out" and a tendency of sparking is set up.

As to the influence of electric radiation, such as ultraviolet light, this question has not yet been settled. Some people claim a very decided influence of ultraviolet light radiation on the working of brush discharges. I have never been able to detect any at all, but this may have been due to the powerful discharges which I used.

CURRENT SUPPLY

(1) *Source of Current Supply.*—For the study of the discharge, a statical machine is a very convenient source for the current but that is all. Any practical use is out of question for well known reasons.

The use of Ruhmkorff induction apparatus is the next step for an experimenter. The wattage that can be obtained is much higher than that from the former machine, but it is still of no practical use. Full credit may be given to the very interesting experiments one is able to make with it, but it can only be employed for research, not for actual use.

A high-tension direct-current machine, or direct-current on the Thury system has for a long time been my desideratum, but so far I have not been in a position to experiment with this kind of current.

Alternating current has been my regular source of supply, the generator giving 110 volts, the transformer up to 60,000 volts for experimental work. Under practical working circumstances I have used 10,000 volts. (I may call attention here to the fact that at the time when I started my experiments, viz., in the year 1898, such high tensions were an absolute exception in my country, Holland, and the only possibility of using 60,000 volts was to build the transformer myself.)

(2) *Tension of Current Supply.*—It goes without saying that there is an intimate connection between the width of the air gap and the tension of the current. It is easy to understand that the general formula will be $I' = a + bl$ which means that in general the potential difference required to send the discharge through the air gap will be made up of a constant factor a and one depending on the width l of the gap.

One may thus adapt either the polar distance of the dischargers according to the tension of the current, or one may regulate the voltage according to the given distance.

When it comes to ozone making, however, and highest efficiency is sought after, there is no longer any such liberty in choice.

Down to a certain limit, the best yield per kilowatt-hour, as well as the highest concentration (two quantities that are in opposition to each other), are obtained with the smallest pos-

sible air gap, called polar distance. This point has to be considered very carefully.

For the true brush discharge there is a lower permissible limit of polar distance. Under the most favorable conditions this lower limit is about $\frac{1}{4}$ in. for safe working, however, it is hardly possible to go below $\frac{3}{8}$ in., and as a rule we have worked our machines at $\frac{1}{2}$ in.

This settles the question of tension, which may be about 10,000 volts (a.c.). Of course, it should be as high as possible in order to get the highest current density and thereby maximum of wattage, the limit being set by the production of sparks between the dischargers instead of the regular brush

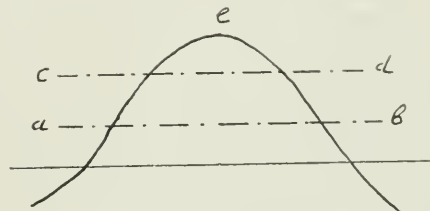


FIG. 9.—EFFECT OF WAVE FORM

(3) *Frequency of Current.*—The danger of sparking can be very materially diminished by raising the frequency of the current to an extra high value. We have tried frequencies up to 600 periods per second instead of the usual 50 (100 in my country in 1898).

The advantage of the possibility of using a higher frequency is counteracted by the fact that the output per kilowatt-hour is thereby reduced. When taking resource to high-frequency Tesla currents, there is hardly a possibility of the occurrence of sparks, but we have the drawback of an enormous decrease in output.

These facts indicate advantages in the use of direct current for ozone making purposes. Perhaps somebody will take this up in future.

(4) *Wave Form.*—The wave form of the electromotive force is of theoretical, rather than practical importance.

Still when we consider the question of wave form, it becomes clear why we can have a brush discharge without dielectrics.

Referring to Fig. 9, let the line $a-b$ represent the minimum voltage at which a discharge will occur. Then, if the top c represents the voltage at which sparking will take place, it is quite clear that if we know of a way to keep the actual voltage between the dischargers below that point c , say at the line $c-d$, there will be no sparking, since the voltage is not high enough for the spark discharge, the polar distance being given

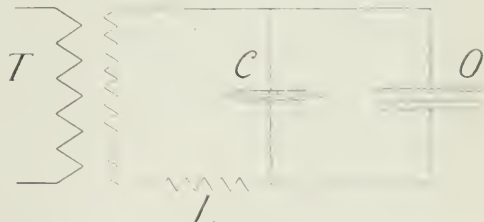


FIG. 10.—CONNECTIONS

That is what is actually being done when we arrange a discharge to take place from sharp-pointed electrodes.

A spark discharge presupposes a certain amount of electrical charge. For just this reason it is not continuous but an intermittent, oscillatory discharge. The brush, however, is a steady flow and for this reason it does not give the electric charge an opportunity to accumulate for a spark; from a sharp point the electricity flows off so easily that there is no chance for a spark.

From this inference it should follow that a peaked curve would give more security against sparking than a flat one. In actual practice the deviation from the true sine wave is not so large as to make any difference for the brush discharge.

(5) *Safety Devices in the Current Supply*.—The discharge from either one point against a flat surface, or from a series of points of any desired length against a flat strip, all of suitable and harmonic design, can be a true brush discharge and will remain so for any length of time, *unless* there comes some disturbance, either caused by a raise of tension, or by a particle of dust or oxide that may cause a spark to be formed.

Now a spark means a lowering of resistance, of course, at once followed by an increase in current density at that spot. This means raise of temperature, causing again a decrease of resistance, etc. Evidently when a spark is once started it will not only remain but will increase.

When we deal with an ozone apparatus of many hundred thousands of points, it would not be possible to avoid all and every disturbance that might happen to come. But as it is of great advantage to have the apparatus work without any attendance at all, some safety device has to be applied and this is what we will now briefly discuss.

Fig. 10 shows a special way of making the connections (patented by me in 1902); in the circuit from the secondary terminals of transformer *T* there is a self-induction *L* in series and a capacity *C* in shunt with the wires that lead to the ozonator *O* which represents a certain resistance.

Through the combined effect of self-induction and capacity the secondary tension will be raised (Ferranti effect) to such an extent that, while the ratio of transformation of the transformer is 70, so that under ordinary circumstances the secondary tension would be 7700, yet in this case it rises to 10,000, but not so unless there is a *certain* resistance at *O*.

Suppose this resistance drops off, say, on account of a spark, then the rise just mentioned from 7700 to 10,000 volts will not take place. The voltage will be less. As soon as this is the case, the cause for the spark has also ceased or the possibility for its maintenance has vanished, the result being that any spark just about to start or already started, will, by its own being, make its continuity impossible. It checks its own existence and it does this in a most beautiful and startling way with surprising accuracy.

The raise of tension is the greater the more the product *LC* approaches unity. It can never be greater than under this condition.

From a theoretical standpoint it is immaterial whether we obtain the raise of voltage through increase of *L*, or increase of *C*, but when the load is heavy, *i.e.*, the resistance is small, then one should rather increase *C* as an increase of *L* causes too much phase lag.

From the foregoing it will be understood that in the design of a large-sized ozonator every item has to be carefully calculated and the whole system has to be in perfect harmony. If this is so there is no trouble, no attendance is required. When the apparatus has once been adjusted to the point of best working the occasional sparks are taken care of by themselves (some little sparking does not interfere with the proper working and is hardly to be avoided on account of particles of rust on the electrodes).

The principal advantage lies in the fact that with this arrangement the arc discharge is an *absolute impossibility* (even when the tension is raised 20 per cent purposely, to demonstrate this fact).

It should be borne in mind that in this arrangement there is no spark gap, as in the Tesla method, which would cause high-frequency discharge.

Perfect and most strikingly beautiful brush discharges can be obtained in this way, and in an absolutely reliable working condition.

As explained, this way of making the connections has to be considered as a safety device. The discharge itself is not altered at all. It may be just as good when the usual conditions are employed. This brings us to the second part.

Part II.—The Brush Discharge with the Use of a Solid Dielectric

As a result of the desire of having some safeguard against the dangerous arc discharge, several devices have been invented and applied. Since the days of the first laboratory ozonator the principle of placing a solid dielectric between the electrodes has been applied by numberless "patentees." We shall deal with this at greater length and, therefore, will first mention other systems.

The drawback of the use of glass is its liability to crack and that is why many have endeavored to do without it.

It was in 1894 that Schneller announced that he was able to obtain the brush discharge without making use of any solid dielectric. This was at first considered to be an impossibility, but Schneller proved the soundness of his assertion by constructing a 2-kw apparatus which did the work, to everybody's surprise.

Schneller inserted a "high resistance" in the high-tension circuit, said resistance to be of the order of 20,000,000 ohms and made of a glass tube filled with 80 per glycerine. The principle, though interpreted in an altogether wrong way, had the merit of being quite new and Schneller's experiments were pioneer work.

For his 2-kw apparatus he used seventy-five of these glycerine-filled tubes of 20,000,000 ohms each, in parallel with his 50,000-volt circuit. The discharge he obtained was a beautiful sight but viewed from the commercial standpoint the system proved to be a failure because of its extremely low efficiency.

From the scientific standpoint it is interesting to point out that said "resistance tubes" do not work as ohmic resistance, but as a pseudo capacity. It has taken me quite some time to find that out, but it is beyond question.

It is easy to demonstrate that an ohmic resistance does not do the work but, on the contrary, provokes arcing instead of preventing it. This fact is quite in accordance to the theory of the discharge.

A few years later Schneller took out a patent for the use of enamel as dielectric, but had the shrewdness not to call it such. He said it was used for having a more equal and smooth surface of electrode!

The next thing that came out was the use of high frequency for the purpose of ozone making. Now it happens to be so that the unsurpassed beauty of the Tesla dark discharges has no bearing on their efficiency and that killed the system.

It is true that with this system there is no danger at all from arcing but as after all the object is ozone manufacture rather than display of beauty the system has never entered the market.

Another device was that of Otto, who tried the use of revolving electrodes in order to either prevent arcing or when arcs were formed to extinguish them by drawing them out.

The system never did any good because arcing should not be *corrected* but *prevented*. Arcing causes nitrous compounds to be formed and causes great damage to the apparatus.

After having been experimenting with all known systems and after studying them carefully, I came to the conclusion that in order to obtain the brush discharge and nothing but this particular one, it is not necessary either to use solid dielectrics between electrodes, or a high resistance in series, or a high-frequency current, or revolving electrodes. All that is really necessary is a thorough knowledge of the subject and according to that, we should endeavor to realize in actual practice what theory indicates as advisable.

This has led me to my system and the fact that I admit frankly that my system has not any more *raison d'être* on account of its low efficiency will make it clear to the reader that it is not as a rival or competitor that I now classify all systems as dead ones, except one which is based on the use of a solid dielectric.

The reason why I have been dwelling so long on those others is just to point out the *importance* of the latter one and to prove that its survival is not a mere chance or commercial attainment but the logical outcome of merit.

In order to fully appreciate the glass-dielectric ozone ap-

paratus, it was necessary to learn by experience that it is the best we have at present and probably for the near future.

When using a solid dielectric there is no chance for a spark nor for an arc to set up. The discharge is always right, whatever the arrangement, and there is but one difficulty left, viz., that of failure of the glass.

Many failures on this account are due to overloading. Too much energy on a given surface causes too much heating up and this will always result in a breakdown, sooner or later, but usually sooner.

The perpetual heavy dielectric strain on the glass also causes occasionally or perhaps always, a change in the nature of the glass, which as is well known is very far from being a body in molecular equilibrium. Dielectric hysteresis is another source of heat and danger.

The success of a glass ozonator, evidently, depends therefore, for the greater part on the quality of the glass. This is a physical and a chemical problem rather than a technical one for the present.

The technical side of the glass ozonator is no problem at all for those who are familiar with the business. The failure of so many "systems" is the logical result of ignorance.

The second great advantage of glass-dielectric ozonators is that they allow of a very small polar distance of dischargers, and that means high concentration as well as high output if the current density is great, the difference in yield being up to ten times as much. This puts all the other systems out of service.

Thus it happened that after twelve years of elaborate experimental and practical research, I finally had to give up my system for a better one, the original glass dielectric system. But facts are facts and science takes no notice of personal pride and ambitions.

One advantage I have gained and that is impartiality, since now I have not a system of my own to bring to the front and to recommend and am at liberty to consider the matter from the impersonal scientific standpoint.

There are numberless patents for ozonators. It would fill up a book were we to consider or even mention them. Most of them are valueless, some of them are ingenious.

Some people prefer glass plates to glass cylinders, others do the reverse. Some believe in the beneficial effect of cooling the air, others do not. Some people think much of complicated arrangements, others believe more in simplicity.

There are, however, some details of interest, but as we intend to discuss ozonators on another occasion, we shall not refer to these here. If the dielectric system has so many advantages and but one disadvantage, namely the treacherous character of glass, why not use any of the many other solid dielectrics? That is a question likely to be put forward. It is because no other dielectric will do.

Ozone being such a powerful oxidizing agent, is detrimental to most dielectrics of organic base. Rubber, for instance, is destroyed in a few seconds. So none of that group can be used.

Moreover, the material in question has to possess certain mechanical properties and has to have a certain thickness. For this reason shellac or other varnishes cannot be used as they cannot be had in sufficient thickness.

The whole group of those conglomerates lack homogeneity and for that reason cannot be used. Others, again, like fibre, asbestos, paper, etc., are too porous for electricity, which "eats" its way through.

Then come some minerals. Most of them are not to be had in convenient shape. Slate is also porous. Mica will answer the purpose but poorly, as it cannot bear very well the heating up; its lamellation is a further disadvantage, the layers becoming more and more separated from one another until finally it is not one thickness but several. Micanite is no better.

Bakelite did not yet exist at the time when I tried every one of the dielectrics, so I do not know its behavior.

Of the fused materials there are enamel, porcelain, and vit-

rified silica. The first can never be applied ruthlessly to a surface, and even if it could it would be no good, its dielectric strength being too low and its thickness too small.

Porcelain fails through lack of homogeneity. So does, for the present, the vitrified silica; it is very fragile and its fusion has not been quite so complete as to remove all flaws, air bubbles, etc. It may be, however, that it will play an important rôle in ozonators in future because its quality is rapidly improving every year.

So it is that only glass combines all the requirements of dielectric strength, mechanical strength, chemical strength and variety of shape and thickness.

One more point of scientific interest is the fact that the dielectric strength of ozone is much higher than that of air. If one leaves the ozone in the apparatus so as to get an increasing concentration the tension of the current has to be raised continuously in order to keep the wattage constant. This circumstance puts a limit to the maximum concentration of ozone obtainable; it may be in the neighborhood of 30 grams per cubic meter.

A fact, generally overlooked, is that a large load of ozonators brings a considerable amount of capacity into the circuit which should be checked by self-inductance.

The Metallurgy of Sudbury Copper-Nickel Ores

One of the two important commercial deposits of nickel-bearing ores in Canada is in the locality of Sudbury, Ontario; the other is in the Cobalt district in the same province. The treatment of the Sudbury ores is covered in some detail in a 200-page *Monograph* recently issued by the Department of Mines, Ottawa, Canada. The work is by Dr. A. P. Coleman, and is entitled *The Nickel Industry*.

The Sudbury ores include essentially only four ingredients: a magnetic sulphide of iron, which is practically free from nickel and copper; a sulphide of nickel and iron, which is non-magnetic; a sulphide containing equal parts of iron and copper, which is non-magnetic; and a variable amount of rock matter which may be of several different kinds, in the main non-magnetic, norite or the products of its weathering predominating.

Some of the ores are low in copper and high in iron, and an attempt has been made to convert these directly into ferro-nickel, but the experiment was a failure on account of the presence of too much copper. Magnetic concentration naturally suggests itself as a method of treating an ore which contains a magnetic mineral and several non-magnetic or feebly magnetic minerals, and a great deal of work has been done along this line, but without success. The reason for the failure is disclosed by an examination of photomicrographs, which show that the pentlandite ($2\text{FeS}, \text{NiS}$) and chalcopyrite ($\text{Cu}_2\text{S}, \text{FeS}$) ramify through the pyrrhotite (magnetic pyrite) in a very intricate manner.

At most of the mines a certain amount of hand-picking is done, in order to reject ore of too low-grade to be profitably treated, but no efforts have been made at concentrating this low-grade material. The mechanical separation of the sulphides from the rock can never be complete because of the intimate mixture of the two in the pyrrhotite-norite, into which the solidier ore usually passes. In any event it is unlikely that much concentration will be attempted until the supply of high-grade ore becomes less abundant.

Metallurgical Processes

The usual metallurgy of the Sudbury ores includes four distinct processes. (1) roasting to remove part of the sulphur; (2) smelting in blast furnaces to produce standard matte; (3) resmelting the standard matte in converters to make a matte of 75 or 80 per cent nickel and copper; and (4) the separation and refining of the nickel and copper. The first three processes are carried on in the Sudbury district, and the methods employed do not differ essentially at the different plants. The refining operation is carried on in other countries (United

States, Wales and Norway) and is accomplished by a different process at each place. More or less secrecy surrounds the refining processes.

Heap-roasting is usually adopted for reducing the sulphur content of the ore to a point where it is suitable for smelting to standard matte. In preparation for a roast-heap, a layer of cordwood or dead pine is laid down to a depth of a foot or 18 in., the space between the larger sticks being filled with smaller pieces to make a rather level surface. Small channels filled with kindling wood lead at intervals of 8 or 10 ft. toward the center of the pile, so as to light the whole heap uniformly. Coarse ore is dumped on this bed, making up about two-thirds of the whole. This is followed by medium sized ore, and the pile is then covered with fine ore. The whole pile will contain from 2000 to 3000 tons, arranged in rectangular form, with flat top and sloping sides.

The wood is fired and burns for about 60 hours, after which the sulphur in the ore continues to burn without further aid. From three to four months time is required to roast such a heap, and at the end of that time the sulphur content has been reduced to about 10 per cent and the iron is more or less completely oxidized. In the earlier stages of the roast a good deal of sulphur is sublimed from the interior of the heap and condenses in the upper part, but eventually this is burned away. Some leaching occurs when rains come on, and the colored water running from the heaps gives evidence of the loss of soluble metal. The roasted ore is usually well sintered, and may require the application of explosives to break up the larger masses.

Smelting at Copper Cliff

The metallurgical practice of the Canadian Copper Company, at Copper Cliff is described in the monograph by Mr. D. H. Browne as follows: When properly roasted the ore contains about 10 or 11 per cent sulphur, and is removed from the roast heaps by steam shovel and taken in 50-ton drop-bottom cars to the ore bins. The ore trains which supply the blast-furnaces consist of eight or nine side-dumping cars. The first three carry the coke charge, amounting to from 10 to 12 per cent of the weight of the charge; the next three carry the ore charge, which usually amounts to 9000 or 10,000 lb. of roasted ore; another car carries 2000 or 3000 lb. of Crean Hill ore, which does not require roasting as it contains only 12 to 14 per cent sulphur. Sometimes, if the ore is well roasted, 2000 to 3000 lb. of Creighton green ore can be added to the charge. Furnace scrap and quartz, when used, are carried in separate cars. The quantity of the latter varies from nothing to 2000 lb. per charge, according to the silica in the ore.

If the furnaces are choked by silicious ore or fine ore, lime may be used for a few charges instead of quartz. The coke is dumped in the furnace first, then the ore with its flux, then the scrap or floor screenings. The blast is normally 24,000 cu. ft. free air per furnace at about 25 to 35 oz. pressure. If the furnaces develop blow-holes due to choking from fine ore, the heat is thrown to the top of the furnace and crusts are formed. These may be cleaned out by charges of green ore and lime, which melt easily and carry the heat to the lower part of the furnace, undercutting the crusts, after which they can be easily removed by barring. There is no regular rule for running the blast furnaces, as the charge varies from day to day, and even hour to hour, with changes in the chemical or physical quality of the ore.

The five blast-furnaces smelt from 40,000 to 45,000 tons of ore per month, the amount of green ore varying with the condition of the roasted ore.

Green ore alone could be used in the blast-furnaces, but would produce a low-grade matte and throw an undue amount of work on the converter department, as shown by the following interesting comparison: To make 100 tons of Bessemer matte per day from a 10 per cent furnace matte would require fourteen basic-lined converters; from 15 per cent matte would require seven basic converters; from 20 per cent matte, five converters; from 25 per cent matte, four converters; and from 30 per cent matte, three converters.

Adoption of Basic-Lined Converters

Prior to March, 1911, the Canadian Copper Company had in operation ten stands of acid converters, with shells 84 in. by 126 in. lined with the usual quartz and clay mixture. Each shell consumed 3000 cu. ft. free air per minute at 9 to 11 lb. pressure. On a 36 per cent furnace matte, one lining lasted about eight hours and produced seven tons of finished bessemer matte, 80 per cent copper-nickel. With a 30 per cent furnace matte, one lining produced only 5.3 tons of bessemer matte.

As the amount of metal lost in furnace slag depends very much on the grade of furnace matte made, this pointed to the desirability of making lower grades of furnace matte, with cleaner slags, throwing on the converters every year an increasing burden. The lower grade furnace mattes contained not only less copper-nickel, but also more iron; in a 40 per cent furnace matte, 1 lb. of copper-nickel is accompanied by 0.88 lb. iron; in a 30 per cent matte by 1.28 lb.; and in a 20 per cent matte by 2.4 lb. It is evident that lower grade mattes will require longer to blow than high grades, since the iron must be oxidized and the air capacity of the converters is fixed. One hour and five minutes is required to blow a 36 per cent furnace matte to produce a ton of 80 per cent bessemer matte, while one hour and fifty-five minutes is required for the same operation with a 30 per cent furnace matte. Further, since the amount of matte thrown out of the converter by the blast is about the same per hour, the capacity of the converters on low grade matte is reduced by reason of the greater amount of matte blown out during the longer period of blowing.

Basic-lined converters have been adopted in place of those just referred to, and considerable advantage has accrued from the change. The basic units are larger, and can handle larger quantities of matte. Owing to the manner of construction, there is practically no material slopped from the converter during the blow, and hence less furnace matte is required to produce a ton of bessemer matte. The slag is made lower in silica, which means economy in flux, and the converter slag contains less copper-nickel than in acid converting.

These basic converters are 37 ft. 2 in. long by 10 ft. diameter, outside measurement, running on four tread rings 12 ft. diameter. The opening for escape of gases during the blow is in the center of the cylinder, and there are no tuyeres directly under this opening. There are forty-four tuyeres, 1¼ in. diameter and 7 in. apart. The length inside the lining is 33 ft. 3 in.; the bottom is 2 ft. thick; the back or tyure wall is 18 in. and the front wall 15 in. thick. The roof is a 12-in. arch, and the brick directly around the tuyeres is 24 in. thick. There are two openings in the front wall opposite to, but above, the tyure line. Slag and matte are poured from these openings.

The basic converter takes an initial charge of about 60 tons of furnace matte. About 10 per cent of quartz rock, previously dried, is dumped into the converter and the blast turned on. Blowing is done entirely by the clock; the first slag is skimmed in from one-half to three-quarters of an hour, and while the slag is being poured off, 5 or 6 tons of furnace matte is poured into the converter. After the slag is poured, about 3 tons of quartz is added and the charge blown for another fixed time. The length of the blow, the amount of slag removed, the weight of matte added after each skim, and the per cent of flux required are all factors of the grade of matte and have to be determined by experience. The blowing and addition of matte is continued until the converter contains from 70 to 80 tons of finished product, after which the matte is cast into molds. This may require from 300 to 400 tons of furnace matte, and from 30 to 50 hours blowing time, depending on the matte grade.

Reverberatory Smelting

In order to provide a suitable method of treating an accumulation of flue-dust and fine ore, reverberatory furnaces were built in 1911. Briquetting and sintering had been tried on these materials, but without success, either on account of failure of the method or extravagant cost. The reverberatories

are interesting as being the first designed to burn pulverized coal. Pulverized-coal firing had been tried at Cananea, Mexico, and at the Highland Boy smelter in Utah, but the furnaces were not specially designed for this purpose.

The reverberatories at Copper Cliff are 112 ft. x 19 ft. hearth area. The extreme height inside is 6 ft.; the bottom is an inverted arch of magnesite brick with a spring of 12 in. The foundations were made by building a trestle about 14 ft. high, and pouring slag from this to build up within walls a solid block of slag 10 ft. high above the yard level. On this foundation the furnace walls were built, and slag was poured inside these to form a furnace bottom 2 ft. thick. Tunnels were formed in the slag foundation at either side of the furnace at the yard level, and through these two tunnels the melted converter slag is brought to the furnaces and matte taken back.

The hearth as formed by leveling up the poured slag with concrete, to provide an inverted arch of the same curve as the magnesite lining. On this one layer of fire-brick was laid flat, $2\frac{1}{2}$ in. thick; over this 2 in. of ground chrome ore was laid, and on this the final bottom of 9 in. of magnesite brick, laid in a mixture of ground magnesite and linsed oil. Expansion strips of wood were laid between every six courses, allowing an expansion of $\frac{1}{4}$ in. to the foot. The tap hole is placed to retain about 12 in. of matte in the hearth, so that the bottom is always protected by a pool of matte.

Slag is removed at either side of the furnace, about 11 ft. from the front. The space at the front usually occupied by the slag door, slopes up gradually from the hearth to form a straight outlet for the products of combustion, which pass through dust-chamber flues to the stack.

The most interesting feature of the reverberatory practice is the method of firing. Coal is pulverized to about 200-mesh and delivered to bins at the end of the furnace. From these bins it is taken by five variable-speed screw conveyors 4 in. diameter, which deliver the coal to five burners, dropping it in front of the nozzles which carry air from a fan. The air blast sends the coal into the furnace in the form of a spray of dust which burns like fuel oil. Each burner can be run independently, and the amount of coal or air varied at will.

The *Monograph* from which the foregoing excerpt has been taken, contains also copies of the patents of the Mond, Hybinette, and Orford refining processes, which also have been noted in earlier issues of this journal.

Solvay Foundations

In connection with the fiftieth anniversary of the commercial birthday of the Solvay process, Ernest Solvay has made a number of splendid foundations. Our Berlin contemporary *Chemie*, probably the only daily chemical paper in the world, writes in its issue of September 26, as follows:

In the year 1861 Ernest Solvay, then 22 years old, an employee of a gas plant near Brussels, Belgium, applied for a patent for making soda from brine with the aid of ammonia and carbon dioxide. People thought then, however, that soda made by this process would be more expensive than Leblanc soda and no financier was willing to invest the money for starting a factory.

Only after many efforts, sufficient capital was raised to start a small experimental factory near Brussels. It soon appeared that the new process was essentially superior to the Leblanc process and in 1863 the first factory was built by Solvay at an expense of 136,000 francs (\$27,200).

Today, after 50 years, the Solvay soda factories of Belgium and other countries have 35,000 employees and workmen.

In connection with the 50th anniversary of the starting of the first factory, Solvay has made the following foundations and donations.

Donations were made to every official and employee as follows: Every official received a check to the amount of one monthly salary for every four years of service with the company, every workman received ten francs and an extra check

to the amount of one monthly salary for every two years of service. Three million francs were given by the Solvay Works to the Workingmen's Pension Fund, so that every workman, after 30 years work with the company, now receives a minimum yearly pension of 720 francs (\$144.) Further, every workman will have in future eight days' vacation with pay during the year. Two hundred and fifty thousand francs were donated to the Workmen's University in Charleroi, 500,000 francs for prizes to be awarded by the International Congress of Hygiene, 500,000 francs were given to the chemical laboratory of Paris University, 500,000 francs to Nancy University for a new chair of electrical engineering, 300,000 francs to the hospitals of Ixelles, a suburb of Brussels, 100,000 francs were donated for fighting tuberculosis in Belgium and 75,000 francs for charity work in the little Belgian towns in which the Solvay works are located.

The German Solvay works have donated 3,500,000 marks for their Workingmen's Pension Fund. They also grant to their workmen a vacation of eight days per year with pay and have finally granted 250,000 marks for a scientific prize which is to bear the name of Ernest Solvay.

Russian Heavy Chemicals

(Special Correspondence from St. Petersburg.)

Under the inspiration of the protective policy, which is the governing fiscal policy of Russia, several branches of the chemical industry have been made remarkably successful, the leading ones being the various soda and caustic soda factories. There is little new to be noted with regard to soda in Russia recently. The business, as is known, has developed to such an extent as to make Russian consumers quite independent of foreign sources of supply. This, however, is a luxury which is not particularly relished by the soda consumers, who find that for want of competition they have to pay an enormously higher price to the home producer than they would have to pay to the foreigner were the products of the latter admitted into the country duty free. But that is an old as well as a present-day story. It applies to all countries and need not be insisted on.

It is interesting, however, in connection with this situation, to state on behalf of the soda producers of Russia that whatever their motive may be, the price of soda in the country has been repeatedly reduced lately, so as to respond to a well-sustained agitation on the part of the consumers. As has previously been pointed out, a reduced price which stimulates an increased consumption reacts as a rule in favor of the producer, and figures have already been published and circulated showing that up till the beginning of this year, at all events, that policy had been a very successful one. So successful, indeed, has it been proved to be since these returns were issued that the demand, after having been about equal to the supply, has again outstripped it. The producers, however, anxious to keep control of the market have combined together for the purpose of erecting a fresh large soda factory to meet not only the present increased demands, but also the prospective ones over the next few years, and doubtless they will be successful in maintaining a fair equilibrium between supply and demand in Russia for some years to come with the aid of the new plant to be erected.

It is like repeating what is already world-wide knowledge to say that in bleaching powder, too, the Russian production has come to be quite equal to the demand and under the guidance of such houses as Solvay and Uschloff, etc., it may be relied on to relieve Russian bleachers of all need to apply to foreign chemical works for supplies.

It is a strange thing in connection with the soda branch of Russian chemistry, which has developed so remarkably and so satisfactorily, that the production of salt in Russia should have progressed with very indifferent speed. It might almost be called a stationary industry, particularly if we take into account the immense strides the population of Russia is making in point of numbers and the naturally great increase there must

be in the demand for salt, even for domestic purposes. However, this is the fact. Certain of the salt fields are held to be strangled simply by the syndicates that have been formed to work them, as it was said, in consonance with the demand of Russian industry and the Russian householder. It is possible, though not likely, that a large quantity of salt produced escapes registration, and that the industry may be much larger than represented in official figures; but such is exceedingly unlikely. The suggestion simply arises because of one's inability to explain how it is that such an important essential industry as salt production, should be of such slow growth in a country where the population increases by leaps and bounds, where salt is abundant and where the development of industry, including heavy chemicals, has been extremely active over the last five or six years.

One need only make a cursory reference to the artificial fertilizer industry which has grown very rapidly, but by no means so rapidly as the awakened industrial conscience of the Russian farmer might have led one to expect. If the production at home is proceeding at a much greater rate than before, so is the importation movement, and as the Russian farmer realizes more and more the advantages to be gained from the use of fertilizers, it is obvious that the immense grain and root-growing area of Russia will call for far greater supplies of fertilizers than the home factories can hope to produce, excepting to the extent of, let us say, a fractional supply. It certainly will not be due to Russian chemical enterprise if the home production should increase very largely. It is an enigma, indeed, that in a country like Russia where great chemists are not scarce and where natural resources are not few, with an enormous demand, not at its own doors, but inside its own walls, the fertilizer industry, after a good many years of existence, should not have got beyond the initial stage or little better. It is frequently pointed out, and with reason, that although the raw material is there, it is not quite convenient for manufacturing purposes; but it strikes a student of national and international economics that if the German or English races had been the inhabitants of Russia the phosphates and pyrites of that country had long ago been turned to much better account than they are to-day.

Allied to the fertilizer business, of course, is the industry of coal coking which is extending so rapidly in the South—that is, in the Donetz basin, where the main iron industry of Russia is now located. The coking ovens there equipped with by-product recovery apparatus that might have been counted practically on the fingers of both hands a few years ago, now number thousands and the result is, it need not be pointed out, a relatively large supply of ammonia, which, with sulphuric acid (either produced at home or imported) is converted into sulphate on terms, however, that do not now very strongly tempt the South Russian farmer. Perhaps when he learns to count better he will appreciate its value and not judge it altogether by its invoice cost.

The significance, however, of the by-products recovery in South Russia extends beyond the field of fertilizers and comes closely into touch with the production of finer goods in the form of colors, etc. Time and again word has gone round that companies have been formed for the purpose of establishing factories to extract the more valuable elements from the by-products of the coking ovens, and one or two such have been established; reports on which, however, are exceptionally scarce, and are by no means sufficiently exhaustive to enable one to judge whether the higher chemical industries have got a hold on South Russia or no. One is sometimes led to the suspicion from the reports issued, that the results come rather from the laboratory of the works than from the industrial plant. All the same the raw material is there in increasing quantity; for all the records in coal production in South Russia have been easily broken both last year and this, as also of coke production. There must consequently be a great volume of valuable raw material recovered by the coke makers on which the more complicated chemical industries might easily rest.

From past experience one cannot hope that any immediate

or proximate development of the chemical industry in Russia will result from Russian initiative. Anything that is done in that way will almost certainly be done by German chemists, who have developed from their point of view an excellent system of capturing the Russian market; namely the flotation of Russian companies with a predominating German financial element, for the purpose of constructing in Russia what are euphemistically termed Russian chemical companies; but which in reality are simply manufacturing branches of leading German chemical works.

Manufacture of Aluminium in France

An interesting summary of the present status of the manufacture of aluminium in France is given in *La Revue Electrique* of September 5, 1913.

Five French companies produce aluminium. These are the Société Electrométallurgique Française, the Compagnie des Produits Chimiques d'Alais et de la Camargue, the Société des Forces Motrices et Usines de l'Arve, the Société des Produits Electrochimiques et Electrométallurgiques des Pyrénées, and the Société d'Electrochimie.

These five companies have formed another company, *L'Aluminium Français*, which acts not only as sales agent but is also intended to carry out research work on the applications of aluminium and to devote itself to a propaganda to encourage greater consumption of aluminium. This company also manufactures in its plants at Selzacte, Belgium, and Mennessis. France, considerable quantities of alumina, sulphate of aluminium, etc. It is now erecting at Arrean, in the Pyrenees an aluminium factory and at Chambéry in Savoy, a rolling mill for manufacture of aluminium sheets, etc. At Kremlin-Bicêtre, near Paris, it has a factory for making tubes, etc.

The present status of the five constituent companies is briefly as follows:

Société Electrometallurgique Française.—This is Dr. Paul Héroult's company, formed in 1888—the "dean" of the French electrometallurgical companies. Its capital is 15,500,000 francs (\$3,100,000). In the different plants of the company at Froges, La Praz, Saint-Michel-de-Maurienne, and L'Argentiere (Briançon) 65,000 to 70,000 hp are available during the greatest part of the year. Besides aluminium ingots, bars, wire, cables, tubes, aluminium cooking utensils, and pure alumina, the company produces ferro-alloys, electric steel (Héroult process), and carbon electrodes.

Compagnie des Produits Chimiques d'Alais et de la Camargue.—This company was founded in 1855 and its capital is now 10,500,000 francs (\$2,100,000). Its plant at Salindres (Gard) was originally intended to manufacture soda by the Leblanc process, was later devoted successfully to chlorine manufacture, and began in 1861 to produce aluminium by the purely chemical process of Henri Sainte-Claire Deville. For 30 years it was the sole producer of aluminium. In 1897 it bought the Calypso aluminium plant and later equipped another hydroelectric plant, now producing "considerable quantities" of aluminium.

Société de Produits Electrochimiques et Métallurgiques des Pyrénées.—Founded in 1906, this company has now a capital of 6,000,000 francs (\$1,200,000). Its Ausat plant has a capacity of 16,000 hp and its output is 6 tons of aluminium, 9 tons of chlorates or perchlorates, and 10 tons of calcium carbide or ferro-alloys.

Société des Forces Motrices et Usines de l'Arve.—Founded in 1895, this company has now a capital of 4,100,000 francs (\$820,000). Its Chedde plant has a capacity of 22,000 hp and its maximum output is 6 tons of aluminium and 20 tons of chlorates and perchlorates per day.

Société d'Electrochimie.—This company was founded in 1889 by H. Gall and A. de Montlaur for the electrolytic production of potassium and sodium chlorates and has now a capital of 10,000,000 francs (\$2,000,000). In its numerous plants the company produces chlorates, aluminium, calcium and calcium hydride, sodium, and potassium and sodium cyanides and other chemical products.

Quantitative Spectrum Analysis

III—Industrial Application

By G. A. Shook

We have already considered some of the typical forms of colorimeters and spectrophotometers which may be used for the determination of the concentration of a colored solution, and we will now take up in detail the method of determining the constants necessary for the calculation of concentrations.

The theory of the colorimeter is so simple that it requires no special detailed study, but with the spectrophotometer the case is quite different.

As the essential points can best be brought out by concrete examples we will consider at length some data obtained on simple solutions like copper sulphate and cobalt chloride by means of one of the more complicated instruments, for instance, the Koenig-Martens polarization spectrophotometer. Salts of these solutions may be quickly and easily made up, and they are, therefore, good material for practice work.

A thorough knowledge of a particular instrument, combined with a little manipulative skill in using it, is almost indispensable for the execution of a complicated problem in colorimetry and such experience may be most easily obtained by beginning with the simplest cases.

The data used in this paper is obtained from observations made either by the writer or students of his laboratory, and may give some idea of the accuracy obtainable with limited experience.

Calibration of a Koenig-Martens Polarization Spectrophotometer

There are three general methods which may be used to calibrate a spectrophotometer, and the choice of either must be determined by the conditions under which the instrument is to be used.

(1) When only a rough estimation of the concentration of a solution is required and when only a few samples of the particular substance are to be examined, then it is sufficient to make up a solution of known concentration and to determine the corresponding extinction coefficient for some definitely chosen color.

The absorption ratio may then be calculated, by means of which an unknown concentration may be readily determined.

Consider a special case: A 10 per cent solution of copper sulphate gave a reading on the Nicol scale of 43.2° ($43^\circ 12'$) when the reading of the observing tube was 3.045. The color corresponding to this arbitrary number is green.

The extinction coefficient is therefore

$$e = \log \cot \phi = \log \cot 43.2 = 0.027$$

and the absorption ratio is

$$A = \frac{c}{e} = \frac{10}{0.027} = 370.$$

A 25 per cent solution gave a reading of 40.4° , and its extinction coefficient is therefore

$$e = \log \cot 40.4 = 0.070.$$

Hence the concentration by the optical method is

$$c = Ae = 370 \times 0.070 = 25.9 \text{ per cent.}$$

(2) If an accurate determination is required, then it is preferable to make up several solutions, varying in concentration over a range that will include all the concentrations of the unknown solutions.

Suppose, for example, that the samples to be tested range in concentration from 1 per cent to 5 per cent, then it is well to make up five solutions of 1, 2, 3, 4, 5 per cent concentration respectively, or, if they range from 10 per cent to 50 per cent, make up five solutions of 10, 20, 30, 40, 50 per cent concentration respectively.

The absorption ratio may then be determined for each of the five solutions and the mean may then be used for subsequent calculations.

The data of Table I was obtained from copper sulphate, the results being plotted in Fig. 1.

TABLE I.—OBSERVING TUBE 3.440 (RED) CuSO_4 (DILUTE)

Concentration in %	Rotation of Nicol ϕ	Extinction coefficient e	Absorption ratio A
0.25	42.8	0.033	7.58
0.50	40.5	0.069	7.25
0.75	38.7	0.096	7.81
1.00	36.6	0.129	7.75
1.25	34.1	0.169	7.47
1.50	32.2	0.201	7.50
1.75	30.0	0.239	7.33
2.00	28.1	0.272	7.37

Mean A , 7.51

Now, if we calculate, by means of the average value of A determined from the above data, the concentration of the solution made up to 2.00 per cent we obtain

$$c = Ae = 7.51 \times 0.272 = 2.02 \text{ per cent.}$$

This represents an average accuracy, and if we consider the 0.75 per cent solution whose absorption ratio deviates farthest from the mean, we obtain the following value for its concentration:

$$c = 7.51 \times 0.096 = 0.72 \text{ per cent.}$$

In all of these numerical examples, the concentration of one of the standard solutions is calculated by means of the average value of A merely to illustrate the method and to give a general idea of the accuracy obtainable.

Instead of taking the mean of the five values of A , for subsequent calculations, we may plot e against c on cross-

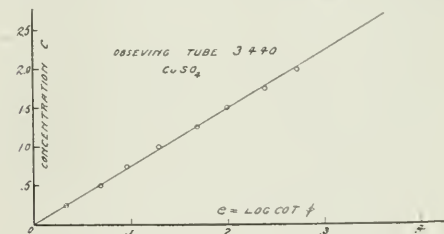


FIG. 1.—CALIBRATION CURVE FOR COPPER SULPHATE

section paper. Such a curve is shown in Fig. 1. The data used in plotting this curve was obtained from Table I.

According to theory this curve should be a straight line and should, moreover, pass through the origin of the axes, but, of course, all the points will not lie on a straight line due to unavoidable experimental errors. If the departure from a straight line is not greater than might be expected we may assume that Beer's law holds for the particular substance, at least for the range of concentrations chosen.

As long as this linear relation obtains, we are not concerned, at least in practice, about the ultimate cause of the color; that is, whether it is due to the molecule, the positive ion or the negative ion.

When such a curve is once constructed, then for future work it is only necessary to set the observing tube to 3.440 and to obtain the reading of the Nicol for the particular solution in question. If the log cotangent of this angle is obtained from a table, then the concentration may be read directly from the calibration curve.

(3) A third method is to construct a table giving the concentration corresponding to every circular degree on the Nicol scale. In this way any calculation whatsoever is avoided, it being only necessary to determine experimentally the angle of rotation of the Nicol, such that a photometric balance is obtained for the particular solution.

This expedient is justifiable whenever a large number of samples are to be tested. The greater the accuracy required the greater must be the number of standard solutions made up. Portions of such a concentration table, which were con-

structed from the above data upon copper sulphate are given in Table II.

TABLE II.—OBSERVING TUBE 3.440 (RED).
CuSO₄ 10 mm. cell

ϕ	c	ϕ	c
20	3.29	31	1.66
21	3.12	32	1.56
22	2.95	33	1.40
23	2.78	34	1.28
24	2.63	35	1.16
25	2.48	36	1.06
26	2.34	37	0.92
27	2.19	38	0.80
28	2.06	39	0.69
29	1.94	40	0.57
30	1.79		

This table was constructed as follows: Take, for instance, the first angle 20° , $\log \cot 20 = 0.439$, hence

$$c = Ae = 7.51 \times 0.439 = 3.29.$$

Now, if a copper sulphate solution of unknown concentration produces a reading of 30° on the Nicol scale, its concentration is found directly from the table to be 1.79 per cent.

The best color, or wave-length of light, to use may be determined experimentally, if more than one standard solution is made up. If the solution is blue, then it will absorb more red light than blue light and consequently the red part of the spectrum should be used, provided that the sample is not too concentrated.

For if it is very dilute the blue part of the spectrum cannot be utilized, since the rotation of the Nicol, ϕ , necessary to produce a balance, will be so small that an accurate value of the extinction coefficient cannot be obtained.

If it is very concentrated too much light will be absorbed in the red and consequently a good balance cannot be made.

These effects are shown in a striking manner by means of the following data obtained from a blue salt, copper sulphate:

Weak Blue Solutions (CuSO₄).
Observing Tube 3.440 (Red).

CONCENTRATION IN PER CENT, c .				
0.25	0.50	0.75	1.00	1.25
ROTATION OF NICOL, ϕ .				
42.8	49.5	38.7	36.6	34.1

This gives a fairly wide variation of ϕ , so that accurate results may be obtained. For high concentrations the absorption in the green was found sufficiently great, as is shown by the following data:

Strong Blue Solutions (CuSO₄).
Observing Tube 3.045 (Green).

c 10	15	25	30	45
ϕ 43.2	42.1	40.4	38.6	36.9

In like manner for weak red solutions one cannot employ the red part of the spectrum, as is seen from the following results:

Weak Red Solutions (CoCl₂).
Observing Tube 3.540 (Red).

c 1	2	3	4	5
ϕ 43.6	42.6	42.1	41.5	40.7

The difference in the reading of the Nicol in this case is obviously not sufficient for accurate work.

The green part of the spectrum was then used for the same solutions with the following results

Weak Red Solutions (CoCl₂).
Observing Tube 3.000 (Green).

c 1	2	3	4	5
ϕ 41.5	35.6	31.9	27.7	22.9

However, for strong solutions of cobalt chloride the red part of the spectrum is best adapted, as shown in the following table:

Strong Red Solutions (CoCl₂).
Observing Tube 3.540 (Red).

c 10	15	20	25	30
ϕ 37.4	35.7	32.9	30.6	27.9

To determine the best position of the observing tube, fill the cell with the weakest solution and obtain a rough balance in different parts of the spectrum, noting the reading of the observing tube for each position. Repeat the same operation with the strongest solution and then choose the position of the observing tube which gives the greatest change in the reading of the Nicol for the two solutions.

What has been said in regard to calibrating the Koenig-Martens instrument may be applied to any spectrophotometer. In the case of the double-slit instrument, one is concerned with the width of a slit and not with the rotation of an analyzing Nicol. Consequently the expression for e will be different.

For work where extreme accuracy is not required the polarizing instrument has this decided advantage over other types, namely, that the relation between the log cotangent of the angle made by the analyzer and the angle itself is approximately linear at least from 45° down to 25° . This simply means that instead of looking up the log cotangent of ϕ , each time, we may simply use ϕ .

To illustrate, the data obtained on copper sulphate, Table I, was used to plot a curve with c and ϕ as coordinates and it is seen from Fig. 2 that the points lie on a straight line which passes through 45° , where the concentration is zero, as would be expected, for if the concentration is zero the fields are equally bright and therefore the Nicol must make an angle of 45° . The extinction coefficient then assumes the simple form

$$e = 45 - \phi$$

whence

$$c = Ae = A(45 - \phi)$$

and

$$A = \frac{c}{45 - \phi}$$

The absorption ratios for the various concentrations are given in Table III.

TABLE III.	
Concentration in Per Cent.	Absorption Ratio.
c	A
.25	0.114
.50	0.111
.75	0.119
1.00	0.119
1.25	0.115
1.50	0.117
1.75	0.117
2.00	0.119
Mean A , 0.116	

The 2 per cent solution gives a reading of 28.1 and therefore by this approximate method its concentration is

$$c = Ae = 0.116(45 - 28.1) = 1.96 \text{ per cent.}$$

An inspection of the two curves, Fig. 1 and Fig. 2, will show at once that for the particular data at hand the points obtained by plotting c and ϕ will not deviate from a straight line any more than the points obtained by plotting c and $\log \cot \phi$.

Of course, it is not necessary to plot a curve if one calculates A for each concentration, for if the variation in the values of A is not greater than might be expected from experimental errors then the exact method need not be resorted to.

If the initial balance is not carefully made then A will not be

constant for different values of e , although a straight line may be obtained by plotting c and e . The straight line thus obtained, however, will not pass through the origin.

If it is found that A is not constant for a particular set of standard solutions it is best to plot a curve before concluding that Beer's law does not hold, for the variation in A may be due only to an error in the "zero" reading.

A rule for determining roughly the best wave-length or color to be used when this particular instrument is employed may now be formulated.

Since the entire angular change of the Nicol should be about 20° , the reading on the Nicol scale for any particular concentration may be calculated approximately. Let us suppose we have a range of concentrations extending from 1 to 5 per cent.

It has been shown that

$$c = Ae = A(45 - \phi)$$

and it follows that

$$c/c' = e'/e = c''/c'' \dots = \text{constant.}$$

Now, if $c = 1$ per cent and $c' = 5$ per cent, then since the reading of the Nicol for the 5 per cent solution should be

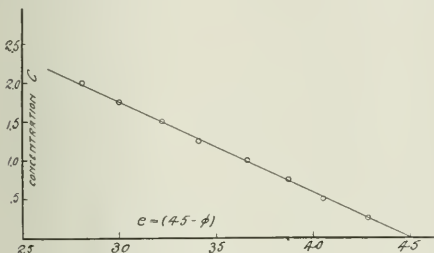


FIG. 2.—CALIBRATION CURVE FOR COPPER SULPHATE (APPROXIMATE METHOD)

about 25° , the reading for the 1 per cent solution must be about 41° for

$$c/c' = 1/5 = (45 - \phi)/(45 - 25)$$

or

$$\phi = 41^\circ.$$

Hence, for the five concentrations, the corresponding readings on the Nicol scale must be approximately as follows:

CONCENTRATION IN PER CENT c .				
1	2	3	4	5
READING OF NICOL SCALE ϕ .				
41	37	33	29	25

The method of procedure would be as follows: Fill the cell with any of the five solutions, say the 3 per cent solution, and set the Nicol to 33° . Now, rotate the observing tube until an approximate balance is effected and note the reading of the observing tube scale.

For example, five concentrations of potassium permanganate were made up as follows: 0.01, 0.02, 0.03, 0.04, and 0.05 per cent. In order to make an approximate balance for the 0.03 per cent solution, it was necessary to move the observing tube to 3.400 (red) when the Nicol was set at 33° .

The following readings were then obtained for the five solutions:

CONCENTRATION c .				
0.01	0.02	0.03	0.04	0.05
READING OF NICOL SCALE ϕ .				
40°	35.8	32.5	28.5	25.1

This general method of determining the proper color may always be applied to the Koenig-Martens instrument whatever

method is used to calibrate it. If one wishes only approximate results and makes up only one standard solution, then the best color may be readily found by the above method.

Concentration of Colorless Solutions

The concentration of a colorless solution may sometimes be determined by means of a spectrophotometer if a reagent can be added which will produce a color or which will cause the solution to become turbid.

For example, if a solution of barium chloride is added to a solution of potassium sulphate, a fine white precipitate is formed by the insoluble barium sulphate. Since all the other salts in the solution are soluble and, moreover, colorless, the density of the precipitate, that is, the number of absorbing molecules per c.c. of the absorbing medium is directly proportional to the concentration of the potassium sulphate provided an excess of the barium chloride is added.

As long as the precipitant is colorless and soluble an excess of it merely plays the rôle of a solvent. By means of the spectrophotometer we can get a measure of the absorption caused by the precipitate and hence it is readily seen that such an instrument may be employed for this class of analysis.

When the percentage of potassium sulphate in any unknown solution is once determined, then it is only a matter of calculation to obtain the per cent of SO_4 , S, or O, etc.

In coal analysis we are concerned with the amount of sulphur present, while in water analysis with the amount of SO_4 . We might, of course, add an excess of the sulphate and then determine the percentage of chlorine in barium chloride, etc.

We will now consider a few examples. The average value of the absorption ratio, A , determined from three solutions of potassium sulphate was 0.0577. The solutions were made up to 0.02, 0.01 and 0.005 per cent of SO_4 .

For the solution made up to 0.005 per cent the extinction coefficient was found to be equal to 0.089 and, therefore, the optical method gives, for its concentration,

$$c = Ae = 0.0577 \times 0.089 = 0.00512 \text{ per cent}$$

In the same manner data was obtained for the determination of the percentage of chlorine in sodium chloride. A silver salt was used as a precipitant. In order to keep the precipitate in suspension long enough to make the necessary readings rather dilute solutions had to be used. The following data was obtained:

Estimation of Cl in NaCl by means of the absorption of AgCl.

Concentrations based on the per cent of Cl.

c	0.0570	0.0429	0.0286	0.0143
ϕ	38.5	40.6	42.2	43.8

The average value of A from the above table is 0.651.

For the 0.0286 per cent solution the optical method gives the following results:

$$e = \log \cot 42.2 = 0.043$$

and therefore

$$c = Ae = 0.651 \times 0.043 = 0.028 \text{ per cent Cl}$$

We may consider one more example of a similar character, namely, the estimation of calcium in calcium chloride (Table IV). In this case ammonium oxalate is added to the calcium chloride solution and the absorption of the insoluble calcium oxalate is taken as a measure of the concentration of the calcium.

This is a good example of the ability of such an instrument to detect the presence of very small quantities. The concentrations are based upon the per cent of calcium

TABLE IV.—ESTIMATION OF Ca in CaCl.

c	ϕ	c	A
0.00125	42.4	0.030	0.0320
0.00250	30.0	0.002	0.0272
0.00500	33.4	0.181	0.0362
0.01000	24.2	0.347	0.0289
Mean A , 0.0311			

The concentration of the 0.01 per cent solution is from the above value of A :

$$c = Ae = 0.0311 \times 0.347 = 0.0108.$$

The estimation of iodine in potassium iodide is a good example of the determination of the concentration of a colorless solution (Table V).

Iodide of potassium is colorless, and in order to develop a color a reagent was added which would liberate the iodine and not produce an additional color. Dilute nitric acid was used in this case. We might use $K_2 C_2 O_7$ to liberate the iodine, but this reagent gives an additional color. Alcohol was also added to hold the liberated iodine in solution.

TABLE V.—ESTIMATION OF I IN KI.
Concentrations based on per cent of I.
Observing tube 3.000 (Green).

c	ϕ	e	A
0.25	20.5	0.427	0.586
0.125	31.4	0.214	0.585
0.0625	37.1	0.121	0.516
0.0313	39.7	0.081	0.387
0.0156	42.4	0.039	0.402

The data of Table VI upon resublimed iodine dissolved in alcohol is given to show the systematic variation of the absorption ratio A . This change in the value of A is probably due to the oxidation of the iodine into hydriodic acid which is colorless.

Beer's law evidently does not hold in this case and accurate results could not be obtained except for a limited range of concentrations.

TABLE VI.—ESTIMATION OF IODINE IN ALCOHOL.
Observing Tube 3.000 (Green).

c	ϕ	A
0.25	18.8	0.535
0.125	29.5	0.507
0.0625	35.9	0.446
0.0313	39.4	0.367
0.0156	41.9	0.331

The absorption ratio A can almost always be determined by means of data obtained from controlled experiments. For instance, in order to determine the percentage of P_2O_5 in a sample of fertilizer it is not necessary to have a number of samples of fertilizer in order to determine A . This may be illustrated by an example.

The following method of preparing the solutions and the sample of fertilizer to be tested is due to Professor Spitzer, of Purdue University, who has perhaps done more work in colorimetry than any other chemist in this country.

The solutions were prepared from a standard solution of calcium phosphate. The phosphate was first precipitated as ammonium phosphomolybdate. This insoluble precipitate was then washed with dilute nitric acid, to free it from the soluble ammonium phosphomolybdate, and finally dissolved in a dilute solution of potassium hydroxide.

Hydrogen sulphide gas was then passed through this solution, causing it to turn to a clear yellow color. The color depends upon the concentration and it varies from yellow to brown. This solution was then diluted according to the percentage of P_2O_5 required. The data for six different concentrations of P_2O_5 is given in Table VII.

TABLE VII.—ESTIMATION OF P_2O_5 AS MOLYBDIC SULPHIDE.

c	ϕ	e	A
0.00078	44.1	0.014	0.0557
0.00156	42.0	0.046	0.0340
0.00312	40.1	0.070	0.0445
0.00623	35.6	0.145	0.0430
0.0125	28.8	0.260	0.0480
0.02500	17.5	0.502	0.0499

Mean A , 0.0459

A sample of fertilizer secured from the Agricultural Experiment Station was made strongly ammonical and then a slight excess of nitric acid was added according to the directions of the official methods of analysis.

This solution was heated to about $80^\circ C$. and an excess of ammonium molybdate solution was then added and the heat continued for an hour.

The precipitate, phosphomolybdate of ammonia, was then collected on a filter and well washed with dilute nitric acid to free it from any traces of molybdate solution.

It was next dissolved in a boiling solution of 5 per cent potassium hydroxide. Sulphurated hydrogen was then passed through this solution for twenty minutes, causing it to assume a deep red color due to the formation of the molybdate sulphide.

The solution gave a reading of 30° on the Nicol scale, whence its extinction coefficient is

$$c = \log \cot 30 = 0.239$$

and its concentration is

$$c = Ae = 0.0459 \times 0.239 = 0.0110.$$

But 125 c.c. of the original solution represents 1 gram of fertilizer, hence the per cent of P_2O_5 in the fertilizer is

$$c = 125 \times 0.0110 = 1.37 \text{ per cent.}$$

The value obtained by the station was 1.51 per cent. Professor Spitzer has carried out a number of such determinations and his average error is considerably less than this.

It is sometimes more convenient to use actual samples for making up standard solutions than to attempt to make, as it were, artificial standards. For instance, in the determination of the per cent of combined carbon in steel by colorimetric methods, one can obtain from the U. S. Bureau of Standards samples of steel which have been accurately analyzed by chemical methods. These samples range in carbon content from a few tenths of 1 per cent up to several per cent.

We will now consider at length a practical example. A 1 per cent solution of iron was made from a sample of malleable iron containing 3.40 per cent of combined carbon. This solution was then diluted so as to make four more solutions of different concentrations. Data was then obtained upon these five standard solutions by means of a Lummer-Brodhun spectrophotometer of the ordinary form.

The importance of an accurate initial balance is clearly brought out by the first set of data obtained upon these iron solutions.

Both slits were set at 30 and a cell containing water was placed in front of one slit. The lamps were then adjusted until the two photometric fields appeared equally bright and without making any further check upon this balance both slits were opened to 100. The data given in Table VIII was then obtained.

TABLE VIII.—ESTIMATION OF CARBON IN IRON.
Concentrations based upon per cent of iron.
Observing tube 72.0 (Green).

c	R	e	A
.125	58.0	0.229	0.546
.250	51.0	0.292	0.857
.500	43.0	0.365	1.37
.750	33.6	0.474	1.58
1.000	27.0	0.567	1.77

R is the reading of the slit and therefore

$$c = 2 - \log R.$$

One might infer, at first sight, that Beer's law does not hold in this case, since A varies continuously.

If, however, c is plotted against e it is readily seen that a linear relation obtains, hence we may infer that the law does hold.

The cell filled with water, was again placed in position and an accurate balance was obtained by adjusting the slit which was not covered by the cell. The average reading of the slit was found to deviate considerably from 100, as might be expected.

It was then set at 100 and the other slit (i. e., the one covered by the cell) was carefully adjusted until a balance was effected.

To make sure that the balance was accurate the first slit was again reset a number of times and the average reading was found to be 99.88. The other slit, which read 125.2, was left intact and the data of Table IX was obtained upon the same solutions.

TABLE IX.

c	R	e	A
0.125	88.9	0.051	2.45
0.250	76.4	0.117	2.14
0.500	62.9	0.201	2.49
0.750	50.0	0.301	2.49
1.000	38.0	0.420	2.38

It is seen from the above table that the variation in A is not more than might be expected from experimental errors.

The second solution, 0.25 per cent, was slightly turbid, consequently its absorption ratio A is too low and it should, therefore, not be considered in determining the mean A .

The data of Table X, for the determination of the per cent of sulphur was also obtained by means of the same instrument.

Barium sulphate was precipitated from a solution of potassium sulphate by adding an excess of barium chloride. Due to the rapid precipitation of the barium sulphate only a few readings can be obtained before the solution becomes non-homogeneous.

If the cell is shaken and then allowed to stand a few seconds the solution will again become homogeneous and the readings may be repeated. In this manner it was possible to obtain fairly consistent results.

TABLE X.—ESTIMATION OF SULPHUR AS BARIUM SULPHATE.
Concentrations based upon per cent of sulphur.
Observing Tube 72.0 (Green).

c	R	e	A
0.005	69.9	0.155	0.0323
0.010	55.3	0.257	0.0389
0.015	32.5	0.488	0.0307
0.020	28.3	0.548	0.0366

Mean A , 0.0346

It is thus seen, from these few examples, that any first-class spectrophotometer may be universally used for the determination of the concentration of a colored or turbid solution.

University of Illinois,
Urbana, Illinois.

The Leadville Pumping Association, organized to unwater the mines of Fryer hill, Leadville, announces that it has obtained signatures from over 85% of the owners of Fryer hill ground, agreeing to the terms of the proposed scheme of unwatering and operating the mines.

The International S. & R. Co. now has five blast furnaces in operation on lead ore at the Tooele plant. A feature of construction of furnaces at this smelter is the use of two rows of jackets, the advantages of which are that accretions are less likely to form, and are more easily barred when they do form.

Commercial Relations of the United States.—A concise volume, which contains statistics showing the foreign trade of each country of the world during 1911 compared with the previous year, has just been issued by the Bureau of Foreign and Domestic Commerce at Washington. It shows the principal articles and their value entering into the trade of each country and gives itemized figures for the imports from and exports to the United States. The statistics were prepared by American consular officers, and supplemented by other official data. In addition to trade statistics, the grain crops and mineral output of the principal countries are given, thus presenting in compact form the principal features upon which the commerce and industries of the foreign countries depend. Copies of the book may be obtained from the Superintendent of Documents, Washington, D. C., for 35 cents each.

Condensation of Zinc Gas to Liquid, in the Presence of Inert Gas.

By F. L. Clerc

The difficulty in collecting distilled zinc in the liquid state, reported by numerous experimenters with new furnaces, indicates that the problem of condensing zinc from a gas to a liquid, in the presence of inert gases, is one of general interest. The zinc obtained has been solid, in the form of powder, and more or less oxidized. It is usually described as "blue-powder." In a former paper I have shown how indefinite this term is, and how important it is that there should be a more exact classification of the various products. Some zinc in solid powder is produced in all industrial zinc furnaces, and it is not probable that any furnace can be designed which will entirely avoid its production, or will reduce it to negligible proportions.

The problem of immediate interest is, to get as much of the zinc in the liquid state as possible; and, what is quite as important, to prevent the zinc dust, which is invariably formed, from obstructing the apparatus and interfering with or even putting a stop to the operation. The rapidity with which zinc gas is evolved by some furnaces, is given as a reason why the zinc is not collected from them as a liquid. It is probable that apparatus could be devised to effect condensation in spite of a rapidity of volatilization far exceeding that observed in any furnace working on an industrial scale, if uniformity in the rate of volatilization could be attained. That is, we should rather modify the relative size of condensers, than restrict the rate of distillation to conform to present practice. The real difficulty comes from the irregularity, and not from the rapidity, with which zinc gas is evolved.

It has been generally known for a long time that both chemical and physical causes contribute to the formation of zinc dust. Mr. W. McA. Johnson has proposed the names "physical" blue-powder and "chemical" blue-powder, to distinguish forms of different origin. The suggestion has value, if only to keep in mind the joint action of both chemical and physical forces. If he will still further inform us how these two forms can be distinguished, one from the other, and whether he has observed intermediate forms, he will furnish criteria by which we can diagnose the cause of any particular failure, by examining the blue-powder produced.

The Effect of Oxidation

In the meantime, admitting that both chemical and physical causes often act together, it may be well to consider them separately. The chief chemical cause assigned for the formation of blue-powder is oxidation. But there is a great variation in the amount of oxide present in different samples of commercial blue-powder, and it is difficult to know with certainty that oxidation has not occurred after the blue-powder has formed; that is, during its removal from the condenser and prior to the analysis. If it is argued that the least amount of oxide present in commercial samples is sufficient to account for the formation of the powder, and that greater amounts found in other samples were formed after solidification, this question can be tested by experiment.

In the sample of blue-powder mentioned in my former article, referred to above, which was melted in a Berzelius arsenic bulb, there was evidently not enough oxidation to prevent coalescence. Moreover, Dr. E. Haanel has informed me that he has melted tons of blue-powder, after mixing it with carbon in proper form and amount. It is not probable that any zinc oxide was reduced in this operation. The explanation is that the air between the dust particles acted on the carbon at about the temperature of melting zinc, forming carbon monoxide which prevented further oxidation of the zinc. Evidently, then, there was not sufficient oxide present originally to prevent melting the blue-powder. Some other contributing cause than oxidation must, therefore, be found for the formation of blue-powder. The question arises: Is it possible by chemical analysis to determine what amount of oxidation will

prevent coalescence, or to devise some other test to distinguish "chemical" from "physical" blue-powder?

The oxide formed, being lighter than the metal, is supposed to float on the surface of the liquid globules, and, when present in sufficient amount, to form a protective coating over them. It is interesting to consider how the amount of oxide required to form a protective coating over all the surface of the globules, will vary with their size. The volume of a given weight of liquid zinc is the same, other conditions being equal, whether it exists as one sphere or any number of spheres; but the combined surface of the smaller spheres will exceed the surface of the single sphere. It follows that an amount of oxide just sufficient to paint a single sphere, will not effectively cover two or more spheres, the combined volume of which is equal to that of the larger sphere. Evidently then, if some oxide is present, as condensation progresses in the Belgian condensers and the globules grow by accretion, some size must be reached at which the oxide present is sufficient to form a protective coating. Does this indicate an upper limit to the size of the globules formed, in condensing zinc gas in the presence of some zinc oxide?

Some plausibility is given to this hypothesis by the rather constant percentage of the weight of commercial blue-powder of widely different origin, which will pass through a 200-mesh sieve. I shall, however, offer a different explanation of this uniformity. Other chemical forces than oxidation may contribute to the formation of blue-powder. The presence of cadmium, arsenic, antimony, tellurium, selenium or sulphur may have some effect; but none of them can be considered as generally acting.

Mechanical Forces Considered

The mechanical forces which may produce blue-powder can be considered under two headings: First, atomic and molecular forces; and, second, mechanical forces.

First: Atomic and Molecular Forces.—The force expressed in heat units required to overcome the atomic and molecular forces which hold zinc in the liquid state, is a direct measure of the force which tends to condense it, for on condensation the same amount of heat is given out. But in order that these atomic and molecular forces can come into play, the separate particles must be brought into close proximity. It has been concisely stated that condensation is essentially a cooling operation; but simple withdrawal of heat will not suffice to bring the particles together to its original liquid state.

Condensation a Complicated Phenomenon

Experiments with liquids which boil at a lower and more easily controlled temperature, show that condensation is a very complicated phenomenon, and that the liquid globules increase rapidly in size after their presence in the gas is indicated by the appearance of a nebular zone. Their formation is assisted by the presence of minute solid particles, called nuclei, throughout the cooling gas, around which condensation occurs. When they attain a certain size they fall by gravity at a measurable rate, and collect as a liquid mass at the bottom of the containing vessel. The walls of this vessel act similarly to the nuclei, and doubtless the nature of the walls and of the liquid have some effect.

It is reasonable to suppose that zinc gas in condensing behaves similarly; and that for a complete explanation of condensation we must have a perfect understanding of the laws of diffusion, viscosity, and vapor tension of gases, and of the surface tension, capillarity, adhesion and cohesion of liquids; for all of these forces are involved, either in the collection of liquid globules on the walls of the condenser, or in their union with the liquid mass. Some of them have to be overcome in the operation of condensation; others assist it. As examples of coalescence, distinct from condensation, we may cite the gathering of scattered globules of mercury; the coming together of rain drops on a window pane; and the rolling of drops of water over a wave surface of a larger mass of water, particularly if the latter be salt or muddy, before they merge themselves into the greater mass.

It is fortunate that we do not have to take account of all of these forces, the laws of which are but little known, for the reason that we know they act when the particles are brought close enough together, and that they always act the same way under the same circumstances. Furthermore, we know that a large percentage of zinc gas condenses itself into a single liquid volume, in the small space of a Belgian condenser, and within the very short time it takes the accompanying gas to traverse it.

Effect of Gravity and Mass Action

Second: Mechanical Forces.—The extraneous forces which I shall consider as acting on the zinc globules and accompanying gas, which are to be relied on to bring the zinc atoms and molecules into such proximity that intermolecular forces can come into play, are gravity and mass motion. The action of these forces, while not more mechanical than the action of any physical force, is subject to well-known mechanical laws. Mass motion is due to change of volume by the addition or subtraction of heat; that is, by change of temperature, or change of state accompanying volatilization or condensation.

Gravity, of course, is constant; but it is opposed by several resistances. Here is not a case of bodies falling or moving through a vacuum. The solids and liquids fall through a moving fluid, the density of which varies with the temperature. The surface friction of the globules and the viscosity of the fluid all oppose motion. This opposition, however, is not directive; its effect is simply to lengthen the time required to traverse a given space.

In designing a condenser, we can extend indefinitely the time during which condensation is possible, and thus allow time enough for gravity to perform the same work in a large condenser which it accomplishes in the Belgian condenser. Likewise, in the motion produced by change of volume, we need not consider opposing forces; a time allowance can be made. The relative direction and velocity of these two motions determines the path of the globules, and brings them together against obstructing surfaces. The importance of a uniform current of gas, in a direction favorable to bringing the globules within the range of action of molecular forces, is thus very strongly in evidence. Whatever may be the rate of evolution of zinc gas, the condenser may have a cross-section to give a proper velocity and direction to the issuing gas current.

Range of Temperature Unalterable

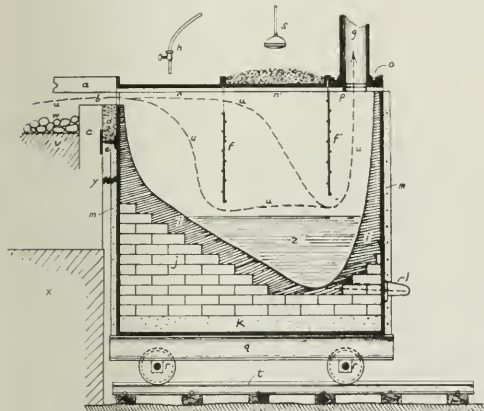
There is, however, one condition which cannot be varied, whether the condenser is large or small, and that is the range of temperature within which condensation is possible. The condenser must be kept at a temperature between the melting and boiling points of zinc, and the gases should cool gradually from their entrance to their exit. It is readily admitted that such close regulation is much more difficult in the case of a large condenser than of a small one. In the case of the Belgian condenser, more than enough heat is carried in by the gases to maintain this temperature under normal working conditions. The same proportion of heat should be supplied by the gases to a large condenser. The Belgian condenser receives heat from the front of the furnace, and radiated heat from the open mouth of the retort. Both condensers lose heat by radiation. A rapid and uniform rate of distillation should lessen the percentage loss from this source. In this connection, also, it must be remembered that the ratio of the surface to the volume is much smaller in the case of a large condenser than of a small one. The effect of these different conditions is to offset each other to some extent. But if the loss of heat in the large condenser is too rapid or too local, precautions must be taken against this, and if necessary, external heat must be applied. This is no more irrational than is the reverse operation of cooling furnace walls with air and water-jackets.

The problem of regulating the temperature of a condenser resolves itself into three simple problems: First, getting it hot enough at the start; second, conserving the heat during normal working, so that the condenser may be as large as

practicable; and third, supplying or dissipating heat locally as desired. The drawing in Fig. 1 will express these ideas applied to a concrete case. It is a condenser designed for use with a copper-bottom retort working continuously.² This condenser should have in front a removable panel with its own lining, in addition to a removable cover to afford ready access to the interior; and it should be nearly red-hot when it is attached to the retort. I think the cut and references will be readily understood, and that the ideas may be applicable to condensers designed for other forms of continuous zinc furnaces.

Suggested Design of Condenser

The length of this condenser will be equal to the greatest length of the retort, and its capacity will be regulated as nearly



CROSS SECTION AT RIGHT ANGLES TO RETORT
(Length of condenser equals greatest length of retort)

• REFERENCES •

- | | |
|---|--|
| a-Cover of retort. | n-Top plate, lined in front of first diaphragm. |
| b-Entrance to condenser. | n-Top plate, lined in front of second diaphragm. |
| c-Outer wall of retort. | o-Trough for fusible metal. |
| d-Sand joint. | p-Top plate, lined, under outlet to washer. |
| e-Rib and wall plate. | r-I-beam supporting condenser. |
| F-Diaphragm of wire-glass with passage top and bottom | s-Wheels and axles. |
| F-Diaphragm of wire-glass with passage at bottom: | t-Water spray (when needed). |
| g-Outlet to washer. | t-Track (for changing condensers). |
| h-Gas burner for heating top of condenser with wasted gas | u-Path of gases. |
| i-Inner lining of refractory material rammed and burned in place. | v-Copper bottom of retort. |
| j-Bottom laid in with blocks | w-Briquetted charge in retort. |
| k-Coarse insulating material. | x-Foundation of retort. |
| l-Spout of tap. | y-Spring clutch holding condenser in place. |
| m-Sectional casting of cast or wrought plates. | z-Bath of molten zinc. |

FIG. 1.—ZINC CONDENSER

Washing the Carbon Monoxide Gas

An iron tank, divided near the bottom into eight downward-pointing compartments, each of which has its own discharge cock, is partly closed at the top by an iron cover, the deep flanges of which are notched to fit the sloping sides of the bottom compartments. The gas is led under the cover through three downtake pipes, each of which receives a shower of water drops admitted, above the entrance of the gas, through finely perforated plates, from a tank containing filtered water. This water, rising between the tank and the cover, prevents the escape of gas. The gas under the cover escapes through the small pipes which encircle the downtakes, and is collected in the gas box above. The downtakes pass through sleeves

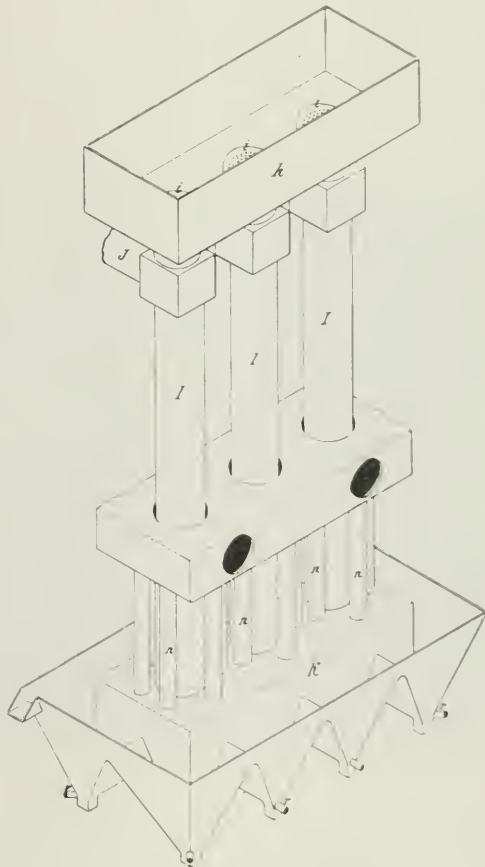


FIG. 2.—WASHER FOR RETORT CASES

as may be to the normal working of the furnace. The gas issuing from the condenser will be nearly pure CO, and of high calorific value. It will be needed to dry and heat up the condensers, and occasionally to blow the residues out of the retort. It will carry out an amount of blue-powder too important to be lost, and which should be recovered by washing the gas. The washed gas should be carefully stored. Its volume will be relatively small; for each ton of zinc reduced there will be formed 863 lb., corresponding to 10.96 cu. ft. of CO, at standard conditions of temperature and pressure. I can suggest no better washer for this purpose than is shown in Fig. 2, which is reproduced from my Belgian patent of July 26, 1876.

inserted between the top and bottom of the air box. The object of this construction is to get as equal distribution of gas as can be had by axial symmetry. The object of the small tanks is to cool and dry the washed gas, and to return to the tank any fume which may adhere to them. Anyone who has thrown very fine blue-powder into water, hot or cold, knows how quickly it settles to the bottom.

The underlying idea of this washer is to avoid eddies and opposing currents in the gas. The falling drops have the same direction as the gas, and will help to regulate the draft. The top of each downtake can be closed by a sagging disc of rubber, which can be lowered to close some or all of the holes in the perforated plate. The water can thus be shut off instantly. Quickly acting dampers should be inserted in the pipes lead-

²This journal, Aug., 1912, page 463.

ing from the retort, so that the operator can quickly close connection between the condenser and the washer. The washed gas should be stored in a gasometer. No washer will prove satisfactory unless it can be shut off from the condenser as quickly as a gas burner can be turned off. I believe that every one will agree with me that the size of the grains of blue-powder carried out of the condenser is limited by the size of particles of zinc which can be transported by the force of the current at its point of exit.

Estes Park, Colorado.

Gold-Milling Progress in California

By Al. H. Martin

The modern stamp mill was evolved from California practice, but California metallurgists have been the most conservative in adopting methods developed by operators in other fields. Stamp-crushing, amalgamation and concentration on blanket or canvas tables completes the ore treatment in a majority of the leading gold-mining districts of the State, and it has only been within the past five years that the cyanide process has been employed to any extent. But recently a more

month, but no attempt was made to secure a better yield until the present owners took charge.

The new mill embraces twenty 1250-lb. stamps, in place of the twenty 750 and twenty 850-lb. stamps formerly employed. The ore is very hard, silicious, and of high milling grade. The crushed product is trammed to the small mill bin, designed to enable a rapid and accurate measurement of bin contents at any time desired. Challenge non-suspended feeders serve the stamps. The mortars are of the high, rapid-discharge type, weighing 8500 lb. each and mounted on foundations composed of concrete made from selected quartz and granite. These foundations were carefully moulded and have given splendid service. Crushing is done in cyanide solution through Tyler ton-cap screens and the pulp delivered to a Dorr duplex classifier. From this machine the coarse product passes to a 5 by 18-ft. tube-mill, and the slime overflow proceeds to the tanks. The tube-mill discharge is returned to the classifier by a bucket-elevator and the product kept in the circuit until sufficiently fine to pass off in the slime overflow.

Air Lifts Supersede Centrifugal Pumps

From the Dorr classifier the pulp continues to a 20 ft. by 10 ft. Dorr thickener. The slime product from the classifier contains 6 parts solution and 1 of solid, and is thickened to a ratio of 2 to 1. The centrifugal pump at first employed to elevate pulp from the thickener to the agitating tanks has been replaced with an air-lift. This was done because of the excessive wear on the pump by the hard and gritty slime. Six hp. is required for operation, and the air-lift has proven most satisfactory. The three agitating tanks are of the Pachuca type, 10 ft. by 30 ft. They are arranged in such a way that any tank can be passed and eliminated from the circuit without interfering with operations. Air is supplied at 40-lb. pressure to agitators, air-lift and filter department by an Ingersoll-Rand 8 in. by 10 in. standard air compressor.

Lead acetate was formerly added to the solution in agitators and stock tanks at the rate of 30 to 40 lb. per working day; but experiments proving that 24 lb. of commercial litharge produced as favorable results, the latter was used instead, resulting in a marked saving in costs. The litharge is added to the tube-mill and agitators in equal amounts. After agitation the pulp goes to a second Dorr 20 ft. by 10 ft. thickener, the overflow from which proceeds to the collecting tank, and thence is pumped to the clarifying and precipitating department. The underflow passes to the filter. As the ore milled averages \$30 per ton, the pregnant solution is high-grade. In order to reduce the gold content of the pulp passing to the filter, the agitated pulp flowing to No. 2 thickener is diluted with the overflow from No. 1 thickener and barren solution by mixing the \$3 per ton solution from No. 1 thickener with the \$8 product of the agitators, and by the addition of considerable barren solution from the stock tanks. In this way the gold content of pulp passing to the Oliver filter is reduced from \$8 to \$2 per ton. The filtrate is combined with the overflow from



FIG. 1.—DARROW-HAMBLIC SLIME CONCENTRATOR

progressive spirit has developed, with the result that thousands of tons of tailings that were formerly wasted are now yielding good profits, and ores that were deemed too refractory to warrant consideration have become sources of excellent profit. Conservatism still reigns in most of the districts, but the stronger companies are gradually adopting the more advanced methods.

Cyaniding at the Black Oak

As an example of the new era in California gold-milling, the recently erected plant of the Black Oak Mining Co. commands particular attention. It is the first all-sliming cyanide plant ever built in the State, and is located at Soulsbyville, Tuolumne county, in the eastern portion of the Mother Lode gold-bearing zone. The mine has been a heavy producer for years, with attention largely confined to the higher-grade quartz. The old mill contained 40 light stamps, and crushing was followed by amalgamation and concentration with vanners and canvas tables. The extraction was never good and thousands of dollars' worth of high-grade tailings were washed to waste every

No. 2 thickener, and the clarified solution is precipitated in Merrill presses by the customary zinc-dust method. The mill extraction has never fallen below 95% in any month, and at times has exceeded 97%. The daily capacity of the old mill using light stamps, was 66 tons. With the 1250-lb. stamps the capacity has been increased to 100 tons.

Improvements in Concentration

The use of canvas tables in concentration is still favored by a majority of California millmen, but progress in this department of milling is also being recorded. The original California concentrator consisted of a wooden frame covered with bullock hides, supported by stringers to give fair rigidity and strength. Later on heavy carpet and specially woven blankets were employed to arrest the gold escaping the amalgamating plates. The employment of canvas later came into favor, and this material is still largely employed in California concentration. Two distinct types of tables are used: the long and narrow, and the broad and short. The latter is generally preferred, because of the ease with which it is operated, but both types have many advocates. The table that has found greatest favor is about 12 ft. wide by 13 ft. long, provided with short lengths of hose at frequent intervals to facilitate easy washing. The tables are built with an inclination of 1 to 1½ in. per ft., and are constructed of 2 by 8 in. stringers with redwood or pine flooring. No. 7 or No. 8 canvas is generally used. At the larger mills the tables are arranged in a double bank.

In early California practice the pulp from the batteries passed direct to the canvas or blanket concentrator, but the method now favored is the treatment of the pulp on Gates or Frue vanners before passing to the tables. In this way a higher concentration is secured. In the Kennedy mill, one of the largest in California, 40 Frue vanners and 56 canvas tables are employed in concentration.

Latest Type of Canvas Table

The tendency in California concentrating is to secure a clean product at a reduced labor cost, and one of the advantages claimed for the Darrow-Hambrick type of table is that it meets this requirement. This table is the latest type of the California canvas concentrator and embraces several features of interest. In place of the usual canvas covering, asphaltic felt or boards coated with asphalt paint are employed to collect the gold. The tendency in concentration is toward the employment of a finer collecting surface, as shown by the gradual change from rough blankets to the smooth canvas surface. The use of asphaltic felt and asphalt-painted and sand-coated lumber in the Darrow-Hambrick machine is a further step toward the fulfillment of the smooth surface requirement by California millmen. In the case of the painted and sand-coated tables the loads are less than when employing canvas or felt, but the product is said to be cleaner and the gold extraction higher. In combination with the asphalt-felt tables their efficiency is stated to be particularly good, and their use in mills along the Mother Lode is finding much favor.

As shown in Fig. 1, the Darrow-Hambrick tables consist of a circular supporting steel frame 20 ft. in diameter, divided into 14 sections, 10 decks high, with 140 distributing boards and an equal number of plane, rectangular concentrating surfaces or trays. Only four of the latter are shown in Fig. 1, extending in front of the distributing boards. Each tray is 3 ft. by 4 ft., giving a total concentrating area of 1680 sq. ft. The surface of the trays is either painted and coated with sand, or covered with fabric or asphaltic felt. These specifications can be modified to give a table 28 ft. in diameter, divided into 21 or 24 sections 6 to 10 decks high, with total concentrating areas of 1512 to 2880 sq. ft.

The circular supporting frame is mounted on a central vertical shaft, with arms radiating to the circumference, and is revolved by a rope drive. A complete revolution is made in from 15 to 20 minutes, according to the desire of the operator. The cycle of treatment during one revolution of the table consists in (1) distributing the slime pulp to the different trays, forming a thin layer over its entire surface, (2)

gently washing the loaded trays with water to remove the light sand, and (3) discharging the concentrate into a collecting launder by means of a strong spray of water from a perforated pipe. The device thus becomes, in effect, a form of continuous and automatic canvas plant.

The free-milling character of most California gold-bearing quartz has contributed to the conservatism prevailing among the millmen of the State, as the ease with which an excellent extraction has been secured by stamp-crushing, amalgamation and concentration created a prejudice in many districts against the employment of more advanced methods. While it has been recognized for years that the tailings from many of the mills carried an appreciable gold content, many operators doubted the ability of the cyanide process or any other method to make a sufficiently large saving to justify costs attending installation and operation of necessary equipment. But the success attending the employment of the cyanide process by the North Star, Empire, Black Oak and other progressive companies has convinced even the most skeptical operators that the method will increase their earnings and many of the companies that have clung to oldtime practices for two generations are discarding their time-honored methods for the improved practice of a newer and better age.

Pitot Tubes for Gas Measurement

In view of the increasing importance of the exact measurement of the flow of gases in the chemical and metallurgical industries, an elaborate paper on this subject, by Mr. W. C. Rowse, published in the September issue of the *Journal of the American Society of Mechanical Engineers*, is of considerable interest.

The pitot tube as a means of measuring gases has the advantage of being correct in principle, inexpensive, portable, and is, in general, easily applied. But the accuracy of the different forms of pitot tubes has long been questioned, the maker of each form supposing that his tube is correct, while as a matter of fact no two forms of tube agree.

The purpose of Mr. Rowse's experiments, which were made

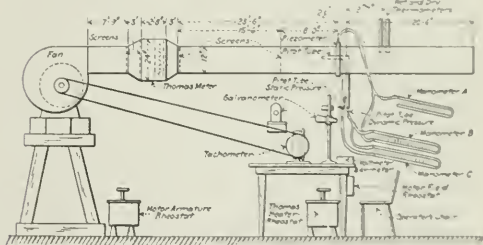


FIG. 1.—ARRANGEMENT OF TESTS

in the laboratories of the University of Wisconsin, was, first, to investigate the reliability of the pitot tube as a means of measuring gases, and second, to ascertain which forms of the pitot tube now in common use give correct and which incorrect results in the measurement of gases.

The method of procedure was to force air through a pipe in which the pitot tube to be tested was inserted, together with a standard gas meter. As standard, a Thomas electric gas meter (described in some detail in our Vol. IX, page 162, March, 1911) was chosen.

The apparatus used in the experiments on pitot tubes is shown diagrammatically in Fig. 1.

A No. 2½ Sirocco fan driven by a 7½-hp direct-current shunt-wound motor forced air through the Thomas meter into a galvanized iron pipe, 12 in. in diameter, in which the pitot tube to be tested was inserted.

Variable resistances were placed in series with both the field and the armature of the motor, thus making possible a wide variation in speed. A stationary tachometer belted to the fan

was so located that it could be observed at all times by the experimenter at the pitot tube. The field rheostat was brought within reach of the operator so that the fan could be maintained at any desired constant speed during a test.

Screens were inserted at the points shown in Fig. 1 in order to break up eddies and whirls and to have the air flow as nearly parallel as possible at the point where the pitot tube readings were taken. All joints between the Thomas meter and the pitot tube were made thoroughly air-tight to prevent leakage. The barrel of the Thomas meter was lagged by three thicknesses of heavy blanket in order to prevent any possibility of error due to radiation to or from the meter casing.

The experimenter was stationed directly under the pitot tube and all readings were taken at this point. A mercury barometer, hung on the adjacent wall, gave the atmospheric pressure, and a manometer inclined at a 10 to 1 slope made it possible to determine accurately the static pressure in the pipe above atmosphere. The dry-bulb thermometer indicated the temperature of the air flowing in the pipe and together with the wet-bulb thermometer gave readings from which the humidity could be determined.

The pitot tube is a well-known measuring instrument and needs only a brief description. It consists essentially of two parts, a dynamic tube pointing upstream, which converts the sum of the pressure energy and the velocity energy into a head which may be measured, and a means of determining the pressure head (or static pressure) alone. The difference between the dynamic head and the static pressure head is the velocity head h in the fundamental formula for the flow of fluids

$$v = \sqrt{2gh}$$

where

v = velocity in ft. per second,

g = 32.2 ft. per second per second,

h = mean velocity head in ft. of the fluid flowing.

It has been satisfactorily proved and accepted that the dynamic tube gives the correct pressure if the tube points parallel to the current. But it is a very difficult matter to obtain the correct static pressure on account of secondary velocity effects. Therefore the study of the accuracy of the pitot tube resolves itself into a correct study of the correct method of obtaining the static pressure at the given cross-section where the tube is inserted.

Each pitot tube tested had as a part of the tube a means of determining the static pressure, and readings of the velocity head were obtained by using this pitot-tube static pressure. Simultaneous readings of the velocity head were obtained by using a piezometer ring for the static pressure together with the dynamic tube of the pitot tube under test. This is further illustrated by reference to Fig. 1. Manometer B gives the velocity head by using the piezometer static pressure, and manometer C gives the velocity head by using the pitot-tube static pressure. In both cases the same dynamic tube pressure is used. This piezometer, even if it were possible to be in error, would always indicate the same pressure under the same conditions irrespective of the tube under test and thus it afforded an additional means of comparison independent of the Thomas meter.

The piezometer is shown in Fig. 1 and is simply an air-tight annular space about the pipe, connected with the interior of the pipe by six small holes 0.04 in. in diameter.

The velocity of a gas flowing through a pipe is much greater at the center than near the walls of the pipe. In addition, it was apparent from the tests that the gas flows through the pipe with a wave or spiral motion even when many screens are inserted to straighten out the stream lines. Therefore, it was necessary to take a large number of readings across two diameters of the pipe. The total area of the pipe was divided into five concentric annular areas and four readings of the velocity head were obtained in each test at the center of each annular area, thus giving 20 readings from which the mean velocity head could be calculated. Since the velocity varies as the square root of the velocity head it was necessary to average

the square roots of each of the 20 readings, and the square of this average represented the mean velocity head. Readings were also taken at the center of the pipe in order to determine if any definite relation existed between the mean velocity head and the velocity head at the center of the pipe.

The pitot tube under test was held in place by means of a brass bushing or holder which could be fastened by screws to brass facings soldered to the outside of the galvanized air pipe. The facings were 90 deg. apart, thus permitting readings to be taken on both the horizontal and vertical diameters.

The principal characteristics of the various pitot tubes tested are indicated in the table given in the summary at the end of this article.

The author proceeds to describe in detail the gages for measuring the velocity heads, and then gives an account of the procedure in making the experiments and calculations. The results of the tests are given in a long series of tables and diagrams.

The principal results are summed up in table 1, in which the terms M , N , Q , U , Z have the meaning explained by the following formulas:

$$C_1 = MC_2 \text{ or } M = \frac{C_1}{C_2}$$

$$C_1 = NC_3 \text{ or } N = \frac{C_1}{C_3}$$

$$C_3 = QC_2 \text{ or } Q = \frac{C_3}{C_2}$$

$$U' = \sqrt{2gh_1} = \sqrt{2gUh_3} \text{ or } U = \frac{h_1}{h_3}$$

$$U' = \sqrt{2gh_2} = \sqrt{2gZh_4} \text{ or } Z = \frac{h_1}{h_4}$$

where

C_1 = cu. ft. of air per minute by Thomas meter

C_2 = cu. ft. of air per minute by the pitot tube using the pitot static pressure

C_3 = cu. ft. of air per minute by pitot dynamic tube and the piezometer static pressure

U' and U = velocity of the air flowing in ft. per sec.

h_1 = mean velocity head in ft. of air flowing, obtained by the pitot tube using the pitot static pressure

h_3 = velocity head in ft. of air at the center of the pipe obtained in the same manner as h_1

h_2 = mean velocity head in ft. of air flowing, obtained by the pitot dynamic tube and the piezometer static pressure

h_4 = velocity head in ft. of air at the center of the pipe, obtained in the same manner as h_3

M and N are, therefore, coefficients by which the actual results obtained by the pitot tube must be multiplied to obtain the correct flow of gas. Q gives the coefficient by which the results from the pitot tube alone must be multiplied to obtain the same discharge as that shown by the pitot dynamic tube and the piezometer. U' and Z state the relations between the velocity head at the center of the pipe and the mean velocity head as determined by the two methods used in the experiments.

(The significance of the figures in table 1 will be clear from the above definitions. For instance, if we want to know what type of pitot tube used alone for getting the static pressure gives the highest accuracy, we have to look up the figures for Q which represent the coefficient by which the results from the pitot tube alone must be multiplied to obtain the same discharge as that shown by the pitot dynamic tube and the piezometer. The nearer to unity the value of Q , the more exact is the pitot tube in question. From the table we see that Q is nearest unity in case of pitot tube V , the standard tube of the American Blower Company. The value of Q in this case is 0.9992, the error, therefore, only 0.08 per cent.)

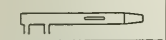
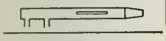
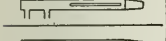
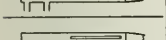
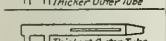
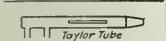
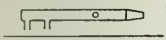
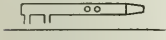
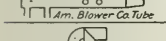
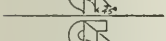
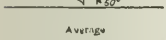
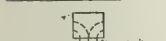
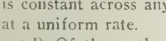
From a study of the summary, table 1, Mr. Rowse draws the following general results and conclusions:

(a) The pitot tube as a means of measuring gases is reliable within approximately 1 per cent when the static pressure is correctly obtained and when all readings are taken with a sufficient degree of refinement; in order to obtain this degree of accuracy the pitot tube should be preceded by a length of pipe 20 to 38 times the pipe diameter in order to make the flow of gas as nearly uniform across the section of the pipe as possible.

(b) All the methods of obtaining the dynamic head used in these experiments, including the German Stauscheibe (*S*), give accurate results.

(c) The most reliable and accurate means of obtaining the static pressure is the piezometer or its equivalent, the results of 138 separate tests using the piezometer static pressure agreeing with the Thomas meter within an average of 0.33 per cent;

TABLE I

	Name of Tube	<i>M</i>	<i>N</i>	<i>Q</i>	<i>U</i>	<i>Z</i>
	A	1.0614	1.0175	1.0413	0.7856	0.7936
	B			1.0576	0.8019	0.7888
	C			1.0343	0.7996	0.7880
	*C	1.0346	0.9952	1.0384	0.8107	0.8066
	D			1.0369	0.7927	0.7942
	E			1.0489	0.7885	0.7898
	X	1.0987	0.9925	1.1074	0.7656	0.7887
	G	1.0676	1.0085	0.9989	0.8100	0.8164
	H	1.0218	1.0152	1.0065	0.7968	0.7957
	Y	1.0024	1.0017	0.9992	0.8002	0.7996
	K	1.0669	0.9924	1.0626	0.7617	0.8115
	L	1.0104	1.0032	1.0064	0.7593	0.8112
	A		1.0033		0.7884	0.7986
	S	0.9861	1.0016	0.9844	0.7780	0.8228

these results show beyond any doubt that the static pressure is constant across any section of a pipe in which gas is flowing at a uniform rate.

(d) Of the methods of obtaining the static pressure by the pitot tube itself, the most reliable and accurate is by means of a very small hole in a perfectly smooth surface, as in pitot tube Y.

(e) The long slots for obtaining the static pressure are not reliable and give results which are in error from 3.5 to 10 per cent. The length of the slots or the thickness of the outer tube do not appear to affect the accuracy of the tube.

(f) The beveled tube for obtaining the static pressure as used in pitot tubes K and L is not reliable. A very slight change in the angle of bevel produces an appreciable change in the result. In taking a traverse of a pipe the sides of the pipe affect the readings. But the greatest error is produced by the uncertainty as to whether the tube is pointing directly upstream. If the tube is off 20 deg. in one direction an error of 85 per cent in the velocity head is introduced.

(g) The Stauscheibe gives accurate results using either the static reading from the Stauscheibe and a special formula or by

using the piezometer static with the usual formula for the pitot tubes. In the first case the agreement is within 1.4 per cent and in the second within 0.16 per cent.

(h) It appears that an approximate relation exists between the mean velocity head of a gas flowing through the pipe and the velocity head found by placing the tube at the center of the pipe. For a 12-in. galvanized iron pipe results within 2 per cent may be expected from using the formula

$$v = V 2g / 0.80h_c$$

where

v = velocity in feet per second

$g = 32.2$ feet per second per second

h_c = velocity head in in. of gasoline at the center of the pipe obtained in a correct manner.

Electric Furnaces, Their Design, Characteristics and Commercial Application.

By Woolsey McA. Johnson and George N. Sieger.

The Electric Furnace Structurally Considered.

Certain electric furnaces, as the carborundum furnace or the graphite furnace, are not strictly metallurgical structures, for they are simply built up and torn down after each run. This is a perfectly proper procedure, for the high temperature

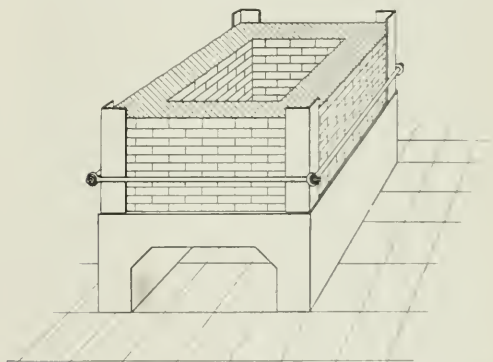


FIG. 1.—SECTIONAL VIEW OF OBLONG OR SQUARE FURNACE SHOWING SIMPLE MANNER OF TIEING. OUTSIDE COURSE IS TO BE LAID IN CEMENT MORTAR UNLESS IN METAL CASE. FURNACE BUILT ON TABLE OF REINFORCED CONCRETE

(2200 C. to 2500°C.) would make any permanent furnace an engineering impossibility at present and it would not be good business with the present handsome margin of profit to bother about the unattainable.

But certain other furnaces which have a direct competition with pyro-metallurgical furnaces must be built so that the cost for repairs and renewals is within fixed industrial limits. Such a furnace must have its two basic mechanical elements—the refractory lining and the structural "iron-work" that holds lining in place,—of a design suitable for its metallurgical needs. The refractory lining must resist the action of such hot slags and metals as it is to contain. The "iron-work" (a somewhat indefinite term used to cover a multitude of virtuous devices) holds the lining and brick work together so that it is a self-contained piece of apparatus and one that will not fall apart.

Just as in any industrial concern we have older men giving advice and acting for conservatism, and young men full of energy and tireless refractoriness, so in a metallurgical furnace we have the complementary parts of firebrick and steel each doing a different duty.

From an idealistic standpoint firebrick work is supposed to possess no mechanical strength, except resistance to crushing.

We must then consider that the bricks are bound together by the inwardly-directed force of compression, which comes originally from the tension rods of the outside framework and that structural coherence comes from friction. Practically it is possible to depart from theory, for the strength given by the mortar is sometimes great.

When it comes down to a question of facts, the design and erection of a furnace is largely a matter of practical exper-

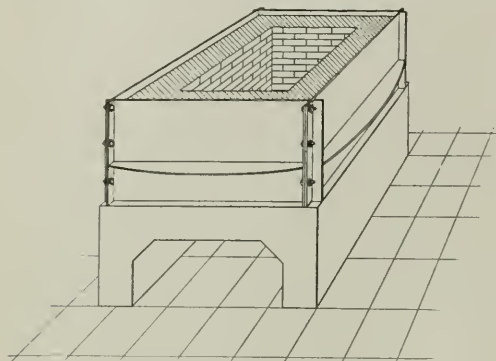


FIG. 2.—SECTION OF FURNACE BUILT OF CAST IRON PLATES ALSO ON TABLE OF A REINFORCED CONCRETE

ience and "get-there-ness." But if interpreted with engineering common-sense, theoretical considerations are of value. Indeed they are always present unconsciously in the mind of many so-called ignorant workmen and it is the wise engineer who avails himself of truth and knowledge from any source.

Undoubtedly the simplest form of "iron-work" for a furnace is a box made of sheet steel $\frac{1}{8}$ in. to $\frac{3}{8}$ in. thick, reinforced on the corners with riveted angle bars. See Fig. 1. This furnace can be securely tied by tie rods made in identical manner to tie rods on a square brick chimney. This form is a good one for an experimental furnace. The brick work is laid inside in dimensions of multiples of $4\frac{1}{2}$ inch.

Cast-iron plates with flanges running both horizontally and

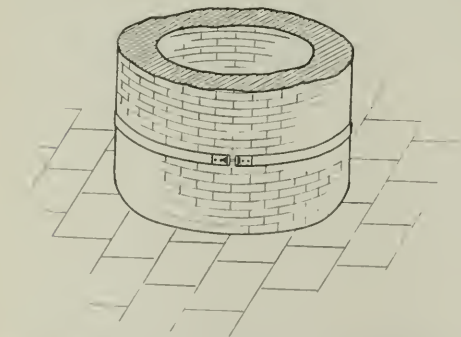


FIG. 3.—SECTIONAL VIEW OF SIMPLE "POT" FURNACE BUILT OF "RADIUS" BRICK OR OF "KEY" BRICK. OUTSIDE COURSE MUST BE LAID IN CEMENT MORTAR UNLESS STEEL SHELL IS USED

vertically makes a better job of this design often, especially if the furnace is a permanent one. See Fig. 2. This specially cast stuff costs more than sheet steel.

Cast steel plates are practically perfect, but their cost is the highest.

Electrodes are inserted in holes. Such furnaces are built of insulated bolted sections.

Cooling of the outside by water is the way when any severe duty is placed on the refractory lining. These water-jackets, spray-plates, or bosh-plates have to be held in place in a majority of cases by the inwardly-acting force of steel rods or steel hoops.

Where a round furnace is used the steel shell is made in the form of a hollow cylinder. Steel hoops with adjustable means of loosening or tightening can be used when great mechanical strength is demanded. See Fig. 3.

An oblong furnace is tied with rows of rods running in all three directions and engaging buck-stays. See Fig. 4. Buck-stays are made of cast iron, I-beams or old rails. A buck-stay is a mechanically strong piece of metal backing up the brick work and taking the outward thrust of expansion.

Brick walls laid in cement mortar or reinforced concrete walls can be used for the outer walls, provided the metallurgical duty is not severe. This makes the outer wall one-piece.

In any case we must conceive of the lining holding the contents, the outer walls holding the lining and the "iron-work"

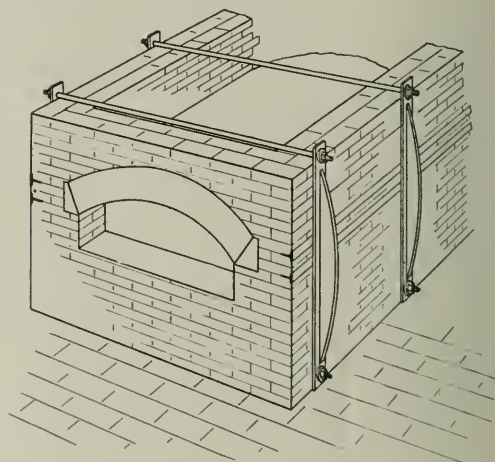


FIG. 4.—FURNACE WITH CAST-IRON BUCKSTAYS AND TIE-RODS RUNNING AT RIGHT ANGLES TO TAKE THRUST OF ARCH

holding the outer walls. These things must be grasped by the mind at the same time. The relative duty and the association of different functions of the parts operate in good design for a common end. For instance, the ultimate force is the tension on the tie rods, coming from the turning of the nuts or twisting the turn-buckles. Opposed to this is the expansion of the firebrick on heating. These forces must be in equilibrium. Otherwise distortion or possibly rupture ensues.

Proper allowance for this fact must be made in design and the furnace must be watched on heating up and tie rods slackened. Probably the simplest way to compensate such forces is by the use of spiral steel compression springs acting on ends or in middle of tension rods. In such a manner the force is progressively compensated and without trouble and without care on the part of furnace-man and engineer. Were it not for increased expense, the authors would never design any furnace without use of spiral compression springs. Accordingly, where question of expansion is serious, springs on rods can be used. Fig. 5 shows one manner of using compression springs.

Tie rods are usually made by welding bightened "bolt-ends" to steel stock, but the threads can be cut if convenient. Turn-buckles and clevis nuts can be used but do not seem to be as serviceable for all 'round work as nuts and bolts. In welding a bolt-end to a tie-rod allowance must be made for the fact that the cut of thread has diminished the effective area of bolt-end

by just so much as twice the distance from the outer circumference to the root of thread. To use square $\frac{1}{2}$ -in. rods to a $\frac{3}{4}$ -in. bolt-end, square $\frac{3}{4}$ -in. rods to 1-in. bolt-ends, and so forth, is a practical rule. In spite of proper personal pride of the blacksmith, the weld has only the strength of from 40% to 80% of original stock. The weld should be a "scarf weld" near the bolt-end made so as to thicken up the stock as much as possible.

The preceding description can be summed up as follows:

(1) There should be a lining that is refractory and this lining must be held in place by compression.

(2) There should be a shell or outer wall that distributes this compressive force uniformly to the lining.

(3) The "iron-work" should receive this inwardly acting force from the strain of tension of steel.

(4) All of the above must be so proportioned that each does its proper work.

When the above is achieved, the furnace is perfect.

Electro-Metallurgical Idiosyncrasies

In general, an electric furnace for conditions below 1600°C. resembles in its construction ordinary pyro-metallurgical fur-

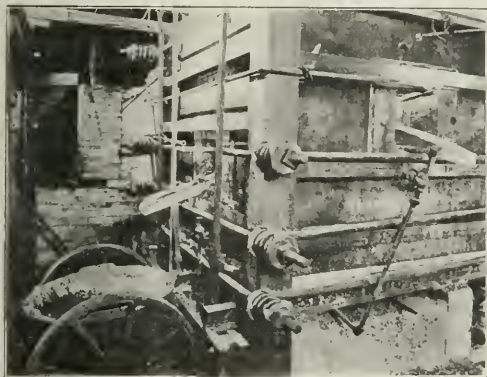


FIG. 5.—ELECTRIC FURNACE CONSTRUCTION SHOWING USE OF COMPRESSION SPRINGS

naces. There are three differences, however, one electro-metallurgical, one electrical and one magnetic in cause.

If at any time the furnace does not functionize, or if too much electrical energy is applied, the temperature of the contents will rise in short period to an extraordinary degree, before the radiation or smelting will cool it down. This is liable to happen at any time. Accordingly the lining must be able to stand self-abuse and extraordinary hard work.

So also with electric current; the tie rods and steel are conductors and if accidentally in contact with both sides of power line, a bad short circuit ensues. By inserting in convenient places $\frac{1}{4}$ in. strips of John B. Pratt Co.'s "Vulca-beston" with bearing plates of $\frac{1}{4}$ in. bar iron this cannot happen. This is a wise precaution although the firebrick alone is generally enough of an insulator.

Most electric furnaces employ alternating current. The "iron-work" therefore, must be broken where possible by air gaps, by brick or by "Vulca-beston," with structural strength still preserved. This point can be seen in the design shown in Fig. 5. Otherwise, the inductive drop due to voltage absorbed for no other purpose but to magnetize the "iron work" with each alteration becomes commercially important and power factor especially in large units drops to low figure.

Refractories

The subject of refractories is a large one and also a peculiar one. The backward state of this art can be well seen by the

fact that the same substance, aluminum silicate, is used for firebrick to keep in heat and as a heat insulator, and for retorts, muffles, and crucibles to bring in heat, which should be heat conductors. (Cf W. McA. Johnson, *Electro-chem. & Met. Industry*, Vol. III, page 214 and Vol. IV, page 6.)

But at present the great amount of work which is being done in metallurgical and chemical engineering circles on the subject of refractories is sure to bring eventually discoveries of great commercial worth. If some practical scientist should invent and perfect a refractory, inert to usual metallurgical substances as slags at a temperature up to 1700° C. and a fair non-conductor of heat and electricity at that temperature, he would be performing an invaluable service to the profession. Furthermore, he might make a fortune.

The general kind of refractory is made of silicate of alumina and goes by the name of common firebrick. This is manufactured of two physically different forms of material. (1) plastic or hot clay and (2) "grog," i.e., coarse particles of crushed burnt fireclay. The former material gives firebrick strength, the latter gives them cohesive powers.

Firebrick vary in refractoriness depending on impurities in clay on manufacture and physical distribution of impurities. In general, it can be said that any substance that tends to make a blast furnace slag more liquid, as soda, iron oxide, or lime, diminishes the refractoriness of firebrick. Good firebrick will stand a heat of 1475°C. to 1525°C., while "rebrick" are made that will stand 1600°C.

Firebrick at any temperature will not long stand hot liquid slag, especially in an electric furnace, unless they are cooled in some manner. In the latter case they simply are eaten away until the cooling is such that a protective layer of congealed slag stops further corrosion.

Firebrick are made in many shapes and sizes. Ordinary stock shapes are as follows:

Straight	9" x 4 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ "
Soap	9" x 2 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ "
No. 1 Split	9" x 4 $\frac{1}{2}$ " x 1 $\frac{1}{4}$ "
No. 2 "	9" x 4 $\frac{1}{2}$ " x 2"
Large Straight	9" x 6 $\frac{3}{4}$ " x 2 $\frac{1}{2}$ "
Plate	12" x 12" x 2"
Slab	23" x 5" x 3"
Slab	28" x 8" x 5"
No. 1 Arch	9" x 4 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ " to 2 $\frac{1}{8}$ "
No. 2 "	9" x 4 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ " " 1 $\frac{3}{8}$ "
No. 3 "	9" x 4 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ " " 1 $\frac{1}{8}$ "
"Bung" "	9" x 4 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ " " 2 $\frac{3}{8}$ "
No. 1 Key	9" x 4 $\frac{1}{2}$ " to 4" x 2 $\frac{1}{2}$ "
No. 2 "	9" x 4 $\frac{1}{2}$ " to 3 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ "
No. 3 "	9" x 4 $\frac{1}{2}$ " to 3" x 2 $\frac{1}{2}$ "
No. 4 "	9" x 4 $\frac{1}{2}$ " to 2 $\frac{1}{4}$ " x 2 $\frac{1}{2}$ "
No. 1 Wedge	9" x 4 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ " to 1 $\frac{1}{8}$ "
No. 2 "	9" x 4 $\frac{1}{2}$ " x 2 $\frac{1}{2}$ " to 1 $\frac{1}{2}$ "
No. 3 "	9" x 4 $\frac{1}{2}$ " x 3" to 2"

Radius brick 9" x 6 $\frac{3}{4}$ " x 4 $\frac{1}{2}$ " to 2 $\frac{1}{2}$ " are especially convenient at times as well as big blocks 18" x 12" x 8", and "skew backs." Tiles, slabs and plates are made in a variety of other sizes.

With an intelligent mason, furnished with plenty of sharp "skutches" and chisels, almost any furnace can be built of above shapes. Not unoccasionally, however, it is business to have made special shapes, as for instance for a "flat" arch, or for ring around charging hole. But by use of beveled edges and by applying plenty of compressive force many things can be done, if ingenuity and persistence are at hand.

There are three other kinds of refractories that are much more expensive than firebrick, but better for use where high heats are used and special conditions are to be met. These are magnesite brick, chrome brick and silica brick, made respectively of basic, neutral, and acid material. All these brick have high coefficients of expansion on heating and expansion must be liberally allowed for. See Fig. 6.

Magnesite brick are used with a basic slag, for instance, in

modern copper converting or in basic open hearth bottoms. They can be laid in fine dust, milled from old brick mixed with hot tar, linseed oil or water-glass. They are also laid dry. A bottom can be made of old magnesite brick, dust, and water-glass, just as concrete is laid, and if dried slowly

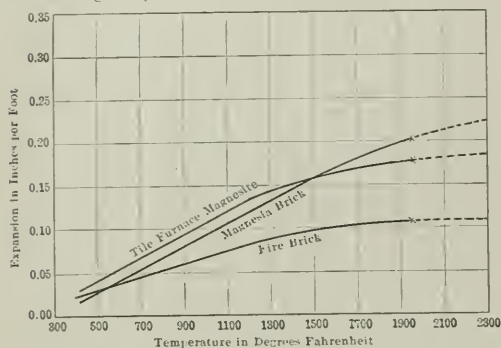


FIG. 6.—TEST OF EXPANSION OF REFRACTORIES

and thoroughly baked it will hold metal fairly well. The water-glass, of course, diminishes the refractoriness of the work, but adds to other properties, such as coherence.

Chrome brick are used at times as an equivalent of magnesite brick. They make an inert insert between courses of magnesite brick and silica brick which might flux together, and also are suitable around tap hole of furnace. Both chrome and magnesite brick conduct heat several (2 to 4) times as well as ordinary firebrick.

Silica brick are laid for arches which are subjected to considerable heat, e.g., in Heroult electric steel refining furnace. Silica brick are made out of special silicious material, e.g., ground sandstone mixed with a little lime "putty" as a bond, and then burned. They are laid in mortar of same material out of which it is made. All three kinds are only made in few stock sizes. They are all apt to be warped to some extent in burning, and are not true to shape or size at all. All these as well as common firebrick are porous to a considerable degree.

Other kinds of brick can be used but are less common, e.g., "bauxite" and "alundum" brick, in special cases to meet special conditions. Of course, in an electric furnace, with its reducing atmosphere and in an electric furnace plant where there are always plenty of stub ends of electrodes, carbon brick are fine. But sooner or later they burn out by accidental or incidental oxidation. They are all right for experimental work or where it is commercial to run a furnace a short time and then rebuild it. They can be laid up in a mixture of tar and carbon dust.

It is well to have several barrels of pugged firebrick mixture bought from a firebrick manufacturer on hand in any small plant. This can be rammed into the interstices when necessary,—such as when changing from "headers" to "rowlock" or

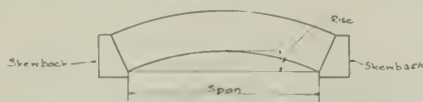


FIG. 7.—SIMPLE ARCH SHOWING "RISE," "SPAN" AND "SKEWBACKS"

"stretchers" the courses do not work even. "Asbestos-retort" cement makes a good filling where not exposed to temperature above 1200 deg. C. Johns Manville or American seal are good brands.

Electrically sintered magnesite with a special bond, which is furnished the trade by the Norton Co. is of value in special places in furnaces with basic lining. It should be tamped in warm with red hot steel tampers.

All brick work should be laid tight, brick to brick, with expansion joints where needed and where designed. Where the firebrick are subjected to severe duty, they should be dipped in thin "fire-clay soup" and tapped with a mason's hammer into place. *The mortar should be same stuff as brick*, very fine and mixed thin with water or other liquid. In certain cases, the brick can be laid with thicker mortar and by use of a trowel. Not unseldom a little cement can be mixed with fire

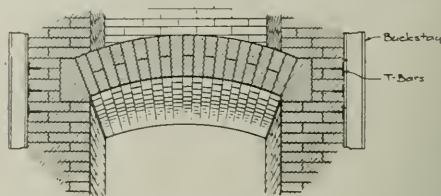


FIG. 8.—SHOWING ARCH AND MANNER OF BREAKING JOINTS IN AN ARCH. ALSO THE BACKING OF THE "SKEWS" WITH T-BARS HELD IN PLACE BY BUCKSTAYS

clay, especially where brickwork is exposed to weather or where mechanical strength is needed. But this is a matter of judgment, for cement diminishes refractoriness.

But, when all things are considered, the use of firebrick and other refractories so as to produce results cheaply and efficiently is an art, not a science,—an art of understanding conditions and ways of making things practical, not a science or an academic theory.

Laying an Arch

The laying of an arch is a matter that requires care and conscience. The first thing to do is to figure out the "sweep." The "sweep" is rise in inches per running horizontal foot of arch. This changes in a circular arch but as most metallurgical arches are built with practically constant curvature, the term sweep can be used all right. The "span" is distance between lower points of "skew-backs." The "rise" or "raise" is distance from bottom of "key-brick" to line running from lowest point of skew-back. See Fig. 7.

The "sweep" of an arch has ordinarily a minimum safe value of $1\frac{1}{2}$ " to $1\frac{1}{2}$ ". But the work must be carefully done and on intermittent service of furnace an arch with sweep of $1\frac{1}{2}$ " to foot or $4\frac{1}{2}$ " "rise" to $7\frac{1}{2}$ " "span" would not stand unless made of key or wedge brick, and unless the "skew-backs" were securely tied in. A sweep of 2" to one foot is a good figure to keep in one's head.

Arches, as everyone knows, are laid on wooden forms, which are burned out on drying furnace. The important part of an arch is the "skew-back." See Fig. 8. These should be of large blocks and slightly bigger on skew-face than thickness of arch. The skew face is flat to receive the first brick in perfect contact. *Skewbacks must be backed up on rear with first-class brickwork.* The final backing is an angle bar or a tee bar set in the brickwork and this should be held in place perfectly by the "iron work." It is the first rule of arch construction to "look after your 'skew-backs,'" and to see that they cannot be displaced any fraction of an inch.

Every surface of bricks in an arch must be in good mechanical fit with surface touching, since there can be no sliding along planes of contact. For many places where "span" is short, big blocks can be used to replace the arch. For a short span a single block supported at both ends on walls, suffices. By holding these down securely it is possible to span with two of these set "cantilever" fashion a full 30". See Fig. 9. In other places these can be cut in form of "jack arch" or "Dutch arch," to use another colloquial term.

An arch is necessarily laid tight and with "joints broken" as in all brickwork and with a smooth fit on all surfaces. See Fig. 10. The final key-bricks are driven home with a hammer. The blow should be received on a wood, such as a short end

of a 2" x 4" and never on the brick directly. Arches are laid with especial care when they have heat on both sides of them.

It can be remarked that any big block such as one for a "skew-back" or a "Dutch arch," had best be cut on a bed of sand. The bed of sand absorbs the jar of the cutting tools and stops the generation of fissures and possible ultimate cracking of the block.

As brick are laid wet, the furnace must be dried out. This drying should be slow with a gradual raise of the heat. Twenty-four hours drying for each $4\frac{1}{2}$ " or for each course of brick is a safe rule. Thus a furnace with 9" wall would take 48 hours to dry, and with an 18" wall, 96 hours. As the brick

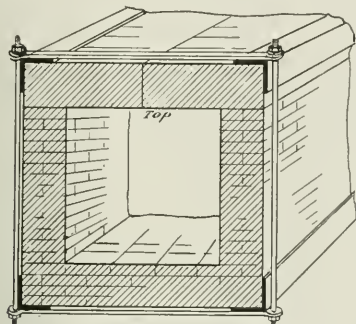


FIG. 9.—SHOWING WAY OF COVERING OVER A FLUE ON A SMALL FURNACE WITHOUT USE OF AN ARCH. THIS IS PRACTICAL TO SPANS AS LARGE AS 30" TO 36". THIS FORM OF CONSTRUCTION IS PARTICULARLY APPLICABLE WHERE A FLAT COVERING IS ADVANTAGEOUS

retain moisture and are decidedly hygroscopic a great heat applied suddenly would generate steam so fast as to cause trouble and a wood fire is used, particularly at first. Afterward a coke fire or an electrically heated coke resistance can be employed, but this is a little risky.

At the time the furnace is being heated up the tie rods should be adjusted, so as to meet properly the increasing strain.* Care is taken in bringing a furnace up to working temperature and a policy of conservatism is the only one to be employed. The injunction to "play it safe" is about as good advice as can be given.

Water Cooling

We have learned in the case of slag tap-holes, electrode ports, and electrode connectors, that the way to reduce the cost of repairs and renewals to a minimum was to use where we could, water-cooled metal instead of a refractory. For as long as the water was circulated freely the metal could not be destroyed and a permanent piece of apparatus was made.

Now although electric heat is expensive and heat insulation is desirable, provided it is attainable, it is not good business to try to save money in the power item and waste twice as much money in the repair and renewals item. For many purposes water-cooling of an electric furnace in a manner similar to that used where locomotive fire-box, and iron, copper or lead blast-furnace is water-cooled is a proper design.

A characteristic of electric furnaces is in many types an excessive local heating in places and at times. This produces bad results and if the electric furnace is to have any extended commercial application beyond its present restricted limits, troubles arising from the afore-mentioned causes must be eliminated or held under control.

There is, all things considered, no better way or at least no simpler way to attain permanency of furnace shape than by water-cooling. While we have in this connection the primal disadvantage of electrothermanism as against its competitors of high-cost of the heat unit, still we have at disposal the great primal advantage of convenience of use and application of

electrically generated heat, when this use and this application is understood. This will be seen in the following reasoning.

The thermal efficiency of electric furnaces is always high because of direct application of energy without use of considerable matter. In carbon-heated furnaces the waste gases are a source of loss of thermal energy even when recuperators and regenerators are employed. In electric furnaces, the heating agent is the ether, or to use the common-sense term of wise old Benjamin Franklin, the "electric fluid." This generates heat directly in an imponderable form and leaves all its energy in an easily available commercial form. Moreover the rate of smelting by electrical means can be made high. 11 ratio of calories (or B. T. U.s) to cubic foot of working space per hour is higher other things being equal, the proportion of heat lost in radiation, conduction, etc., is reduced. Consequently, the thermal efficiency is higher as "smelting rate" is increased and water-cooling entails only a small percentage of heat loss. Briefly expressed, a lot of work is done in a little space.

Just here is where the electric furnace has an unsuspected thermal advantage over its competitors, for the "smelting rate" can be increased to higher values than in case of "carbon-heated" furnaces. For instance, experimental furnaces of the Continuous Zinc Furnace Company have been operated 300 lbs. of ore charged per cubic foot of laboratory space per day whereas zinc furnaces fired by gas operate in certain cases at 5 lb. of ore charged per cubic foot of laboratory space per 24 hours. Such a state of affairs puts a new light on the competition of electricity and carbon for heating furnaces. And we find that water-cooling does not possess great terrors for the designs of electric furnaces, if at the time the true principles are understood, but rather that water-cooling instills a wise and moderate respect and liking for the great service it can effect in the way of reducing cost of repairs, and increasing length of campaign.

A peculiar fact is disclosed by a study of water-cooling. The thermal conductivity of all substances increases greatly with rise of temperature. Accordingly if we have an outside

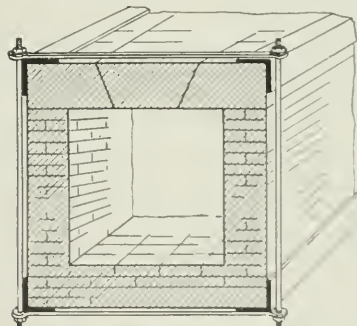


FIG. 10.—SHOWING MANNER OF LAYING A "DUTCH ARCH" SHAPED AND CHEAP FOR SPANS UP TO 30" TO 40" OR EVEN LARGER. THIS FORM OF CONSTRUCTION IS PARTICULARLY A GOOD WHERE FLAT COVER OF SOME SIZE IS DESIRED

temperature of 250 deg. C. in one case and outside temperature of 60 deg. C. in the case of water-cooled, we have a much less resistance to flow of heat in the second case. So, a thin water-cooled shell can equal as a heat-insulator a thick brick wall. It can be conceived that the "flow of water frees the walls of the furnace and so keeps heat inside."

Water-cooling can be applied to electric furnaces in any of the ways known in ordinary metallurgical practice.

These are as follows:

- (1) Cast-iron water jackets.
- (2) Pressed steel water-jackets with joints and stay-bolts welded by oxy-acetylene flame.

*Peters.

(3) Internal bronze-cooling plates same as used in back of iron-blast furnace, where a thick structural stony wall is desired.

(4) Spray-plates such as used on iron-blast furnaces. These are cheap and simple. The spray should be under a head not to exceed 10 feet and the plates should be smooth. Spray-plates are "sloppy" even when well designed, but they are the most efficient and most practical of all devices in all other respects.

In the three first-mentioned cases, care should be paid to see that no steam pockets are formed or that no sediment is deposited. By having *water enter at bottom and come out at top* and by having baffle plates to secure good circulation these troubles are obviated. In all cases the cooling metal surface is held mechanically firm to lining.

In considering case of water-cooling versus refractory walls it is well to hold in mind the nature of work to be done. If a slow "soaking" heat is the kind that work requires, design your furnace with no water-cooling. If a rapid transfer of heat is required, design your furnace with adequate cooling devices.

Or if one part of the furnace needs a "soaking" heat and the other a "rushing" heat, combine two forms of design in a composite manner.

The difference in extreme cases is deep-seated and a rational regard for this fundamental is an essential in making a good design of a commercial electric furnace.

Heat Insulation of Furnaces

We have just seen that such furnaces as are operated with a slow transfer of heat from its source to the point of absorption or of the "soaking-heat type" are not suited for water-cooling. These furnaces are usually resistance furnaces of the Cowles type. For instance, in the several forms of annealing furnaces now on the market it would be unwise to attempt to water-cool, especially as they are built in small sizes and thermal efficiency is diminished by such a procedure. In the case of small units water-cooling proves to be an undesirable complication.

In these cases the resistors are some form of carbon or of inert metal wire. The sides can be of some form of carbon with less electrical conductivity as charcoal. This forms an effective heat insulator and one resistant to fusion. But of course, if any air should touch it, total oxidation would quickly follow. Carborundum or silicon are other infusible substances that are used for wall next to resistor.

The study of the heat gradient will allow a use of several refractories with different properties as the heat diminishes in intensity. Magnesite brick can be used up to a temperature of 1700° C. or a pure electrically sintered magnesite, such as is made by the Norton Co., up to 2000° C. Likewise, brick made of carborundum base such as were lately put on the market by the Carborundum Co., and "alundum" brick as offered by the Norton Co., have special uses. Carbon brick are perfect as long as oxygen with any free energy is absent. Aluminum Nitride also has possibilities for refractory linings.

All these special and novel forms of refractories have their uses but can only be used where practical trial shows them to meet the needs. Some of them are too expensive for ordinary purposes. But where duty is hard, expensive brick makes for cheaper final result. It is merely a platitude to say that the choice of brick is a balancing of first cost against cost of repairs in each individual case.

For the outer layer fire-brick, red-brick, or reinforced concrete will suffice. In building a furnace of the "soaking heat" type, effective heat insulation can be gained by the use of air spaces between different courses of brick, or by separating the courses as "stretcher" courses with no "header" courses.

Mineral wool or "kiesel guhr" on the outside either plastered on or packed in a shell of asbestos board or sheet steel will keep in the heat as long as the temperature is below 700° C. Asbestos or magnesite packing such as is furnished by the Johns-Manville Co. is another convenient form.

All these varied forms of refractories have varied uses and

applications depending on the nature of the work and on the special scientific, practical, and commercial conditions that are to be fulfilled. The commercial conditions are the weighty ones, for no matter how scientifically perfect an electric furnace is and no matter how well the engineering and practical part is, unless an electric furnace is a commercial success, it is a miscarriage. Applied science is a business and the engineer must understand the principles of business and economics to a greater extent than was once thought requisite.*

Specific Volume and Constitution of Alloys

A paper on this subject was presented by Dr. W. M. Guertler at the recent meeting of the (British) Institute of Metals in Ghent, Belgium.

Dr. Guertler's paper is of a theoretical character and follows up a previous communication of his to the Institute of Metals in which he discussed the connection between the electric conductivity of alloys and their constitution.

Diagrams are given to demonstrate the dependence of the volume of alloys on concentration and temperature and explanations are given of the influence of mechanical treatment on the volume. The latter effect is generally two-fold where a normal alloy is hammered or rolled, (1) small hollows, pores, etc., closing up and the process ceasing when all these hollows have closed up; (2) where the individual crystal grains of the aggregate are broken up the grain becomes finer and this breaking up never ceases as long as mechanical work continues. The co-operation of these two factors causes the specific volume first to decrease, then to attain a minimum and again to increase as soon as the first factor is removed by the closing up of all the cavities. This explains Wertheim's discovery that the volume of metals and alloys can be increased by intense mechanical treatment. Figures are given to show the combination or the process of crystallization with that of transformation. These figures also represent and give a general idea of the "volume" surfaces of alloys.

Arsenical Copper

In a recent paper before the (British) Institute of Metals Mr. F. Johnson discussed a method of improving the quality of arsenical copper.

In this paper the author records the results of an investigation of the properties of specially alloyed arsenical copper with a view to proving its suitability for use as locomotive stay-rods, wire for rivet manufacture and any material required to have specially high malleability, ductility and toughness combined with tensile strength and resistance to reducing-gases at high temperatures.

An improvement in quality in these respects has been obtained by adding silicon and iron to the copper in various small proportions. When added to the copper in the form of a copper-silicon-iron alloy (of low melting-point) complete deoxidation is effected and, in addition, degasification is so thorough that perfectly sound ingots are obtained. In the absence of bismuth and other like impurities, which cause red-shortness in deoxidized copper, the ingots behave perfectly when rolled at a bright-red heat, even if silicon and iron are present together in considerable excess of the amount required for deoxidation.

It is shown that silicon is a more efficient deoxidizer than iron, but that iron has a greater strengthening effect than silicon.

Complete analyses of the rolled bars have been made and it is noteworthy that the added elements confer greater toughness than is possessed by "tough-pitch" copper containing oxygen and a similar quantity of arsenic. Heat-treatment of the experimental bars in a reducing atmosphere also proves their immense superiority over "tough-pitch" arsenical and non-arsenical bars.

The experiments have been backed up by works' tests on a larger scale.

* In the article of this serial, published in the October issue, the heading of the table on the bottom of page 563 should read: Safe carrying capacity in amperes (instead of amperes per square inch).

Tensile tests at the temperature of high-pressure steam show that the specially alloyed bars are in no way inferior to "tough-pitch" copper of similar arsenic contents.

The above-mentioned improvements have not been gained at the expense of any of the properties considered to be essential in commercial "tough" copper for locomotive stay-rods.

Certain physical characteristics of the experimental bars are revealed and differences in behavior explained by the aid of the microscope.

Notes on Chemistry and Metallurgy in Great Britain

(By our Special Correspondent.)

The British Association

The presidential address of Sir Oliver Lodge (published in full in the October issue of this journal), was generally very favorably received by the daily press, but among scientific men the tone of that section of his address which dealt with "continuity" in relation to "life after death" is by no means so generally approved. There are many men with more than thirty years of "accumulated experience" of so-called psychological matters who are still without any evidence, positive or negative, on the matter.

It was a pity that Sir Oliver did not give some indication of the nature of the evidence furnished by his experience, because at present there is really nothing in the shape of evidence one way or the other. We have only the disreputable tricks of spiritualistic mediums, which would not delude an average child of ten; and scientific men are not inclined to regard such impostures as evidence or even to notice them at all. In really scientific circles the number of men is steadily increasing who are content, like Huxley, "to follow Socrates in the belief that 'the knowledge of what we do not know is perhaps the surest; and to hold that those who do not attain that knowledge, who 'presume beyond human limitations, are rightly visited with 'the punishment of becoming the slaves of their own delusions, 'the worshippers of idols, which are their own works as much 'as if they were hand made.'" [Huxley, Nineteenth Century, "March, 1895.]

The true essence of the whole matter is to be found in the preface to the Pious Editor's Creed in James Russell Lowell's "Biglow Papers," and although discussion may be just as much out of place in these columns as it is in the meeting, to the British Association, the following brief extract from that preface may perhaps be of service in preventing anyone with psychological tendencies from searching too deeply after the unknowable:

"But the clergyman chooses to walk off to the extreme 'edge of the world, and to throw such seed as he has clear 'over into that darkness which he calls the next life. As if 'next did not mean nearest; and as if any life were nearer 'than that immediately present one which boils and eddies all 'round him at the caucus, the ratification meeting, and the 'polls! Who taught him to exhort men to prepare for eternity 'as for some future era of which the present forms no integral 'part? The arrow which Time is now turning runs through 'the everlasting, and in that must he plant or nowhere. Yet 'he would fain believe and teach that we are going to have 'more of eternity than we have now. This going of his is 'like that of the auctioneer on which gone follows before we 'have made up our minds to bid."

Transmutation of Metals

Mr. Frederick Soddy in his discourse on "Radio Active Elements and the Periodic Law," explained how it was now possible in consequence of recent discoveries to assign their proper places in the periodic table of the elements to the disintegration series of radio-active materials such as uranium, thorium and actinium.

He said that the chemical analysis of matter had hitherto appeared to be ultimate, because it had been impossible to dis-

tinguish between elements which were apparently chemically identical and not separable unless they were in process of change one into the other. But for those elements of which the evolution was still going on the places in the periodic table were found to be occupied on the average by no less than four, the atomic weights of which varied as much as eight units.

The same conditions were probably true for the rest of the table, and every known element might be composed of a group of non-separable elements, apparently occupying the same place, and the atomic weights would not be a real constant, but a mean value.

These advances showed that matter was much more complex than had hitherto been revealed by chemical analysis, and at the same time indicated that atomic constitution might be more simple than had been supposed from the lack of simple numerical relations between the atomic weights. These group-elements all gave the same spectra owing to similarity in their electronic system.

Madame Curie possessed about a gram of radium, and intended to compare the spectra of radium-lead and ordinary lead. He thought they would be exactly the same; but Madame Curie did not agree. He did not see any reasons why, under suitable conditions, it should not be possible to change thallium or mercury into gold. It only needed the expulsion of one alpha particle from thorium, which could be effected at a potential of 1,000,000 volts; but at present it was difficult to work with over 100,000 volts. Also, the expulsion of one beta particle and two alpha particles from lead ought to form gold, and the intermediate product would be bismuth.

Radio-Elements

Mr. Alexander Fleck, in the course of an interesting paper, said that since last year's meeting the study of the chemistry of the radio-elements had been continued, and the chemical nature of eleven additional radio-elements had been experimentally ascertained. It appeared that Uranium X and radio-actinium were chemically identical with thorium; meso-thorium —2 was chemically identical with actinium; radium-A was chemically identical with polonium; radium-C, thorium-C, actinium-C, and radium-E were chemically identical with bismuth; radium-B, thorium-B, and actinium-B were chemically identical with lead; and finally thorium-D and actinium-D were chemically identical with thallium.

The Effect of Exposure to the Atmosphere on Metallic Wires

Mr. Ernest Wilson presented a report on the results of tests designed to show the effect of exposure to London atmosphere on the electrical resistances of various metallic wires having a diameter of 0.126 in. and a length of 70 ft. These tests are a continuation of Mr. Wilson's earlier experiments of the same nature, the results of which have been communicated to the Association for some time past. The figures now given show the percentage increase in electrical resistance, at 15 deg. C., on the value found in 1911, and are for two years' exposure. 11gh conductivity copper showed an increase of 2.0 per cent; commercial aluminium, 4.4 per cent; and "Duralumin", 8.2 per cent. In the last named metal exposure has apparently induced some increase in brittleness.

Wrought Iron for Rolling Stock

The Engineering Standards Committee has issued a new specification embodying considerable alteration from the specification of August 1910. Among the more important changes are:

(1) A reduction in the number of tensile tests for best Yorkshire bar iron; (2) A similar reduction in the number of bend tests; and (3) the deletion of the quenching test; (4) the inclusion under grade "B" of the tests for plates which were included under grade "C", but with a slightly increased minimum elongation per cent, and somewhat more severe bend tests, and the insertion under grade "C" of plates of Cleveland iron quality; (5) an increase in the minimum contraction of area per cent. of original area for grade "A"

round and square bars up to $1\frac{1}{2}$ in. in diameter or section; (6) an increase in the minimum contraction of area per cent. of original area for grade "B" round and square bars up to $1\frac{1}{4}$ in. in diameter or section; (7) the optional use of standard test pieces B or F for the angles and flat bars up to 1 in. in thickness; (8) the provision of bend test pieces for blooms and billets, for bars over $2\frac{1}{2}$ in. in diameter or section, and for flats over 1 in. thick; (9) provisions for testing other blooms and billets; (10) the deflection from the tables of tensile tests of the requirements for round or square bars above 4 in. in diameter or section; and (11) the addition of a quenching test for grade "B".

The Anti-Corrosive Properties of Coal Tar

Experiments conducted by the Military Engineering Department at Brest show that the value of dry tar as a preservative of ironwork from corrosion is greatly increased by the addition of quick-lime which has the effect of rendering the coating more durable, especially in the neighborhood of sea-water. Two parts of coal tar were heated to boiling in a cast iron pan and were then triturated with one part of lime. At a temperature of about 50 deg. C., the mixture can be applied by a brush in a thin, uniform coating after the iron has previously received a coat of red lead, and drying is complete in about two days. A metal door so treated showed no sign of corrosion after 12 months, whereas ordinary paint did not withstand the action of the brackish atmosphere for more than six months. The total cost of red lead, tar compound and labor amounts to about 8d. per square yard.

Coal-Dust Explosions

The recently-published fourth report of the Explosions in Mines Committee gives the very interesting results of the experiments carried out by Dr. Wheeler in the Altofts gallery for the Mining Association of Great Britain.

The combustion of a mixture of coal-dust and air appears to proceed in two distinct stages; first, inflammation, and next explosive combustion; and in the earlier stage flame is propagated to a great extent by the combustion of gases resulting from the ignition of the dust, while in the explosive combustion stage the dust particles are entirely consumed—fixed carbon as well as volatile matter—without notable preceding liberation of gas. The times of travel and pressures at various distances recorded in experiments in which ignition was effected 500 feet from the open end of the unobstructed gallery pointed to the conclusion that combustion remained of the "inflammation" order for about 425 feet and then suddenly assumed the explosive character.

The violence of the explosive effect of a mixture of air and coal dust was found to be just as great with dust containing 16 per cent. of volatile matter as with dust giving 35 per cent. of volatile matter, and the effect produced with wood charcoal was fully as great as with coal dust. When the gallery was free from any obstruction the pressure produced at a distance of 450 feet from the point of ignition was only 16 lb. per sq. in., but when timbers, to represent roof-props in a mine, were introduced a pressure of 50 lb. per sq. in. was registered at a distance of only 225 feet.

Other experiments in which constrictions formed by 6-in. flanged angle iron, reduced the diameter of the gallery from 7 ft. 6 in. to 6 ft. 6 in. showed conclusively that the maximum pressure developed increased very largely, and, under certain conditions, was attained at much less distance from the point of ignition, while the time in which it was reached was very much reduced.

With three constrictions at 300, 350 and 400 feet from the point of ignition a maximum pressure of 152 lb. per sq. in. was developed at a distance of 450 feet as against 16 lb. per sq. in. in the clear tube. Three constrictions placed at 200, 250 and 300 feet from the ignition point resulted in a maximum pressure of 57 lb. per sq. in. at a distance of 250 feet in 66 hundredths of a second as against a pressure of 10.5 lb. at 450 feet in 1.03 second with one constriction at 200 feet.

The Co-Ordination of Engineering Research

At the summer conference of the Institution of Mechanical Engineers Mr. G. H. ROBERTS presented notes on engineering research and its co-ordination. He said that whilst engineering had reached a high stage of development as an applied science, it was remarkable that no definite system existed for communicating to the profession as a whole the results of numerous private researches which are carried on. He referred more particularly to that class of work which had a direct object in view and did not extend to side issues, and which in this country alone must be very extensive. Typical examples were furnished by the reports of the American Board of Ordnance; and there had also recently been an increasing tendency on the part of the engineering departments of our largest railway companies to interchange information.

The President, Sir H. F. Donaldson, in his presidential address delivered in April, had advocated the establishment of an engineering research committee on the lines of the Engineering Standards Committee, and the President of the Institution of Water Engineers had last June expressed the opinion that the results of very much experimental work were lost because they were not properly recorded; and when satisfactory records were kept the investigator retained his results. Railway companies and government establishments were probably least affected by considerations of competition, yet even in their cases certain investigations must be of a confidential nature.

At Woolwich Arsenal there were the ordnance factories and the inspecting and stores department of the army and navy; and the ordnance factories comprised three separate manufacturing establishments for the production of warlike stores. Each was especially concerned in its own class of manufactures; and it therefore followed that the laboratory researches were in some degree restricted to their own productions; but the great variety of material produced necessarily resulted in a certain amount of over-lapping, and a recognition of this fact led to arrangements being made some time ago for all experimental researches to be communicated to all departments. In addition, all the leading English and American engineering journals had been extracted, and the information so obtained was systematically recorded and indexed, and had constituted a valuable source of collateral information. With regard to methods of reporting results he did not advocate the use of stereotyped forms in every case, but he thought that for most of the tests now generally carried out standardized forms would be advantageous.

In the discussion Sir H. F. Donaldson said that as the Ordnance Factory at Woolwich had found that inter-departmental communication of research results was useful, it was to be hoped that the system would be followed in a wider field, and that private firms would assist in the co-ordination of research.

Professor Arnold said that at Sheffield the difficulties attending the co-ordination were being met by the establishment of a club consisting of three classes of members, namely, university graduates and associates, practical works, metallurgists, and the manufacturers.

Mr. Alexander Siemens spoke of the way in which the Siemens works in different countries co-operated in research work, and said that the plan which had been described was actually in operation between their establishments in London, Berlin, St. Petersburg and other places. But it was questionable if firms less intimately connected would take part in arrangements for making the results of their research generally available. There were firms who would not be willing to publish the results of their research, whilst they would gladly take advantage of the published results of what others had done.

Your correspondent is not alone in the opinion that, however desirable the co-ordination of research may be, it is not likely to be brought into operation in the immediate future, at all events, in England as far as manufacturing firms are

concerned, although something is to be hoped for in the direction of the collaboration of technical institutions and university laboratories for systematized research. The analytical and scientific chemist almost invariably publishes the results of his research, but this is not the case with the manufacturing chemist, who, as a rule, jealously guards against any divulgence of his processes, which lie, with some justification, regards as private property; and it is not to be expected that manufacturing engineer will be more ready to publish the results of his research than the manufacturing chemist, unless and until it can be demonstrated to him that he as well as others would benefit. The results of research undertaken for a definite object are just those which a private manufacturer or firm would be most unlikely to give away. There may be here and there an instance of the class last alluded to by Mr. Siemens; but vastly preponderating numbers of English manufacturers prefer to rely on their own experience and their own investigations.

The Supply of Tungsten in England

In *The Times* of the 13th August an anonymous correspondent stated that all the wolframite and scheelite produced in Cornwall or imported to Great Britain from the Colonies and abroad is sent to the Continent to be smelted into tungsten, of which a proportion is returned for the manufacture of electric lamps and a very large amount for the manufacture of tungsten steel. He further stated that there is no factory for producing the metal in this country, and pointed out that in the event of a European war in which England is involved the supply of tungsten would be cut off whilst the demand would largely increase.

But a week later *The Times* published a communication from a Luton firm stating that their works are now turning out many tons of ferro-tungsten a week in England, and are not only sending large supplies of that product to Sheffield but also to Russia, Austria, Germany and elsewhere.

Engineering Imports and Exports

The returns issued by the Board of Trade for the eight months ended on August 31 present figures which compare very satisfactorily with those for the corresponding period of 1912; but the statistics for the month of August exhibit considerable decreases in the value of most classes of engineering materials and products.

For the eight months the imports of iron and steel, including manufacturers, totalled £10,000,475, an increase of £1,038,248, and exports touched £36,841,732, an increase of £6,944,838. Other metals, including manufactures, were imported to the value of £21,571,818, an increase of £1,678,011, and were exported to the value of £8,949,586, an increase of £1,451,170. In electrical goods the imports amounted to £688,025, a rise of £67,118; and exports reached £3,477,685, an improvement of £764,671. Imports of machinery with a total of £4,022,334 show an increase of £415,517; and exports went up to £24,422,003, an advance of £3,260,284. The value of imports of new ships was only £24,223, a drop of £684; but exports rose to £8,318,951, an increase of £4,666,702.

The month of August, however, taken alone, shows extensive decreases. Imports of iron and steel were £9,018 less, and exports fell £51,583; imports of other metals decreased £254,215, and exports were £28,011 lower; electrical goods increased £16,781 in imports, but exports went down by £67,477; imports of machinery rose £3,021 and exports improved by £151,746; imports of new ships declined £2,354, while the exports increased by £2,466,506.

Market Report, September, 1913

	£.	s.	d.
Aluminum ingots in ton lots, per ton	96.	0.	0
Alum, lump, ton	5.	10.	0
Antimony, ton	30.	0.	0
Borax, Brit. refined crystal, cwt.	18.	6	
Copper Ore, to 25% unit	12 10 ¹ / ₂	13.	4 ¹ / ₂
Copper sulphate, ton	24.	0.	0

Caustic Soda, 70% ton	1	0	
Ebonite Rod, lb	4.	6	
Hydrochloric Acid, cwt.	5	0	
India Rubber, Para fine, lb.	3	7	
Mica, in orig. cases, medium	3 6 10	0	6
Naphtha, Scotch, gal.			10
Petroleum, Russian spot, gal.			7 4
Quicksilver, bottle	7	5	0
Sal Ammoniac, lump 1st, del. U. K., ton	44.	0	0
Tin Ore, 70%, ton	£123 to 125	0.	0
Sulphuric Acid, cwt.		5.	6
Lead, peroxide, ton	27.	10.	0
Potassium Bichromate in casks, lb.		3.	
Zinc, sheet, Vieille Montagne	26.	10.	0
Shellac, cwt.	4.	18.	0
Sulphur, recovered, ton	5.	10.	0
Platinum, oz.	9.	5.	0

Tin opened £193 and rose smartly to £197 on the 4th, then dropping away to £193, recovering to £194.15.0 on the 16th then again falling away, till £190.5.8 was reached on the 22d. During the last week it rose to £195.5.0, but demand being satisfied, subsequently fell touching £191, and closing £189.5.8.

Copper opened at £71 and rose, exhibiting considerable strength, till by the 16th it had reached £75. It then fell to £72³/₄ by the 22d, rose sharply to £74 on the 23d, and after another dip to £72³/₈, rose to £75.10.0 by the 24th, but ultimately gave way to depression caused by labor trouble, and closes £72.10.6.

Lead opened at £20.15.0 and was inclined to go higher during the first week, but afterwards settled down, showing a maximum of 20³/₄ and a minimum of 19³/₄ (rumors of unexpected stocks over the greater part of the month. It closes £20.10.0.

Iron, Haematite opened 67¹/₂, but rose 67²/₃ by the 8th, then flat, afterwards irregular, till it reached 67²/₃ on the 16th and closes 67²/₃.

Scotch Pig opened 62¹/₂ and fell away, being 60¹/₂ by the 15th, then recovering slightly reached 60.0 by the 19th, and closes same price.

Cleveland opened 56¹/₂ but declined after the 3d, being 54¹/₂ on the 10th and 54¹/₃ on the 15th. It had recovered by the 19th to 54.0 and closes at that price.

DIFFERENCES

Higher	£ s. d.	Lower	£ s. d.
Copper Sulphate	2.10.0	India rubber, Para, lb.	2
Copper Ore, unit	2.0	Shellac, cwt.	2.0
Sal Ammoniac, ton	2.0.0	Tin	3 15.0
Tin ore	8.0.0	Scotch Pig	1.3
Zinc, V. M.	1.5.0	Cleveland	1.3
Copper	1.10.0		
Haematite	3		

Market Prices, August, 1913

	£ s. d.
Aluminium, 98-99%, per ton	87 0.0
Alum, lump loose, per ton	5 10.0
Antimony, Star Regulus	30.0.0
Borax, Brit. refined crystal, ton	17 10.0
Copper sulphate, ton	21.10.0
Copper ore, 10-25% unit	10 10 ¹ / ₂ to 11.4 ¹ / ₂
Ebonite rod, per lb.	4.0
Hydrochloric acid, cwt.	5.0
India rubber, Para fine, lb.	3.0 ¹ / ₂
Mica, in orig. cases, medium	3 6 to 6.0
Petroleum, Russian spot, gal.	7 4
Quicksilver (Spanish), bottle	7 5.0
Sal ammoniac, lump, firsts, ton	42.0.0
Sulphate of ammonia, f.o.b. Liverpool, ton	13.5.0
Sulphur, recovered, ton	5 10.0
Shellac, cwt.	5.0.0
Platinum, oz., normal	9.5.0
Tin ore, ton	£115 to £117
Zinc, Vieille Montagne, f.o.b. Antwerp	25.5.0

better; imports of electrical goods increased £26,029, and exports advanced £215,670; machinery showed increases of £80,133 in imports and £373,181 in exports; new ships rose by £1,627 in imports, and exports improved to the extent of £732,965.

Tin has been fairly steady. Opening £186.0.0 it rose till the 12th when it touched £190.10.0, and then fell to £188, remaining for a week about this level till the 26th, when it again rose and closes £194.5.0.

Copper has not maintained the promise of a sharp rise which appeared to be probable in the middle of last month. Opening at £67.2.6 it reached £69.5.0 on the 9th, but fell to £68.15.0 by the 21st, but recovered and closes £71.0.0.

Hematite fell away considerably during the month. Opening at 71.9 it had dropped 2/- by the 6th, and another 2/- by the 21st. It closes at 67/-.

Scotch Pig opened 64/- and closes 62/-.

Cleveland opened 54/10 and did not show much activity till the end of the month. On the 24th it was 54/4, on the 28th 55/1½ and closes at 56/-.

Lead, opening high, £21.5.0, dropped to £19.3.4 on the 18th, and was £20 on the 22d, closes £20.15.0.

DIFFERENCES

Higher. £ s. d.		Lower. £ s. d.
India rubber	1 Aluminium	1. 0. 0
Sulphate of ammonia. 8. 9	Alum	10. 0
Sulphur recovered.... 5. 0	Hematite	4. 9
Shellac	7. 0 Scotch Pig	2. 0
Zinc	10. 0 Lead	10. 0
Tin	12. 0 0	
Copper	3. 17. 0	
Cleveland.....	1. 1½	

Synopsis of Recent Chemical and Metallurgical Literature

Electric Furnaces

Comparison Between Electric and Fuel Furnaces.—Anticipating the wider adoption of the electric furnace in metallurgical work within the next few years, Mr. F. LOUVRIER calls attention to the points of advantage in the electric furnace, in an article in the *Mexican Mining Journal* for September, 1913. Citing the fact that the electric furnace has "completely conquered the metallurgy of aluminium," and is used in the manufacture of steel, steel alloys and pig-iron, Mr. Louvriér makes the forecast that "at no distant date electricity will be applied in large scale to the treatment of all ores." The cause of this progress is ascribed to the superiority of the electric furnace over those now employed, the heat being produced electrically instead of by the combustion of fuel.

In considering fuel-burning furnaces, the author divides them into two classes: those in which the ore is in direct contact with the fuel, and others in which the ore is more or less completely separated from the fuel or products of combustion. It is evident that the thermic efficiency of the former type is greatly superior to that of the latter. By calculating the heat available from the combustion of carbon under operating conditions, and comparing it with that obtained in an electric furnace of the resistance type, the author reaches the conclusion that: "One electric horsepower per year is equal to 2700 kg of coke consumed in the ordinary furnaces." Comparing further the costs of the two thermic agents under different conditions, he arrives at the conclusion that the employment of electric furnaces is from two to five times as cheap as coke furnaces.

To this economy he adds the following advantages of the electric furnace:

- 1—Facility of regulating heat by simple handling of the electrodes or interrupters.
- 2—Constant production of the exact heat necessary, no matter what its intensity.

- 3—Elimination of the chilling of slags, so prejudicial in coke furnaces.

- 4—Easy treatment of all classes of ores, no matter how refractory.

- 5—Higher percentage of the recovery of metals as a consequence of (a) the non-oxidizing action of electric heat, which is produced without air or oxygen; and (b) the production of less and more fluid slag.

- 6—Decrease in manual labor due to less quantity of material to be handled, and because the ores can be treated as they come from the mines without crushing or briquetting.

- 7—Complete suppression of blowers and tuyeres.

- 8—Decrease in expenses of maintenance and repairs due to the maximum temperature being in the center of the ores treated and not on the walls of the furnace.

- 9—Production of purer metals; the electrical heat being chemically neutral and not introducing impurities contained in the usual fuels.

- 10—Possibility of operating profitably those mines situated in remote places, due to facility of transmitting electrical energy.

- 11—In all cases, decrease in transportation expenses, due to facility of treating ores at the mines.

- 12—Yields as high from small plants as from large ones.

Considering the above advantages as accruing in the case of metallurgical furnaces in which the ore and fuel are in contact, the author concludes that the economies of the electric furnace will be still further emphasized in the case of indirect firing. In the latter cases the following advantages of electric energy may be added to those enumerated above:

- 1—In the furnaces used for tin ores, one electric horse-power per year is equal to 3600 kg of coke, and yields a greater quantity of tin because the metal is neither oxidized nor carried away in the form of vapor.

- 2—In furnaces used for zinc reduction, one electric horse-power per year is equal to 7000 kg of coal, and the percentage obtained is not less than 95 per cent against 85 per cent by ordinary processes.

- 3—In the manufacture of crucible steel, one electric horse-power per year is equal to 18 tons of coke, this proportion being due to the high temperature required.

As the generation of one electric horsepower per year produced in gas engines calls for a consumption in coal of not more than 3855 kg, it can be concluded that in the treatment of tin ores, and, above all, of zinc ores, as well as in the manufacture of steel alloys and crucible steels, the use of electric furnaces is more advantageous, even should the electrical energy be produced by thermic engines consuming fuel.

Gold and Silver

Assay of Gold and Iridium in Black Sand.—Among the prizes offered in the Chemical, Metallurgical and Mining Society of South Africa, was one of twenty guineas by the Witwatersrand Co-operative Smelting Works, Ltd., for the best method of assaying gold in the presence of iridium and allied metals in such materials as black sand. The committee on awards selected two of the papers submitted as being superior to all others, although the committee was not fully satisfied that the authors had adduced sufficient evidence that the methods proposed would be wholly accurate and reliable. The following abstract of the two methods is taken from the July, 1913, *Journal of the Society*, in which the papers are published in full, with experimental data and bibliographies.

Method of James Gray.—From 0.5 to 2 A.T. of the material is assayed in the usual manner, and the button cupelled. The weight of this button gives the quantity of silver, gold and iridium. Owing to the tendency of the platinum metals to cause the button to scatter over the cupel, it may be necessary to add silver to the assay in order to obtain a single bead. In any event an excess of silver must be present prior to parting. The button is then flattened or rolled, and treated with hot concentrated sulphuric acid which dissolves the silver. The residue is washed and weighed, giving the weight of gold and

platinum metals. This residue is then dissolved in aqua regia, and the solution filtered to remove any insoluble platinum metals. The filtrate is made alkaline with caustic soda, the change from deep yellow color to almost white being easily noticed. From 10 to 20 cc. hydrogen peroxide (20 vols. quality) is then added, whereupon the gold comes down immediately as a black precipitate.



The mixture is then boiled for about ten minutes and filtered. After washing, it is run down in a crucible with borax, litharge and reducing agent, or scorified, and the resulting lead button thus obtained cupelled. The gold is weighed, and the difference in weight between it and the gold plus platinum metals gives the weight of the latter.

It is advisable to run a blank on pure gold simultaneously and add the loss thus observed to the weight of gold recovered. Ordinarily 99.5% of the gold is easily recovered. In the case of dilute solutions it is advisable to acidulate after the oxygen has been expelled by boiling.

The platinum metals may be recovered from the filtrate by means of sulphuretted hydrogen, and determined according to the methods of Leidie and Quennessen. The residue insoluble in aqua regia, as stated above, may be recovered in a convenient form by cupelling the filter paper.

Method of Chris Toombs.—One A.T. of the black sand is roasted at a scarcely perceptible dull red heat until sulphur-free. It is then fluxed as follows:

Black sand	1	A. T. (roasted)
Soda carbonate	2	A. T.
Borax	1½	A. T.
Litharge	1½	A. T.
Flour	4	gm.
Silver		q.s.

The mixture is placed in a paper bag and dropped into a red hot crucible. The assay is conducted at red heat until fusion is complete, and finished at nearly white heat. Wash with a mixture of litharge and flour, pour and cupel. Blanks of pure gold and silver should be cupelled at the same time. The resulting buttons are flattened, annealed, rolled and annealed, and parted in porcelain crucibles with dilute nitric acid (HNO_3 to $12\text{H}_2\text{O}$). Bring the acid to a boil, cool to about 80°C . and throw in the cornet. Action should be brisk but not violent. Heat to nearly boiling without actual ebullition; decant, wash and anneal carefully to avoid sticking iridium to crucible. Weigh gold and iridium together. Dissolve gold with 1 cc. 10% aqua regia in a long test tube; filter off the iridium, wash, dry and ignite over a bunsen burner. Weigh and make a deduction for the filter ash. Subtract this weight from that of the gold and iridium, obtaining the weight of fine gold in the 1 A. T. fused. Make proper correction as observed in the blank checks. If it is desired to recover and weigh the gold separately, the solution is neutralized with NaOH , added to a little NaCN solution, and precipitated with CuCl_2 . The precipitate is filtered, washed, scorified and cupelled.

Zinc Wafers for Precipitation of Cyanide Solution.—In our synopsis of current literature in April, 1913, page 219, we noted a new method of precipitation for gold solutions, making use of zinc wafers 1 to 2 in. long and $\frac{1}{4}$ to $\frac{1}{2}$ in. wide, instead of the usual zinc shavings. Excellent results were reported by Mr. John S. MacArthur. In the May, 1913, *Journal of the Chemical, Metallurgical and Mining Society of South Africa*, Mr. E. G. Baskett reports his experience with the use of this form of precipitation and gives decidedly unfavorable results, adding, however, that the conditions were not the same as those under which Mr. MacArthur obtained good results, and that more favorable conditions might give better results.

Mr. Baskett tried the method at the City and Suburban Mill, taking the special precaution to fill the lower compartments of the experimental box with zinc shavings in the usual manner. Four compartments were used for wafers and three for shavings, an intervening compartment being left empty and tapped at the side to obtain samples. With a flow of 10.5 tons solution per cu. ft. zinc per 24 hours, the extraction by

the wafers was but 35 per cent. On reducing the flow to 5.27 tons, the extraction was not improved, but fell to 31 per cent. White precipitate accumulated on the zinc wafers, and after 24 hours' running about 25 per cent. of the zinc surface was rendered inert. On cleaning up after ten days' working the wafers in the top compartment were found to have formed almost a solid mass with white precipitate. On examination of the wafers with a lens the whole surface was seen to be coated with a thin film of white precipitate which could not be removed by washing. The gold slime recovered contained only 1.41 per cent fine gold, the balance being white precipitate.

The experiment was repeated with stronger solutions coming from the sand vats, but was not more successful. After two days the whole precipitate appeared. The recovery was only 32 per cent.

Tin and Copper

The crude methods of smelting tin adopted by the natives of Mexico, in the vicinity of San Luis Potosi, are described in an article by Mr. GEORGE C. MASTER in the *Mining Magazine* for September, 1913. The work is done in a desultory manner by roving bands of tin miners who work in cooperation with the owners of ranches on whose lands tin deposits occur. The work is frequently profitable from the beginning of operations, as the miners work only exposed outcrops and quit when the vein pinches. The ore is crushed by hand to about $\frac{1}{4}$ -in. size, sorted and washed in a planilla and finally in a wooden batea. A planilla is a hole dug in the ground, about 5 ft. long, 3 ft. wide at the upper end and 2 ft. at the lower, with a sloping surface of from two to five degrees. At the lower narrow end, there is a cup-shaped hole in which the water collects. The depth is only a few inches below the surface, and stones are placed at the sides to prevent loss of water and mineral. The process of concentration consists in placing from 60 to 120 lb. of ore at the upper end of the planilla, and filling the role in the lower end with water. The latter is then thrown over the ore by means of a horn cup, and the lighter gangue is washed down the sloping surface, leaving the heavier cassiterite above. This process is repeated at least twice, after which the concentrate receives a final washing in a wooden batea, a form of concentrating pan. The concentrate thus obtained averages from 45 to 65% tin, according to conditions.

The blast furnace constructed by these natives is made of a hard stone cemented with clay called *tierra del fuego*. The dimensions of the furnace may be: height, 2 ft. 6 in.; top feed opening 5 in. square; front inside vertical, but back and sides sloping outward; inside diameter at bottom, 9 in. The bottom has a steep slope toward the open hearth in front, the latter being merely an excavation with a clay-lined floor. The blast is produced from two bellows, worked alternately by two men. The bellows are made of ox-hide; the nozzles unite, forming one thyrer made of a piece of iron pipe, placed about 8 in. above the furnace bottom. Before commencing operations the furnace and hearth are lined with clay, and when dry, a charcoal fire is started. A light blast is admitted, and when the temperature has risen sufficiently, the moistened charge is fed, in the proportion of one part concentrate to three parts charcoal, by weight. When the furnace is smelting well, some hard-lead obtained from previous smelting operations, is added. As much as 450 lb. concentrate may be smelted at one time, and when the operation is nearly finished, the slag is broken to 1 in. and resmelted. The treatment of 450 lb. concentrate requires 16 hours. The loss is approximately 10%. The tin is run onto the hearth every few minutes, and later rolled into parcels of 25 lb. weight. The metal contains from 90 to 96% tin. The cost for smelting 450 lb. concentrates is given at 10.20 pesos, or about \$8 gold.

Electric Smelting of Tin Ore.—Particulars concerning some experiments made by the Grondal-Kjellin Co., of London, in smelting tin ores in Cornwall, are contained in the *Revue Industrielle*. Pure ores yielded metal of 98% purity, and Bolivian ores containing about 50% tin and 15% iron, gave

metal of 92 to 97% purity. The latter could be further refined to a purity of 99.75% by blowing air through the molten mass. The energy consumed was 1700 kw-hours per ton; but this may be reduced to 1400 kw-hours, with an efficiency of 55%, by using two furnaces, one for the production of high grade metal and the other for treatment of rich slags. The process may be commercially profitable where hydroelectric power is available.

Copper Smelting at Messina, Northern Transvaal.—Reverberatory smelting of copper concentrates is in progress at the Messina copper mines in the northern Transvaal. According to the *South African Mining Journal*, the furnaces are small having a hearth area of only 23 ft. by 11½ ft. The capacity of each furnace is 20 tons of ore in 24 hours, and the fuel consumption is 7 to 8 tons of coal. Local coal has been found suitable for the purpose, and the limestone necessary for fluxing is also found in the neighborhood of the mines. The first matte produced contained 65% copper, but later operations were conducted to yield a matte of 50% copper. Additions are being made in the way of roasting furnaces, and it is expected that ultimately converters will be added.

Ore Dressing

Need of Better Classification.—In considering the improvements in ore treatment necessary to permit the utilization of lower grade ores, Prof. ROBERT H. RICHARDS, of Boston, Mass., believes that better and more extensive classification is one of the requisites. His paper on the subject appears in the September *Bulletin* of the American Institute of Mining Engineers. The following incident illustrates Prof. Richards' contention: A three-pocket cone classifier in a small mill had a feed ranging from 2 to 0 mm., and supplied four Wilfley tables. The first spigot had about all the sand and a great deal of slime; the other spigots and the overflow were about all slime. As a result all tables carried wide bands of slime and their tailings were charged with fine free mineral. The mill work was greatly handicapped by lack of good classification, for it is a proved fact that when the feed to a Wilfley table carries slime, then the tailing will carry fine free mineral. When this same ore was tested in the laboratory, it yielded easily 12 spigot products, all of which, when fed to Wilfley tables, gave no bands of slime on the tables; and free fine mineral was absent from the tailings of the coarser tables, and present only in small quantity in a few of the finer.

The benefit of this extended classification will be apparent from the following tables, showing the relative sizes of quartz and galena in the spigot products and overflows from a 3-spigot and a 12-spigot classifier.

Three-spigot Classifier

I
—of the quartz
3.5

Spigot	Quartz. Diameter, Millimeters	is galena. Diameter, Millimeters
1	6.30 to 2.15	6.30 to 0.614
2	2.15 to 0.73	0.614 to 0.208
3	0.73 to 0.25	0.208 to 0.0715
Overflow	0.25 to 0.00	0.0715 to 0.0000

Twelve-spigot Classifier

1	6.30 to 4.79	6.30 to 1.37
2	4.79 to 3.66	1.37 to 1.05
3	3.66 to 2.80	1.05 to 0.80
4	2.80 to 2.14	0.80 to 0.61
5	2.14 to 1.63	0.61 to 0.47
6	1.63 to 1.26	0.47 to 0.36
7	1.26 to 0.96	0.36 to 0.27
8	0.96 to 0.73	0.27 to 0.21
9	0.73 to 0.56	0.21 to 0.16
10	0.56 to 0.43	0.16 to 0.12
11	0.43 to 0.33	0.12 to 0.094
12	0.33 to 0.25	0.094 to 0.071
Overflow	0.25 to 0.00	0.071 to 0.000

Here we have limited and extended classifications of the same material, say quartz and galena. With free settling, the classified products, after the first spigot and before the final overflow, will contain grains where the quartz is approximately 3.5 times the galena in diameter. If we compare any two products, eliminating the first and last, both of which require special treatment, it will be clear that a Wilfley table treating spigot 2 of the three-spigot classifier, which has to separate 0.614 to 0.208 mm. galena from 2.15 to 0.73 mm. quartz, will not have so easy a task as the table which treats spigot 2 from the twelve-spigot classifier and has to separate 1.37 to 1.05 mm. galena from 4.79 to 3.66 mm. quartz. The same opinion holds with regard to the third spigot of the three-spigot classifier and the twelfth spigot of the twelve-spigot classifier; and is true also of all intermediate spigots.

"Next, let us see what a long classifier with many spigots will do, which a short classifier cannot do. We shall find, as follow on down from the coarser spigots toward the finer, that Wilfley tables will yield tailing products free from fine free mineral and, therefore, at the minimum assay that can be obtained without further regrinding. After a time, however, we shall arrive at a spigot where the heavy mineral in the spigot is the same size as the normal size of Wilfley fine free mineral. Then the Wilfley table breaks down, can no longer yield clean tailing, and the vanner should take its place, because the fine free mineral in the vanner tailing is smaller than that in the Wilfley table. If we follow still further down the spigots we shall find a spigot or the final overflow where the size of the heavy mineral is the same as the size of the fine free mineral in the vanner tailing. Then the vanner breaks down, can no longer yield clean tailing, and we go to the new great Anaconda round table, which takes all the rest down to the size of the heavy mineral that floats away in a slow-moving stream of water. With this the methods of water-gravity separation of fine material, that are now in sight, close."

The author goes on to show that the slime is not unduly diluted by using so many pockets in a classifier, provided steady and uniform feed is supplied. He concludes by showing the enormous sums that could be saved by good classification, and says "that we need better classification, more classification, and higher-priced, broader-minded men to see that the greatest benefit is derived from the improved machinery."

Recent Chemical and Metallurgical Patents

Iron and Steel

The preliminary preparation of fine ore and flue dust for smelting in the blast furnace, is the subject of a patent granted to Messrs. HARTLEY C. WOLLE and EDWARD F. KENNEY, of Westmont Borough, Pa. The invention consists essentially in compressing fine materials into a compact mass by passing a steady and uniform feed of these materials between rigid rolls, forming a thin sheet which subsequently breaks into flakes suitable for blast furnace work. The inventors' idea is to treat ore and flue dust that is finer than 1/16 or 1/8-in. mesh, and pass the same between rolls set 1/8 or 1/4 in. apart. The rolling pressure compacts the loose material, giving a product which has a greater surface density than exists in its interior portion, and which is at the same time porous enough for blast furnace smelting. Actual experiments with natural ores show that moisture must be present within the limits of three and eleven per cent. (1,073,381, Sept. 16, 1913.)

Preheating Air for Metallurgical Furnaces.—In Fig. 1 is illustrated the embodiment of an invention of Messrs. JOHN W. HAMILTON, JR., and JOHN W. JOHNSON, of Coatesville, Pa., designed to preheat air in a manner that will eliminate the use of costly checkerwork. The heat radiated at the hearth of the furnace is utilized to bring the air to the proper temperature, particularly for use in gas-burning furnaces. Referring to the figures, A is a furnace of the continuous type for heating ingots and blooms *x*. B is the heating chamber and C is the outlet to the stack. D is the inlet flue for gas,

and E is an air duct communicating with D by means of *e*. The ingots or blooms travel on water-cooled rails *b*, *b*, which are supported at intervals on piers *b*². The hearth *B*¹ is directly under the main combustion chamber of the furnace. Located under the hearth are two air chambers I, side by side,

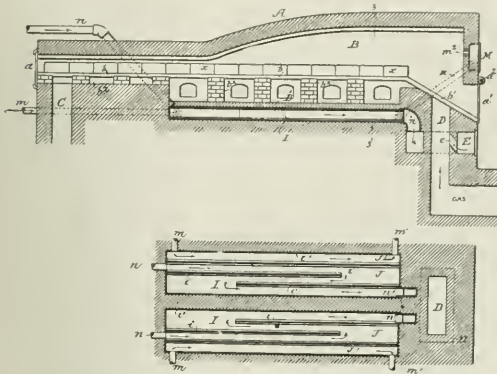
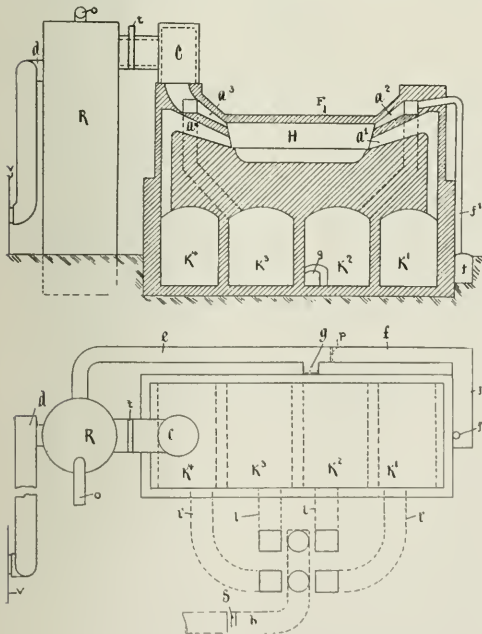


FIG. 1.—METHOD OF PREHEATING AIR

and there is only one layer of fire brick between these chambers and the combustion chamber. The plan of the air chambers is shown more clearly in the lower diagram of Fig. 1. Air is forced from a fan or blower into the passage *n*, traversing the chambers J and I, and issuing at *n*¹ to the air chamber E which communicates with the gas flue D, where it is mixed with gas prior to combustion. Auxiliary heated air

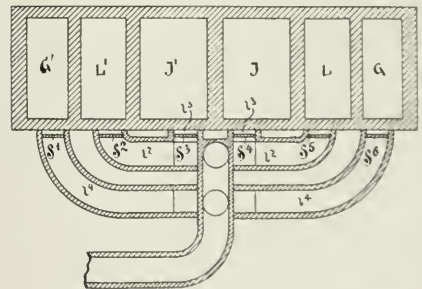
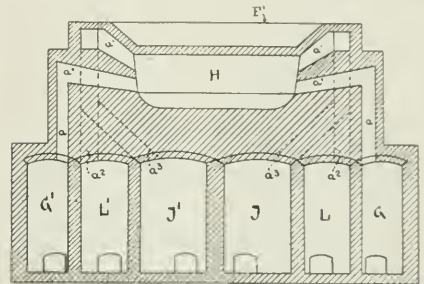


FIGS. 2 AND 3.—OPEN-HEARTH FURNACE CONSTRUCTION

passes through the straight side chambers J¹, entering at *m* and issuing at *m*¹ to the chamber M located above the outlet door of the furnace, and passing into the combustion chamber by passages *m*². The amount of gas and air can be controlled by any suitable valve mechanism. (1,072,331, Sept. 2, 1913.)

Air-cooled Blast Furnace.—RICHARD H. STEVENS of Mun-nace by a current of air. This air is furnished by the Low-hall, Pa., patents the idea of cooling the walls of a blast furn-ing engines or compressors, passes through the hollow metal jackets which form the shaft lining and proceeds in this pre-heated condition to the hot blast stoves, where it receives the high temperature required by the hot blast. (1,074,097, Sept. 23, 1913.)

Siemens-Martin Furnace.—When it is desired to recover and work up volatile metals or compounds contained in iron or in ferric oxide, KURT ALBERT of Wiesbaden, Germany, recommends to divide the open-hearth process into two periods, during the first of which the valuable volatile matters are recovered in an additional chamber arranged like the checker chambers. During the second period the steel is finished in the usual way and is safe from any admixture of undesirable elements, and furthermore the chambers will not become choked up. Thus the volatile metals or their oxides



FIGS. 4 AND 5.—OPEN-HEARTH CONSTRUCTION

from galvanized iron, tiniferous calcined pyrites and the like can be recovered. Figs. 2 and 3 show one arrangement of applying this process. The gases containing the volatile matter are drawn by a fan through the recuperator R and filtered in bags or other devices not shown. The chambers K₁ and K₂, also K₃ and K₄, serve for heating gas and air respectively. In Figs. 4 and 5 two additional air chambers, L and L¹, are shown through which the gases of the first period may pass, and in another arrangement shown in the patent specification the same purpose is aimed at by building two chambers, laterally outside of the regular furnace structure. The partitions of these auxiliary air chambers can be closed, each one individually, by a damper. This arrangement provides for a cleaning of the chambers while the furnace is working. (1,073,053, Sept. 23, 1913.)

Refining Iron and Steel by Means of Ferrochromium Alloy.—JOSEF BECHEL of Wengern-Ruhr, Germany, claims that the known objectionable formation of carbides when alloying commercial ferrochromium with steel will be avoided

when the ferrochromium contains neither carbides nor oxides. When free from these elements it has a much lower melting point than ferrochromium produced by the ordinary electro-metallurgical process [2]. To this end the chromium alloys are treated for protection from oxidation by alloying with silicon, aluminum, calcium, barium or an alloy of iron, chromium and aluminium. This helps an expeditious and uniform distribution of the chromium in the metal bath. (1,071,873, Sept. 2, 1913.)

Lead and Zinc.

Process for Sublimed White Lead.—In the manufacture of sublimed white lead from ores and concentrates, difficulties are encountered in subliming all of the metal and in producing a pigment of uniform white color. To overcome these obstacles Mr. ERNEST E. BANES, of Strathfield, near Sydney, N. S. W., has patented certain apparatus and method of operating the same, whereby he claims complete sublimation of the lead, and zinc if contained in the ore, and the production of a pigment of uniform white color and consumption. In carrying out the invention, finely divided lead sulphide ore or concentrate is fed continuously into a furnace through a blow-pipeflame introduced through a tuyere opening. The particles are thus dispersed, and complete volatilization occurs in the enveloping oxidizing atmosphere, matting and slagging being avoided. The fume passes from the furnace and is collected in a suitable bag house.

The material to be treated is ground preferably finer than 40-mesh, and introduced into the furnace in a blast of air and combustible gas, through two or more opposed blow pipes, which are directed toward a point a few inches above the level of the contained fuel bed. Hot air is blown through tuyeres toward the same point, and supplementary cold air enters the furnace tangentially through other tuyeres above the hot-air tuyeres. A plant for the manufacture of sublimed white lead according to this inventor's process would consist of a shaft furnace fitted with tuyeres as described and containing a bed of fuel; a gas producer; an air blower forcing air through a preheating chamber in the furnace flue; cooling chambers and bag house.

A sectional view of the furnace is given in Fig. 6, and the ore-feeding tuyere is shown in Fig. 7. Within the furnace a fuel bed 68 rests on a grate 13. Fire box and ash pit are accessible through close-fitting doors. Ore-feeding tuyeres are oppositely disposed, one being shown at 43. Hot-air tuyeres are placed at 45 and cold-air tuyeres at 47. The ore-feeding tuyere, shown in Fig. 7, consists of a rotating auger 58 which advances the ore fed at 62 to the nozzle where it is dispersed by a whirling jet of air entering at 53. The gas port is at 50. The rate of feed of galena is regulated by the speed at which the auger is rotated, and bears a fixed relation to the air pressure and gas volume delivered to the nozzle. This fixed relationship between supply of ore, gas and air is a matter

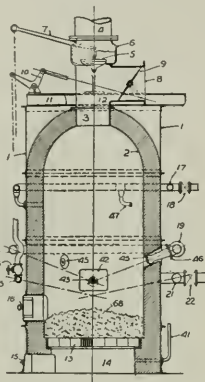


FIG. 6.—FURNACE FOR SUBLIMED WHITE LEAD

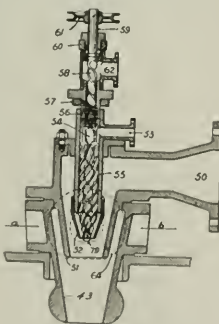


FIG. 7.—ORE-FEEDING TUYERE

of importance in the uniform operation of the furnace. The non-volatile elements contained in the ore fall as dust onto the fuel bed or are collected in the dust chambers. By grading the feed so that it contains a uniform percentage of galena, the product of the operation will be uniform. (1,073,462, Sept. 16, 1913.)

Lead Oxide for Storage Battery Plates.—An improved method of making amorphous lead oxide of low specific gravity and density, suitable for use in battery plates, is disclosed in a patent granted to Messrs. OLIVER W. BROWN and ALPHEUS R. NESS, of Bloomington, Ind. The product of their process is said to be more plastic and of higher tensile strength than similar pastes made from crystalline oxides. The required amorphous condition is obtained by heating basic or neutral lead carbonate, lead sponge, amorphous litharge or metallic lead at a temperature between 425 deg. C. and 440 deg. C. for a period of from three to three and one-half hours. (1,072,205, Sept. 2, 1913.)

Extraction of Zinc from Slags.—Processes for the extraction of zinc from furnace slags have been proposed, based on mixing the slag with a reducing medium, briquetting and re-treating in a furnace, whereby the zinc was volatilized and

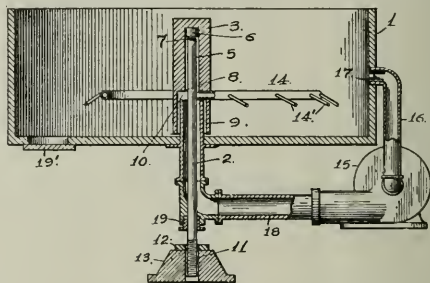


FIG. 8.—AGITATOR FOR ORE SLIME

collected as oxide. A novel process for treating furnace slags while in the molten condition, and removing the contained zinc oxide, is patented by ADOLPHE H. DESGRAZ, of Hanover, Germany. The operation is carried out in a reverberatory furnace, and depends on the principle that zinc oxide can be displaced from its combination with the ferro-silicates of the slag by adding a strong base, such as calcium oxide. The quantity of calcium oxide added to a molten slag is such as to give the resulting slag a lime content of from 25% to 35%. The zinc oxide being volatile, is collected in any suitable manner. (1,072,209, Sept. 2, 1913.)

Gold and Silver.

Agitating Apparatus.—A device for agitating, mixing and aerating mixtures of solid and liquid, such as ore slime

and cyanide solution, is shown in section and plan in Fig. 2, being the invention of Mr. WALTER E. TRENT, of Reno, Nev. It comprises a hollow convolute distributing arm, provided with a series of outlet nozzles projecting from the forward surface of the arm and extending in a direction opposite to the travel of the distributing arm. Its principal object is to overcome the objections to similar devices in which the distributing arms extend radially from the center. Among such objections is the difficulty of starting the agitating mechanism after it has been allowed to remain at rest for a period, due to the packing of solid material around the arms.

Referring to Fig. 8 (p. 656) the material to be treated is fed into the receiving tank in any suitable manner, until it overflows the pipe 16 and submerges the tubular arm 14. The pump 15 is then started and the slime is circulated through the pipe 18 into the distributing reservoir 10, and thence through the arm 14 and nozzles 14'. Motion will be imparted to the arm, which will rotate and direct streams of slime from the nozzles to every part of the tank. Owing to the arrangement of the arm and nozzles, the material agitated will not be banked in front of the arm following, as the nozzles project from the forward surface of the arm. (1,073,878, Sept. 23, 1913.)

Chloridizing-Roasting of Ores.—Apparatus for applying a chloridizing roast to ores for the purpose of rendering their base and precious metals soluble, is shown in Figs. 9 and 10, being the patented invention of Mr. NIELS C. CHRISTENSEN, of Salt Lake City, Utah. The object of the apparatus and process is to treat an ore moistened with a saturated solution of alkali and alkaline earth chlorides, by successively roasting,

the mechanism 10. Beneath the deck is an annular sealed space divided into two portions 11 and 12, by the wide partitions 13 and 14. The part 11 is beneath the cooling portion of the hearth, and the part 12 beneath the roasting portion. These spaces are suitably sealed from the outside air. Above the deck is a stationary hood 17-18, divided into two chambers 20 and 21 by the partition 22; the part 20 is above the cooling stage of the deck and the part 21 is above the roasting stage.

The ore is charged onto the slowly revolving, perforated hearth in a uniform layer from the hopper 25, and is removed by the screw conveyor 26. The space above the roasting deck is connected by pipe 42 with the condensing tower 41. The fan 28 is connected with the space 11 beneath the cooling portion of the deck; and the fan 31 is similarly connected with the space 20 above the cooling portion, and also by pipes 33, 34

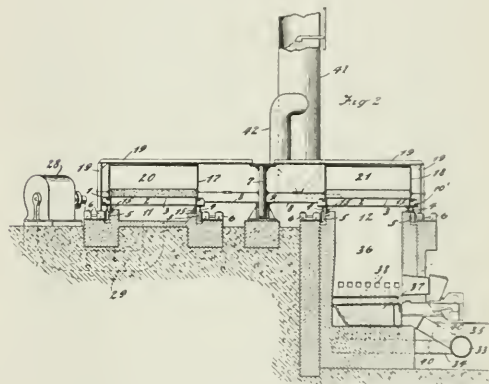


FIG. 10.—SECTION OF CHLORIDIZING ROASTING FURNACE

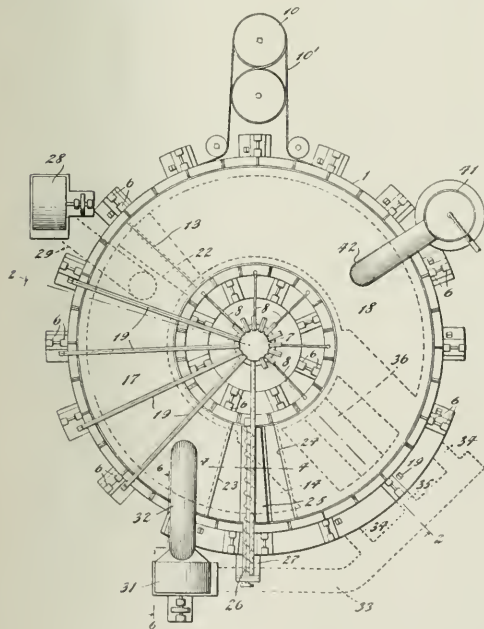


FIG. 9.—PLAN OF CHLORIDIZING ROASTING FURNACE

cooling and leaching. Features of the furnace are the means provided for conserving heat and for recovering the valuable constituents of the roaster gases.

Referring to Fig. 9, which is a plan of the furnace, and Fig. 10, which is a vertical section on the line 2-2 of Fig. 9, the numeral 1 indicates an annular deck with cast-iron frame 2, carrying a perforated grate 3. The deck and frame revolve, being driven by the rope 10' which encircles the deck and

and 35, with the combustion chamber 36 which opens into the space 12 beneath the roasting portion of the deck.

In operation, the ore is fed from the hopper 25, onto the revolving hearth, and passes successively through the roasting and cooling portions 12-21 and 11-20. The fan 28 forces cold air through ore in the cooling chamber, and extracts heat from the cooling ore. This heated air is drawn by fan 31 from above the cooling ore, and delivered by pipes 33, 34 and 35 through the combustion chamber 36 and thence beneath the roasting portion of the hearth and upward through the layer of ore.

From the space above the roasting ore the gases are drawn into the condensing tower. The metallurgical features of the process are treated more fully on page 605. (1,075,011, Oct. 7, 1913.)

Vacuum Filters.

Continuous Vacuum Filter for Thickening Pulp.—Among the various devices tried at Anaconda for thickening slime pulp prior to concentration was the Garred filter, the invention of ULYSSES A. GARRED, of Anaconda, Mont. The filter may be briefly described as a "fixed" type of leaf filter, constantly submerged in the slime pulp, the filtering being accomplished by the aid of a vacuum, and the cake thus formed being discharged by a reverse pressure of water. The novel feature lies in the arrangement whereby the operation of a series of batteries of such filters is automatically controlled by valves which prescribe the periods of time during which filtering and discharging are continued. The operation is thus made continuous; the filter frame is not removed from the tank; a large volume of water is withdrawn, and the thickened slime discharged from the leaves settles to the bottom of the tank and is withdrawn through suitable openings. (1,072,111, Sept. 2, 1913.)

Metallurgical Furnaces

Zinc Ore Roasting Furnace.—An improvement in mechanically rabbled roasting furnaces, particularly in the type in which sets of rabblers attached to an endless cable pass first through the furnace and then in the opposite direction outside the furnace, has been patented by MR. FRANCIS CURNOW, of Pittsburg, Kans. Since it is not necessary to operate such furnaces continuously, an automatic mechanism is provided for stopping the rabblers, by means of a tripping device which is actuated by the rable when it arrives at a predetermined point. An adjustable timing device is arranged to start the rabblers again after a predetermined interval of rest. (1,072,920, Sept. 9, 1913.)

An automatic charging device for roasting furnaces of the superimposed hearth type is shown in Fig. 11, being the patented invention of LUDWIG SINGER, of Bochum, Germany. In the crown 1 of the furnace 2 are arranged a number of

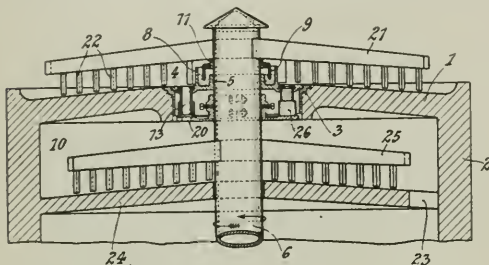


FIG. 11.—AUTOMATIC CHARGING DEVICE FOR ROASTING FURNACES

stationary, bottomless charging vessels 13 seated in a cover plate 4. These vessels are closed at their lower end by means of a false bottom secured to the central shaft and rotating therewith. The false bottom is provided with one or more openings 20 designed to register with the bottomless charging vessels during the revolution of the shaft. The volume of ore to be delivered to the furnace may be regulated by altering the

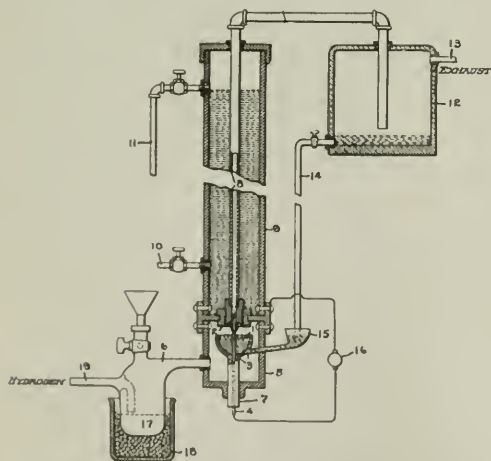


FIG. 12.—ELECTRIC ARC PROCESS FOR PRODUCTION OF BORON AND HYDROCHLORIC ACID

size of the charging vessels. In operation, the rake 21 delivers ore to the charging vessels which retain their charge until an opening in the false bottom allows the material to fall onto the upper hearth of the furnace. The arms of the rake 21 are related to the openings 20 in such a manner as to prevent the simultaneous filling and emptying of the charging vessels

A sand seal formed by the annular walls 8 and 9 prevents the escape of heat and gas from the upper hearth of the furnace. (1,071,962, Sept. 2, 1913.)

Electric Arc Reactions

Boron and Hydrochloric Acid.—JOSEPH L. R. HAYDEN of Schenectady, N. Y., assigns to the General Electric Co. a process and apparatus for producing chemical reactions in an electric arc. Fig. 12 indicates how boric chloride and hydrogen are caused to react in producing boron and hydrochloric acid. The arc is formed between a cathode of easily vaporizable material, such as mercury, zinc, cadmium or alloys, and an anode of perforated copper. The hydrogen for instance passes over the boric chloride (17) through the chamber 3, in which the arc is maintained, between mercury (1) and the copper (2). It carries enough of the liquid off for reaction. The products are cooled in the water-jacketed pipe (8). The excess of liquid returns through pipe (14) into the reaction chamber, the gas can be exhausted through pipe (13). The observations of the inventor indicate that the chemical activity is due, in part at least, to an electrical effect which is distinct from the purely thermal action of the arc. (1,072,945, Sept. 9, 1913.)

Acid-Proof Castings

Silicon Article.—THOMAS B. ALLEN of Niagara Falls, N. Y., assigns to the Carborundum Company an interesting process of pouring articles of metallic silicon. The silicon and its alloys made in the electric furnace are said to contain silicon dioxide, nitrogen and oxygen, all in dissolved form. This fact has made castings of the metal very spongy and produced blow holes, and mechanically unsound castings. A large number of substances are able to combine with these impurities and produce stronger, denser and more perfect castings. Among the substances are the alkali and alkaline earth metals, magnesium, vanadium, titanium, aluminum, boron, etc. The inventor prefers metals like calcium, magnesium or vanadium which deoxidize and also remove nitrogen. Silicon is tapped from the electric furnace into a graphite crucible and held molten for one or two hours in a coke or oil furnace. To prevent losses, the surface is covered with a layer of coke. When the silicon is in a very fluid condition, from 0.5 to 3% of magnesium are added. When the magnesium produces no further reaction the silicon is poured, preferably from the bottom of the crucible. The mold is made of dry sand, and its surface coated with talc. The metal shows a marked absence of blow holes and flow marks, and consists of very fine crystals; the articles have considerable mechanical strength. The castings have high acid resisting properties, and are particularly useful as chemical ware. (1,073,560, Sept. 16, 1913.)

Alloys

An aluminium alloy capable of being rolled into thin sheets or otherwise manipulated as a malleable metal, is patented by WILLIAM A. McADAMS, of Bay Shore, N. Y. It contains aluminum, zinc and copper in such proportions that the aluminum will be substantially five times by weight the amount of zinc, and the zinc substantially five times by weight the amount of copper. The aluminum is first melted, and into this molten metal the copper is added in a cold state, preferably in the form of wire. This mixture is then cooled to a dull red, after which the zinc is added in the form of slabs, and the mixture stirred and cast. (1,072,017, Sept. 2, 1913.)

The Alaska Gold Mines company has 1200 acres of ground on which 50,000,000 tons of \$1.50 ore have been developed. In the estimation of the company's engineers there is a total of 200,000,000 tons of such ore. The initial plant to be installed for the treatment of this ore will have a capacity of 6000 tons per day, and it is expected that a net profit of 75 cents per ton will be made. The ore will be mined through two tunnels driven on the 1600 and 2300 levels. The latter will be the main outlet and will be 10,000 ft. long. Eight thousand horsepower will be developed in two hydroelectric plants.

Regulating the Temperature of Liquids

In many chemical and industrial plants it is important to maintain water or any other liquid in a tank, boiler, etc., at a certain predetermined temperature. This is generally done by supplying through a valve, more or less steam (or hot water) to the tank or heating coil.

The problem of automatic temperature control is then simply to regulate the valve automatically in such a way that it admits just the right amount of steam which is required to maintain the temperature in the tank at the fixed value.

This problem is solved by the Sarco water temperature regulator in the following simple way:

As shown in Fig. 1, there are three principal parts: First, *A*, the thermostatic element which is inserted in the boiler or tank and which is thereby affected by the temperature of the liquid in the tank. The effect produced by any temperature variation in the thermostatic element *A* is transmitted from it

extends downwards into the tube *A* can be regulated by means of the screw head *C*.

From this thermostatic element *A* a piece of fine copper tubing *D* passes to the controller *G* which also contains a piece of corrugated copper tubing *F*. This corrugated copper tubing *F* is also capable of compression.

The thermostatic element *A*, the connecting tube *D*, and the controller *G* form one hermetically closed chamber

If now the temperature in the tank should rise, the temperature of *A* also rises, and the oil in *A* expands. The pressure exerted is transmitted from *A* through *D* to the controller *G* (all three forming together one chamber) and the corrugated compressible copper tube *F* in *G* is compressed. This forces out the piston *I* and tends to close the valve *K*. The spiral springs *E* and *J* operating in the opposite direction tend to keep the valve open.

Hence, as a result of an increase of temperature in the tank, the valve *K* will close partly until a new state of equilibrium is reached, so as to admit less steam to the tank and keep the temperature in the tank at the predetermined value.

Of course, should the temperature in the tank tend to fall, the regulator acts in the reverse manner, opening the valve *K* further, etc.

FIG. 2—EXTERIOR VIEW OF WATER-TEMPERATURE RECORDER

The regulator is capable of quite delicate adjustment by means of the screw *C*, which permits to expand or contract the compressible copper tube *B* and thereby decrease or increase the space in which the oil is confined.

An exterior view of the whole regulator is shown in Fig. 2. Some of the chief advantages claimed for the Sarco water temperature regulator are the compactness and simplicity of its design, the fact that it is wholly self-contained and requires no exterior operating means such as compressed air and the avoidance of rubber or leather diaphragms or of metal diaphragms. It is built by the Sarco Engineering Company, 110 Broad Street, New York City.

The Acimet Valve

The economical importance and absolute necessity of sulphuric acid and other corrosive liquids in many lines of manufacture, and the great annoyance and expense incurred in controlling and distributing these solutions by means

of pipe lines, valves, etc., have given great importance to the problem of devising pipe and valves that would resist corrosion.

The solution of the pipe question was comparatively easy, as lead pipe was available for the purpose, and later lead-lined pipe entered this field, but the valve proposition was far more difficult to solve.

The approach to the solution was somewhat crude, as is often the case, but the ultimate result was most gratifying, producing as it did the

Acimet valve, for which its makers claim not only high efficiency, but relative inexpensiveness.

Some of the very first Acimet valves made for experimental

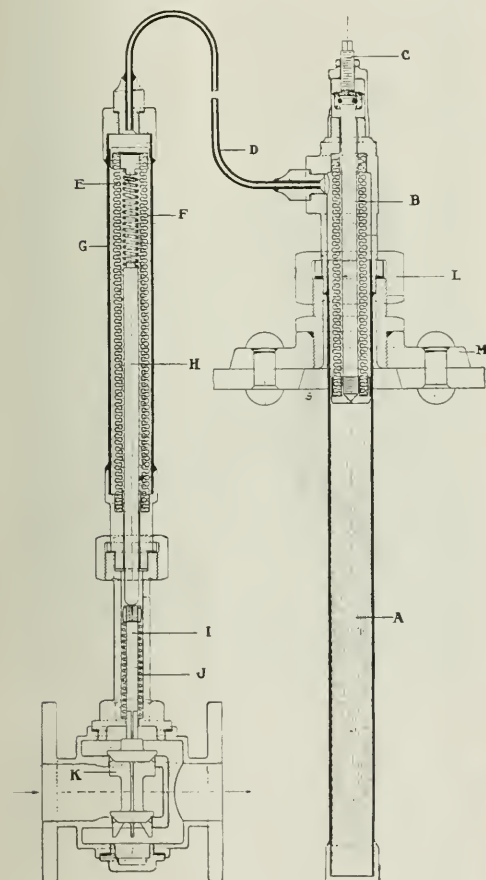
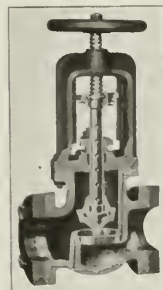


FIG. 1.—DIAGRAM OF WATER-TEMPERATURE REGULATOR

to the second principal part *G*, the controller element, and this acts directly on the third principal part, the valve *K*, which controls the admission of steam to the tank and thereby controls the temperature in the tank. Let us now see how this transmission of effects is brought about.

The thermostatic element *A* is a tubular receptacle containing a heavy hydro-carbon oil into which is inserted at the top a piece of corrugated copper tubing *B*. This corrugated copper tubing is compressible and its length or the depth to which it



VALVE CONSTRUCTION

purposes are still in service, having been used continuously for more than eight years, during which time they have required no repairs or attention except occasional packing.

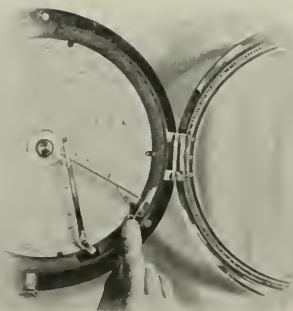
When more recently the question of placing the Acimet valve on the market came up for consideration, it was realized that a symmetrical and harmonious design would be desirable. After much experimenting the formula for Acimet metal was developed and a mixture produced which has all the wearing qualities of sheet lead, and which can be cast and machined the same as brass or iron, and through an interesting method of reinforcing (as shown in the illustration), it is made stiff and rigid, so that no distortion of any kind takes place under any strain due to either pressure or piping.

The Acimet valve, the novel and original features of which are covered by letters patent, is made by the Cleveland Brass Mfg. Company (formerly the Cleveland Bronze & Brass Works), of Cleveland, Ohio.

Automatic Release Pen Lifter for Recording Instruments

The Industrial Instrument Company, of Foxboro, Mass., which manufactures an extensive line of recording instruments, has just perfected a new attachment for their recorders, for which patents are pending.

This attachment is called an automatic release pen lifter and is shown in the adjoining illustration. It can be attached to any Foxboro recorder. It is a very simple device consisting of a German silver strip mounted on a special holder, which is inserted under a screw head which holds the chart disc. A slight pressure on a small lever brings the strip up against the pen arm and lifts the pen from the chart. Friction holds it in the raised position, thus giving the operator free use of both hands for removing the used chart and supplying a new one. When the door is closed, the pen arm is automatically released



AUTOMATIC RELEASE PEN LIFTER

and the pen returns to its marking position on the chart. This automatic feature makes it impossible for the operator to forget and leave the instrument out of commission.

The device is very neat, and not only helps in the changing of charts but, what is more important, does away entirely with the necessity of handling the pen arm, with possibility of affecting the adjustment due to a slip or accidental strain. It also prevents accidents commonly resulting in spreading ink where it does not belong and is not desired.

Seamless Lap-Welded Steel Kettle

The adjoining illustration shows a large seamless lap-welded boiling kettle made by the American Welding Company, of Cardonvale, Pa., for a prominent chemical company of this country. It is made of rolled steel plates bent into the desired shape and size. The inside diameter is 8 feet, the inside

depth 8 ft. 6 in., the width of the flange $4\frac{1}{2}$ in. The shell consists of $\frac{3}{4}$ -in. steel plate, the convex bottom is also $\frac{3}{4}$ in. thick. The capacity is 3000 gallons, the weight 9500 lb.

One of the great advantages of kettles made from rolled plate steel is that the construction is much lighter and stronger than that of kettles made from cast iron; as a result of the thinner walls the heat transmission is very much better. Further the plate-steel kettle can be thoroughly annealed so as to remove all inherent strains, while it is impossible to eliminate the strains in a cast-iron kettle.

The seams are heated by furnaces operated with water gas and are welded by either hammer impact or hydraulic power, forming a seamless weld about equivalent in strength and thickness to the plate itself, the surface of the weld being smooth on both sides. As the edges of the plates are lapped sufficiently to form a perfect weld no material other than



BOILING KETTLE

the edges of the plates is used in making the weld. Consequently a perfect lap-weld thus made is the same as the plate so that corrosion will not act upon the weld more rapidly than upon the body of the plate.

In the standard design of the American Welding Company the shell of the kettle at the top is flanged outward forming a right-angled flange. Supporting brackets of either cast iron, pressed or cast steel may be secured to the shell if preferred.

The kettle may be supported, however, by a specially designed ring base resting upon the furnace walls, the kettle being supported by its flange resting upon the ring. The advantage of such a method of support is obvious, as the shell of the kettle is kept intact and free from objectionable rivets. Replacements can be made at a minimum cost.

These kettles are made either plain or jacketed.

A metallographic study of the ores from the Silverton district, Colorado, is being made by Mr. B. F. Laney, under the joint direction of the Geological Survey and Bureau of Mines.

Multiple-Effect Water Still

The multiple-effect water stills, built by the Pure Water Apparatus Company, Abbott Building, Philadelphia, Pa., are interesting in various ways.

First, an essential difference from the ordinary multiple-effect evaporator may be pointed out which is indicated by the fact that the Pure Water Apparatus Company builds multiple-effect stills with up to 20 effects.

The apparatus consists of a series of similar cells through which the heat units of the primary steam pass in succession, in each cell evaporating water to be purified and producing distilled water by the condensation of the steam which has brought this heat from the previous cell, thus producing at each exchange an additional quantity of distillate by the re-use of the original heat.

This process in itself is not new. The new feature is that while in the process just described a certain amount of the heat is lost by radiation from the outside of each cell, this loss of heat by radiation of each cell is automatically compensated by the addition of just the correct amount of steam to each cell in the Pure Water Apparatus Company's system.

Let us assume an eight-effect still with an initial steam pressure of forty pounds. This steam enters the first condenser and is condensed, and the water surrounding the condenser is evaporated and raised in temperature until a pressure of say twenty-five pounds is reached. But since we began with a pressure of forty pounds and want to use eight effects, we want to get at this point a pressure of thirty-five pounds. In order to get this additional pressure, steam is now let into the header from the main by a reinforcing valve (a patented feature which automatically maintains the desired pressure in each of the cells).

This process is repeated from cell to cell until the final cell is reached and the pressure and heat in the steam are fully utilized, the heat finally emerging as steam at atmospheric pressure from the last cell.

Each condenser is provided with a vent cock by means of which, together with other devices, the poisonous gases and air in this steam are separated from the steam and allowed to escape.

A coke compartment is provided to purify the steam and also to prevent entrained water being carried to the next condenser in the series. The distillate from all the cells passes through a heat exchange where it meets the cool incoming water to be used for supplying the various cells, and conserving the heat for further use in the stills.

From the heat exchange the accumulated distillate is passed



FIG. 1—STILL FOR SOLSCHEID WATER CO., KANSAS, MO.

through a charcoal cell which deodorizes the water, making it sweet and palatable.

After the distillate is finally collected and passed through the charcoal cell it is totally devoid of all deleterious and obnoxious gases. The high temperature at which the distillate is in contact with the steam in these stills makes it impossible to reabsorb these gases so that with this system aeration to destroy them is unnecessary. Aeration is a makeshift at best, the old theory being that contact with the atmosphere would remove these gases; as a matter of fact, contact with the sur-

rounding air simply changes their nature, but they still remain as impurities.

Still another important feature is the pretreatment apparatus or water softener which removes all scale-forming ingredients from the incoming water before it enters the still. This pretreatment employs heat alone as the purifying agent and it is this system of softening water that assures continued economy under constant operation.

There are numbers of instances where these stills are operating after many years' service and no trouble has been en-

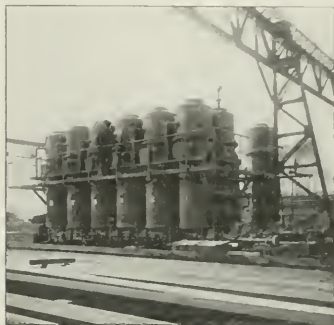


FIG. 2—INSTALLATION AT CURACOA, D. W. I.

countered due to scaling, using some of the worst alkali waters in the southwest.

This system has many other minor improvements which make for such economy of operation on a large scale that it is claimed to produce as many as fifty to eighty tons of chemically pure water per ton of coal—it being understood that the coal will evaporate in the boilers not less than eight pounds of water to one pound of coal.

Fig. 1 shows a small-size still which is at present in very successful operation for the Solscheid Water Co., Kansas, Mo. It is a 2500-gallon per 24-hour outfit of the two-effect type, giving about 1.7 lbs. of distillate per pound of live steam entering the initial cell.

Fig. 2 shows a large installation for the city of Curacoa, D. W. I., for their municipal water supply. It has a capacity 20,000 gallons per 24 hours, and is a 12-effect apparatus, producing more than 7 lbs. of distillate per pound of live steam entering the initial cell.

During a recent drought this machine not only supplied the needs of the city where it was installed, but was operated at 100 per cent. overload, supplying the additional drinking water for several nearby towns.

The Properties of Saturated and Superheated Ammonia Vapor

A very useful and interesting paper of 94 pages with the above title has been published as University of Illinois Bulletin, Vol. X, No. 18 (University of Illinois, Engineering Experiment Station, Urbana, Ill.), the authors being Professor G. A. Goodenough and Mr. Wm. Earl Mosher.

The vapor of anhydrous ammonia first became of interest in the field of mechanical engineering with the advent of Carré's absorption and Linde's compression refrigerating machines. With the development and increased application of the refrigeration industry this vapor has become more important and an accurate knowledge of its properties is highly desirable.

The investigation of Professor Goodenough and Mr. Mosher was undertaken with the object of collecting and correlating the various scattered experimental data on the subject of the properties of ammonia and an attempt made to reconcile these data by means of well-known thermodynamic laws and prin-

ciples so that the results may be consistent with each other; and to express the various properties by means of formulas from which tables and charts in English units could be prepared for use in solving the many problems connected with refrigeration work.

The chief results of the investigation are given in form of four tables on twenty-seven pages.

Of the four tables given, the first and second are for the liquid and the saturated vapor of ammonia and give explicitly all of the properties that are ordinarily needed for use. Table 3 is primarily for superheated ammonia vapor, but includes the saturated vapor as the special case of zero superheat. Table 4 gives the thermal properties of liquid ammonia.

In tables 1 and 4 the argument is the temperature. In tables 2 and 3 the argument is the pressure.

Then follows a chapter on the Mollier diagram for ammonia.

In order to facilitate the solution of refrigeration problems, a heat content-entropy diagram was prepared which accompanies this bulletin. This diagram has two families of curves: (a) curves of constant pressure, and (b) curves of constant quality in the saturated region and constant temperature in the region of superheat. The ordinates are heat contents; the abscissas are entropies.

The tables given in the paper are based upon the most reliable experimental data available at the present time. In the final chapter this body of data is presented together with the derivation therefrom of the formulas from which the tables were calculated.

Anyone interested in refrigeration problems should acquaint himself with this excellent and useful work.

Personal

Mr. Courtland F. Carrier, Jr., who has been chemical director of the Vulcan Detinning Co. for the past four years, has severed his connection with that company and removed to St. Louis, Mo., where he is taking up a general consulting chemical engineering practice with Mr. James C. Lawrence of Memphis, Tenn., under the name of Lawrence & Carrier, Inc.

The Dorr Cyanide Machinery Co. has opened a New York office at 50 Church Street, to take care of its expanding business in metallurgical work and industrial lines. The office will be in charge of Mr. H. N. Spicer, one of the company's engineers. Mr. Dorr also will spend a large part of his time in the East, devoting himself to the company's business and professional work.

Mr. W. H. Gibb, for five years Pittsburgh representative of the Brown Instrument Co., was made general manager of sales of the Stupakoff Laboratories of Pittsburgh, Pa.

Mr. W. Goward, emeritus professor of metallurgy in the Royal School of Mines, London, will publish a treatise on the metallurgy of the non-ferrous metals.

Mr. Simon Guggenheim has been elected chairman of the boards of the several Guggenheim corporations, American Smelting & Refining Company, American Smelters Securities Company, and Guggenheim Exploration Company.

Mr. Franklin Guiterman has returned to New York from one of his periodical visits to the Colorado offices of the American Smelting & Refining Company.

Messrs. Charles Hayden, D. C. Jackling and C. M. MacNeil have completed a tour of the western zinc and copper properties under their control, and have gone to Alaska to inspect their gold mining interests.

Mr. Julius Marcus, until recently chemist with the Ray Consolidated Copper Company, Arizona, has accepted a position with the Braden company in Chile.

Mr. Seabury C. Mastick announces that he has admitted Mr. Henry J. Lucke into partnership under the firm name of Mastick & Lucke, attorneys and counselors at law, 2 Rector street, New York City. Mr. Lucke is a graduate of Johns

Hopkins University, was for two years an instructor of chemistry, physics, and mathematics at Lafayette College and for seven years and a half was an assistant examiner in the U. S. Patent Office, serving in the division of electrochemistry and metallurgy and electricity, and for the past five years was practising law in New York City.

Mr. J. B. Shatzer, Pittsburgh representative of the Schutte & Koerting Co., will give a "Safety First" lecture on automatic engine stops and triple service valves before the National Association of Stationary Engineers in Pittsburgh on November 17.

Mr. F. W. Skirrow, chemical engineer, has accepted a position on the teaching staff of the University of Colorado, at Boulder, and will give lectures in industrial chemistry, at the same time remaining open to engagement in a consulting capacity.

Mr. E. J. Wagor has been superintendent of the melting and refining departments of the United States mint at San Francisco.

Mr. F. L. Zimmerman has been appointed Eastern representative of the Hoskins Manufacturing Company, of Detroit, makers of electric furnaces, pyrometers and heating appliances, to succeed Mr. E. L. Smalley. His headquarters are at 30 Church Street, New York City.

Notes

The Colorado Chapter of the A. I. M. E. has been organized with the following officers: Wm. J. Cox, chairman; C. J. Moore, vice-chairman; C. L. Colburn, secretary-treasurer; T. B. Stearns and Richard A. Parker, members of the executive committee with the officers. The chapter has jurisdiction in the state of Colorado.

The Abbe Engineering Company, of 230 Broadway, New York, announces that they have purchased the Sellman Mill Company and the pulverizing and grinding machinery business of G. M. Ball & Son, together with all the patents, drawings, patterns, good will, etc., of the foregoing concerns so that they are in a position to supply new mills, also repairs and supplies for the machines bought of these concerns. Mr. Henry Sellman, who has been building pebble mills and installing complete grinding plants for the past thirty-four years, is now with the Abbe Engineering Company in their engineering department. The Abbe Engineering Company state that they now control over thirty patents on various kinds of pulverizing and grinding machines.

High-Voltage Testing Transformers and Complete Testing sets is the title of Bulletin 570 of the American Transformer Company, of Newark, N. J. Two standard equipments are described, one for 50,000 and the other for 5,000 volts.

Electroplating with Trisalyt, the useful little pamphlet of the Roessler & Hasslacher Chemical Company, 100 William St., New York City, on the use of trisalyt (a metallic triple salt for electroplating copper, brass, bronze, zinc, gold and silver) has now appeared in the fourth edition. An interesting new edition is a formula for salt water gilding.

Irrigation is the title of a very neat and beautifully illustrated pamphlet of the Henry R. Worthington Hydraulic Works, 115 Broadway, New York City, on the conservation and distribution of water for irrigation, with many technical details of pumping machinery and tables of useful information on irrigation work.

The Log-Book of the Tower Plant is the title of an illustrated circular of the Yarnall-Waring Co., Chestnut Hill, Philadelphia, Pa., on the construction and uses of the Lea V-notch recording liquid meter. This liquid flow meter was described in our October issue, 1911, (Vol. IX, pp. 557), and also was the subject of an elaborate paper by Mr. D. Robert Yarnall before the annual meeting 1912 of the American Society of Mechanical Engineers. This paper has also been reprinted by the Yarnall-Waring Co. in pamphlet form.

Interesting Photographs from the Field is the title of a little and very neat booklet published by the Hardinge Conical

Mill Company, 50 Church Street, New York City, showing rather interesting photographs of some large mill operations in this country, as well as some typical Hardinge mill installations.

Pure Iron versus Copper-Bearing Steel is the title of Bulletin 5 of the Institute of Industrial Research, of Washington, D. C. The sub-title of this pamphlet of 47 pages describes it as "a brief for the plaintiff with historical data bearing upon the development of rust-resisting metals in the United States." The author is Dr. Allerton S. Cushman. In conclusion "the writer can only express the hope that those engineers and others who are accustomed to purchase material under carefully drawn specifications will not be influenced by a movement which proposes to overcome the bad effects produced by the usual impurities in steel by the simple expedient of adding another one (copper). The modern tendency throughout the whole great field of industrial development is towards a higher purity of product. The development of a pure iron as a commercial possibility is in line with this general progressive movement. It would be regrettable if this evolution in the iron industry should be retarded or halted by this retrogressive propaganda in favor of a special type of extra impure steel. It should always be remembered that the best is rarely the cheapest or the easiest to produce in any line of industry."

Tungsten Ore in Burma.—According to a consular report, wolframite has been discovered in the Tavoy and Mergui districts and a large number of concessions are being worked. The greater part of the output comes from float deposits and the ore is concentrated by the natives with pan, sluice-box and rocker. The average concentrates contain about 65 per cent WO₃. The cost of recovery and transportation to England is about \$315 per ton; but with ore selling at \$7 to \$8 per unit a good profit remains.

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

REFINING METALS (Continued).

533,596, dated Feb. 5, 1895, Henry A. House, of Bridgeport, Conn., assignor of one-half to Robert Rintoul Symon, of London, England.

Relates to apparatus for refining metals, particularly silver, by electrolysis. In a suitable tank or vat is a horizontal shaft, with means for rotating the same, and on said shaft I mount one or more cathodes and one or more anodes of the impure metal to be refined. The cathodes upon which the metal is deposited are in the form of segments of a disk carried by an insulating support and adapted to be alternately connected with the circuit while in the solution and cut out of circuit during that part of their movement when they are out of solution. The anodes of impure metal are also carried by suitable supports, and may be cast in the form of annular disks or in segments that are secured to such support and similarly arranged to be brought into and cut out of the circuit. Under each rotating electrode is a removable canvas bag to catch the impurities or particles of metal that fall from the electrodes, and under all the electrodes and bags a still larger removable bag. Above the level of the solution are brushes that sweep over the surface of the anodes for removing bubbles of gas and impurities that collect thereon; and scrapers that bear on and remove from the cathodes the film of deposited metal and cause it to drop into a suitable trough. When the electrodes are rotated, current passes from each anode segment through the solution to the opposite cathode segment, the electrode segments out of the solution being cut out of circuit. By the use of a proper solution and current density, the silver is deposited on the cathode segments in a loose, spongy state from which it may be scraped by scrapers into the troughs or bags beneath.

545,328, Aug. 27, 1895, William H. Wiggim, of Worcester, Mass.

Relates to a process and apparatus for refining metals, etc., by electrolysis. The apparatus consists of a series of tanks having guides attached to the inner sides for holding electrodes in a vertical position. Above the tanks is a shaft on which are a plurality of cams, and levers pivoted on supports and their free ends resting on the cams. At an intermediate point, the levers support the electrodes in the tanks. The rotation of the cams oscillates the levers, which in turn impart a sudden shaking movement to the electrodes, removing any accumulated gas bubbles or impurities adhering thereto. The impure metal plates are electrolyzed as secondary electrodes, the electrolyte maintained in circulation by feeding it in at one end and withdrawing it at the other end. The patent contains a reference to patent 467,484.

546,364, Sept. 17, 1895, Donato Tommasi, of Paris, France.

Relates to the refining of metals by electrolysis. The apparatus consists of stationary anodes of the metal to be refined and a rotary cathode, the latter consists of a radial frame mounted upon a shaft, and adapted to receive sectors of conducting material, such as metal, or an agglomerate of carbon and copper oxide, the latter being reduced by the hydrogen, thereby reducing polarization. The rotating cathode sectors are so connected electrically that only the submerged parts are connected, the exposed parts absorbing oxygen from the air and in turn serving to reduce polarization. Scrapers remove the deposited metal as the cathode leaves the solution; the mechanical rubbing also reducing polarization. When metals are to be plated from solutions, anodes consisting of spongy or reduced metal, such as copper, lead, etc., whose oxide is insoluble in the electrolyte, are used; these take up the oxygen, and are later electrolytically reduced in a separate cell. The apparatus and process may be used for refining and obtaining copper, lead, zinc, nickel, silver, tin, argentiferous slags, matte, speiss, ores, etc., also obtaining the silver from Pattinsonized lead or skimmings, etc., appropriate electrolytes being used.

566,673, Aug. 25, 1896, Charles W. Fielding, of London, England, and Louis B. Walker, of Elizabeth, N. J.

Relates to electrolytic refining of metals, using secondary electrodes; the anodes to be either of cast or rolled metal; and attached cathodes of metal or material either similar or dissimilar to the metal in the anode, but preferably made of a paste spread over one side of the anode and allowed to harden before use. The several compound plates are separated by spherical insulators suspended on a cord or other carrier. The spheres touch in points only, and therefore permit the consumption of the entire anode.

569,722, Oct. 20, 1896, John T. Morrow, of Great Falls, Mont.

Relates to the refining of metals and is for a case or receptacle for containing the ends and scraps of anodes. The case may be made of metal less electropositive than the scrap copper in the electrolyte, in which case only the scrap copper will be anodically dissolved; or of non-conducting material and make contact with the scrap copper through a copper shoe resting upon the scrap which is replenished as fast as it dissolves.

570,133, Oct. 27, 1896, William DeCourcy May, of Niagara Falls, N. Y.

Relates to refining metals, and consists of a series of similarly shaped vertically nested pans, which may be made of the metal to be deposited, or of non-conducting material. Electrolyte is supplied to the top pan, and overflows to the one below at one side; from this pan it overflows to the next lower one from the opposite side, thereby circulating the electrolyte in each pan. From the bottom pan it overflows into a receptacle in which it may be purified before re-use. The anode consists of metallic shot or other fragments, which is placed within each pan, the cathode being the underneath submerged side of the pan above, the electrolyte in each pan rising above the level of the underside of the upper pan. When of non-conducting material, a conducting coating is applied as cathode, and suit-

able connection made from the coating to the copper shot within.

578,953, March 16, 1897, Frederic A. Thum, of Newark, N. J. Relates to an apparatus to be used in refining metals, which consists of a long oblong tank, which may also be annular, in which is placed the electrolyte and the cathodes. Suspended within the tank are smaller cells or carriages, supported by wheels running on a track without, and upon which the cell may reciprocate. The cell has a perforated bottom covered by a filter cloth, on which is placed the metal to be refined, either in a fragmentary condition, or otherwise. Anodes made of metal to be refined rest upon the fragments and make contact through a flexible wire and roller to a rail conductor. The cathode may be a plate in the bottom of the outer tank, or may consist of a number of removable pans in which the deposit is made. The reciprocation of the cell stirs the liquid and removes bubbles, etc. Anode slimes are retained on the filter cloth.

588,035, Aug. 10, 1897, William Thum, of Newark, N. J.

Relates to apparatus used in the refining of metals, which consists of a tank having a slanting bottom, with a trough at the lower end of the bottom. The cathode rests on the slanting bottom and projects into the trough. Anode cells similarly shaped, and lined with filter cloth, are slidably mounted in the tank, and have a perforated slanting bottom parallel to that of the tank. The metal to be refined is placed, in a fragmentary state, in the anode cell, contact being made through a removable casting of the metal placed thereupon. Within the trough at the bottom of the tank is placed a removable receptacle of porcelain, etc., into which the deposited metal is pushed periodically. A similar trough and receptacle are in the anode cell, to collect the slimes. The liquid is suitably circulated by an injector, or by compressed air.

588,524, Aug. 17, 1897, Edward Balbach, Jr., of Newark, N. J.

Relates to apparatus for refining silver and other metals, which consists of a tank of earthenware, etc., having a bottom partly flat and the remainder inclined, the entire bottom preferably covered with carbon plates or other conductor. Within this tank are removable skeleton frames which have grated bottoms; the frames contain removable anode compartments having filter-cloth bottoms. Upon the filter cloths are placed the bars of metal to be refined, to which contact is made by removable castings of the metal electrically connected to the positive pole. The deposited metal collects as crystals, etc., and is periodically removed by a scraper. The anode slimes remain upon the filter cloth.

Book Reviews

Metallurgy. By Cecil H. Desch. Second edition; small octavo (18 x 11.5 cm.); 411 pages, 14 plates, 108 diagrams. Price \$3.00 net. New York and London: Longmans, Green & Co.

For this new edition additions have been made to bring the work up to date, while the chapters on the physical properties of alloys and on the metallography of iron and steel have been practically rewritten. The method of handling the subject matter and the author's style are considerably better than in the first edition (which itself, however, was well done), so that this new edition may be unreservedly recommended as a clear, systematic and skilful presentation of the subject.

Liquid Steel: Its Manufacture and Cost. By David Carnegie, assisted by S. C. Gladwyn. Octavo (14 x 23½ cm.), 520 pages, 10 plates and 252 illustrations. Price 25 shillings; in New York \$7.50. London and New York: Longmans, Green and Co.

The author was formerly a Sheffield steel maker, and is now steel works consultant and engineer; his assistant is a consulting engineer. Their chief aim is to compare systematically the costs of all the steel-making processes, giving details of composition and cost of raw materials, construction and ar-

angement and cost of furnaces and plant, and tabulation of labor costs and cost of living in different countries.

The work, however, contains more than this; it gives incidentally a larger amount of information about the processes themselves, and the principles upon which they are based and worked, given in a clear and succinct manner.

The antiquated £. s. d. make the reading and comparison of costs very unsatisfactory to anyone but a Britisher; but we feel thankful for cwt. and quarters having been replaced by tons and pounds. The illustrations, plates, paper, type and binding are all satisfactory. The book will prove valuable to a large circle of technologists.

The following figures and conclusions from the work may interest others than mere technologists: They give in relative figures the wages received for the same kind of labor, the expense for the same kind of living, and the relative state of "affluence" of the workingman, in the countries named, taking England in each case as the standard:

	Germany	England	America
Wages received	75	100	200
Cost of living	120	100	150
"Coefficient of comfort"	62.5	100	133

In brief, the German workman is on an average 37½ per cent. worse off and the American 33 per cent. better off than the English; or the American may be said to be 113 per cent. better off than the German. These figures are strictly true only of the iron and steel industry, but they agree fairly well with the reviewer's observations and conclusions in other industries; they therefore approximate to being true in general.

* * *

A New Era in Chemistry. Some of the most important developments in general chemistry during the last quarter of a century. By Harry C. Jones. 326 pages. Price \$2.00 net. New York: D. Van Nostrand Company.

"Although the development of chemistry has taken place more or less by leaps and bounds, it is difficult to fix an exact date as the beginning of the 'New Era.' The year 1887 marks the appearance of the first volume of the *Zeitschrift für Physikalische Chemie*, and in this volume the epoch-making papers by Van't Hoff, on the Relations between Solutions and Gases, and of Arrhenius on the Theory of Electrolytic Dissociation appeared. This date, 1887, is, therefore, taken as the beginning of the New Era, which is, then, a quarter of a century old.

"It is, however, clearly recognized that there is a difference between the chemistry of twenty-five years ago and that of to-day. That this is really a difference in kind is not always clearly understood, either by those who have watched the development during that period or by those who have not studied chemistry long enough to have the necessary perspective.

"It has seemed desirable to point out as clearly as possible in what this difference consists; how these new developments were brought about and by whom. This is the object of this little work."

There are twelve chapters dealing with the conditions of chemistry in 1887; development of the law of mass action; the energy changes that take place in chemical reactions; Van't Hoff, Le Bel and Guye and the origin of stereochemistry; the phase rule of Willard Gibbs; chemical dynamics of Van't Hoff and chemical and chemical equilibrium of LeChatelier; the rôle of osmotic pressure in the analogy between solutions and gases; Arrhenius and the theory of electrolytic dissociation; the saltate theory of solution and the importance of solutions for science in general; the work of Wilhelm Ostwald in inaugurating the new era in chemistry; investigations by the students and coworkers of Wilhelm Ostwald; the electron and radiochemistry.

This book is testimony of a sympathetic nature and its charm rests to a large extent in the author's personal reminiscences of Van't Hoff, Arrhenius and Ostwald. The title of the book is undoubtedly too broad for its contents, but those phases of the new era of chemistry which are connected with these three men are covered with genuine enthusiasm.

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Organized Research in America

In his brilliant recent presidential address before the American Chemical Society, Mr. Arthur D. Little remarked pointedly that while "Germany has long been recognized as pre-eminently the country of organized research," yet its pre-eminence in industrial research is now being threatened by a new competitor, which is "strangely enough the United States—that prodigal among nations still justly stigmatized as the most wasteful, careless and improvident of all." While most of the industrial effort in this country still proceeds under the guidance of empiricism with a happy disregard of basic principles, yet many of our manufacturers have already learned that "efficiency of production is a sounder basis for prosperity than mere volume of product, however great" and that the most profitable output of their plant is "that resulting from the catalysis of raw materials by brains." Catalysis of raw materials by brains—this is industrial research.

Mr. Little's record of what industrial research in America has so far accomplished is a long and noble list of achievements, which promises much for the future. But what we want to comment on especially in this note is not so much what has been achieved as how it has been achieved—or as it has been sometimes put, though erroneously, "the question of individuality versus organization."

Most of the early achievements must be credited straight to the genius of strong individualities of superman size. Edison is a typical example. And that the Edison spirit is not yet dead, that specialized intense research is still being carried out by independent individualities and can be carried out with success, Dr. Backeland has proved more than once. But it is true that under the new, more complicated social and industrial conditions of modern times most of the problems inviting research now require the facilities and resources of large corporations, and that organized research is replacing independent research.

Though this is perfectly true, it would be folly to imagine that organization can ever replace individuality. Research can be organized, but it will always require individualities of the most pronounced character. At this place last month we paid a tribute to the latest great achievement of the Research Laboratory of the General Electric Company—the two-candle per watt lamp. In Mr. Little's address we find an interesting account of how work is carried on at that laboratory. Dr. Whitney is quoted as follows: "We see a field where it seems as though experimental work ought to put us ahead. We believe that we need to get into the water to learn to swim, so we go in. We start back at the academic end as far as possible, and count on knowing what to do with what we find when we find it. Suppose that we surmise that, in general, combustible insulation material could be improved upon. We try to get some work started on an artificial mica. Maybe we try to synthesize

it and soon come to a purely theoretical question; e.g., is it possible to crystallize such stuff under pressure in equilibrium with vapor corresponding to the composition of real mica? This may lead a long way and call in a lot of pure chemistry and physical chemistry. Usually we just keep at it so that if you haven't seen it in the market we're probably at it yet." Clearly work of this kind requires strong individualities, but the work of the different research men is held together by their organization under a leading master spirit and by a mutual exchange of ideas between all at weekly colloquies.

Now the problem looms large: What can the Government, represented by the Government bureaus, do to help industrial research in this country? A broad, positive answer is difficult. But this much can be said that being a government of the people and by the people it should never undertake anything that would bring it into competition with individuals or classes of the people. The scope that remains is large. The Reichsanstalt in Germany has shown how to do it; so has the Bureau of Standards in Washington, which was justly congratulated at the last iron and steel meeting of the American Institute of Mining Engineers on its excellent research on the critical ranges A₂ and A₃ of pure iron. The problem is a peculiarly trying one for the new Bureau of Mines, but this Bureau is to be congratulated at least on its willingness to confer concerning the organization of its research work with the representative national scientific and engineering societies.

Competition Among Metallurgical Processes

Progress is rapid and change is frequent in the metallurgical industry. Just as one branch of ore treatment seems to have reached the limit of its efficiency, and the demand seems imminent for a radical change in methods, ideas are evolved that give the older methods a new lease of life and defer the possibility of success of the new proposals. We have in mind particularly the situation in regard to complex base-metal ores, and the relative possibilities of mechanical concentration, hydrometallurgy, and electrometallurgy in their treatment.

Advocates of each type of process are enthusiastic and optimistic as to the advantages of their respective favorites. Perhaps concentration is on the defensive; for the admitted limitations of physical processes have acted as a spur to the hydrometallurgist and electrometallurgist. At the same time, the deficiency of mechanical concentration, fully recognized by the loyal experts in that field, has forced them to perfect and adopt improvements that will strengthen their position against their possible competitors. And so the competition grows keener and keener, to the ultimate benefit of the industry.

There are several matters in this number of the journal that will afford food for reflection to all, and some degree of encouragement to each of these factions. The discussion on the present position and future prospects of zinc metallurgy at the recent joint meeting of the New York sections of the American Electrochemical Society and the American Institute of Mining Engineers, reported in full in this issue, is interesting in this respect, though the general discussion was rather tame compared with the discussion on the same

subject at the last Denver meeting of the American Electrochemical Society. While the remarkable advances made in the concentration of complex zinc ores in recent years were not touched upon in the meeting at New York, yet the relative present position of the retort process, the electric-furnace process, and the electrolytic process came out very clearly, as well as the position of some of the chief representatives in the different camps, optimists and pessimists and agnostics.

Of great encouragement to the ore-dressing fraternity will be the note of improvement in dry concentration and separation, as recorded in this issue in the description and work of the Plumb pneumatic jig. It suggests possibilities that ought to be interesting to those who are not now getting satisfactory returns from their labors. It emphasizes the enhanced value of high-grade as compared with low-grade concentrates, and the necessity of working to meet the smelter's demands.

Imperfections reside in all of our processes, but the different types merit consideration. Their ultimate success or failure will be determined by the net profit resulting from their application. In this regard, a discussion started on another page, for and against a hydrometallurgical scheme, should serve a useful purpose. Several other communications referring to the same subject came too late to be printed in this issue and have been held over for the January issue. It is better to bring ideas to a focus in print than to talk aimlessly and in generalities. As suggested in the discussion, if concentration is to hold its own it must awake to the possibilities of improvement, and reduce the tremendous losses entailed in poorly separated products; and the goal of competitive metallurgical processes must be a cost of production so low that it can not be ignored.

The Modern Blast Furnace Burden

According to the official statistics the manufacture of 29,726,937 gross tons of pig iron in 1912 involved the consumption of about 55,656,000 gross tons of domestic and foreign iron ore, ore briquets, etc., and about 4,319,000 tons of mill cinder, scale, scrap, slag, zinc residuum, etc. As the latter item is only about one-thirteenth the former it does no great violence to add the two together, and compute that the average furnace burden was equivalent to 2.017 tons of ore per ton of pig iron manufactured, making a yield of a shade under 50 per cent. Without undertaking to go into details as to the loss of metallic iron in slag, flue dust, etc., it is evident that the average iron content of the ores used was less than 50 per cent, since the average pig iron contains only about 95 per cent iron.

Between 80 and 85 per cent of all the pig iron of the United States is made from Lake Superior ores, while the imported ores are of relatively high grade, leaving it that the comparatively low-grade ores used by Southern furnaces do not operate greatly to pull down the average yield. The showing is much less favorable than would have been made, say, fifteen years ago, when the cream of the Lake Superior deposits was being used. Unfortunately there are no statistics for such years, but it is readily recalled that ores running well above 60 per cent were the rule rather than the exception.

The ores have been changing and the conditions of operation change with them. The pig-iron producer adapts himself to the changes without inconvenience. The difference in conditions appears on the surface not in the form of its having become more difficult to operate a blast furnace, but from its being impossible to produce pig iron as cheaply as formerly. More ore must be mined and transported, and more fuel and flux must be consumed, simply adding to the expense.

It should be noted that the gradual exhaustion of the Lake Superior deposits which were first selected for exploitation does not always lead to the mining of a lower-grade ore. There are three factors, the iron content, the character of the impurities, and the cost of mining. In the early days of the Mesabi range, when ore was literally "as cheap as dirt"—certainly if one undertook to place orders for "dirt" in hundred-thousand-ton units he would find it rather expensive—if a 60 per cent ore lay in 100-ft. thickness on the surface and a 65 per cent ore of the same thickness lay under 100 ft. of overburden the former would have been mined and the latter left as impractical at the time, but today conditions would be different. Very good ores are now being mined, at great outlay of capital in removing great overburdens, and thus the exhaustion of the earlier mined deposits leads in one direction to the mining of lower-grade ores, but in another direction to the mining of deposits which are very good, but expensive to get at.

The ease with which pig iron is made with a much more expensive burden than that of fifteen years ago suggests how easy it will be to continue using lower and lower grade ores. We shall make tens of millions of tons of pig iron, perhaps nearly a hundred million tons, for every unit that the average ore used loses in iron content, and in making that pig iron we shall obtain ample experience to attack the new problems successfully. Inasmuch as the visible supply of iron ore increases very rapidly for each notch that the limit of commercial availability is dropped it is clear that iron for future generations can readily be found. A higher price, measured by unit of endeavor, will have to be paid, but through the endeavor being better directed the cost will not be enhanced in proportion. If the cost should advance when measured by dollars per ton, the public will undoubtedly have a correspondingly increased ability to pay. The consuming public can now pay the average price of \$14 a ton which pig iron now commands much more easily than it could pay the \$9 to \$10 per ton which pig iron made from Lake Superior ores brought at its historic low point, late in the nineties, when very rich ores were being used.

An Old, Old Question Considered Prismatically and an Humanistic Hortation

Let us take a ray of sunlight and pass it through a prism. We all know what happens. The white light is decomposed into its optical components and we have the spectrum of red, orange, yellow, green, indigo, blue, and violet. What was a co-ordinate entity has been resolved into several entities and each not separate but blending into its neighbors.

Now compare the flux of social thought, which is made of many divers hues to a decomposed ray of light. We have two extreme classes of thinkers, one whose stomachs

are full and whose souls are self-sufficient, and who quack solemnly "whatever is, is right" and we have another class of iconoclasts with lean and hungry looks, who say "whatever is, is wrong." Between these two opposed classes, we have those of lofty cosmocentric souls, who chant "whatever is true, must be right, let us advance slowly and surely onward and upward to the right." These classes grade into each other in many ways for the individual himself will be a conservative in one subject and a progressive in another subject. A man will yell "halt" and try to sit on the tail of progress in metallurgy while he will be a contributor to the funds of the Socialist party. We see that it takes all sorts of people to make up this round world.

Now all these discordant anthropological elements in the eyes of the Great Intelligence make up a white sunbeam of spiritual light. We should, therefore, be not too impatient with the "standpatter" if we have radical views on a question, nor damn with faint praise the efforts of the sane upbuilder if we are recognized by ourselves and by others as authorities on any subject. For physics teaches that a ray of light to be pure white must have many seemingly diverse elements.

Now physics teaches also that outside the visible spectrum, there is the invisible. Thus we have the infra-red which gives heat without light and the ultra-violet that gives light, could our purblind eyes detect it. So too, in politics and technology we have the commonplace reactionary who fires away, sputters and gives no real light on the points at issue and the far-sighted visionary who throws his rays into the projective and transcendental future and is described as a wild-eyed crank by those round about him.

But between our physical vision and our psychological vision there is this important difference. What colors our eyes can distinguish to-day we will see as the same to-morrow, next day, and forever. But what we see mentally as truth and reality does not necessarily see the same to-morrow or next day, for we are being swept along in the great flow of thought and as the world moves, we nolens, volens must move with it. So what once was thought as unbelievable becomes belief, and the agnostic gets to know the unknown and sees the invisible.

This age is an intensely practical age. Such is a plain fact, but it is likewise a plain fact that this age is an intensively theoretical age. Most of the foregoing has been speculatively and theoretically considered. Now we will ask the reader to look at the question in a humanistic and direct-acting way, in a manner and in a way that will affect his own life. Which should a person be—a wise, cheerful, sane optimist or a crabbed, aged pessimist?

Now the sane optimist subdues his pessimism and uses it to cheer his spirit. He knows full well the motto "nisi parer, imperat."

Which should a right-thinking person be? Might he not consider this: "Between the optimist and the pessimist the difference sure is droll, the optimist sees the doughnut and the pessimist sees the hole." We may add that the calm, self-contained optimist usually eats the doughnut as soon as he can see it. And the traditional food of New England is more substantial than the circumspect hole or the circumambient air. The community of pessimist-subduing, early-rising optimists is a good society to join.

Readers' Views and Comments

Reduction of Blue Powder

To the Editor of Metallurgical and Chemical Engineering:

Sir: My attention has been called to an error appearing in my article in your November issue, page 637, in which I state, "Moreover, Dr. Haanel has informed me that he has melted tons of blue powder, after mixing it with carbon in proper form and amount." Although I believed I was stating a correct impression received in conversation, it now appears that the statement is unqualifiedly wrong, and I wish to make a complete retraction to relieve Dr. Haanel from any wrong implication.

We had agreed that the excessive production of blue powder was not necessarily objectionable if it could be cheaply converted into spelter. Probably it was in this connection that I misunderstood Dr. Haanel's remark.

Denver, Col.

F. L. CLERC.

Electric Tin Smelting

To the Editor of Metallurgical and Chemical Engineering:

Sir:—On page 593 of your October issue an article is referred to under the heading "Electric Tin Smelting," given by "A Special Correspondent" in the *South African Mining Journal*, June 7 and 14, 1913.

On reading this through I find that practically all the information given therein is identical with the data and conclusion given in my article, "Electric Tin Smelting," published in your journal, Vol. IX, No. 9, page 453, September, 1911, although no reference to that article is made in the review.

Luton, Beds, England.

JOH. HÄRDÉN.

Annual Tables of Physical and Chemical Constants

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I desire to make public acknowledgment of contributions made this month toward the expenses of publishing the "Annual Tables of Physical and Chemical Constants": New Jersey Zinc Company, \$50; International Acheson Graphite Company, \$25; Castner Electrolytic Alkali Company, \$25; Mathieson Alkali Company, \$25.

Lehigh University.

JOS. W. RICHARDS.

Induction Furnace Notes

To the Editor of Metallurgical and Chemical Engineering.

Sir:—In my article on the above subject, published in the October issue of your journal, a few minor errors occurred, which I now wish to correct.

(1) As to the thermal efficiency (page 561) an error appeared in the formula given. It should read

$$E = \frac{350,000}{804 \times 606} = 67.1 \text{ per cent, not } 63.8 \text{ per cent.}$$

(2) The cooling water was found to be just under 1.4 gal. per min. or round 50,000 gal. for 160 tons of steel. As the cost of the water from the mains used is 6d. per 1000 gal. This works out to a little under 2d. per ton of steel.

(3) The tilting arrangement is not used to any extent at present, as it was found that the use of a sound taphole was more beneficial to the lining generally, and also probably to the quality of the ingots.

(4) The tar-magnesite lining has again been abandoned. The first mentioned method of lining has been taken into regular use, though with a few technical alterations, the nature of which is not divulged for publication. When the men got thoroughly used to its behavior, cracks in the lining are now a very rare occurrence and need not be feared.

(5) The labor casts have been revised and are slightly higher than given in my schedule; in addition, two "shank men" at \$8 are employed per shift. The smelters receive about 98

each per shift, and the laborers' wages are also slightly higher. Against this stands the fact that the necessary margin for sundries has been materially lowered so that the total cost per ton remains pretty much the same as stated.

Luton, Beds,
England.

JOH. HÄRDÉN.

Micrometry as Applied to Alloys

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The review of the very interesting paper, "Micrometry as Applied to Alloys," by Dr. C. H. Mathewson, in your November issue, calls to my attention a method of measuring the areas of alloy constituents which I have used very successfully. It is a modification of Sauveur's polar planimeter method combining its accuracy with the shortness of the ruled square method.

Two ways of measuring areas with the planimeter are now in general use: (1) taking a photomicrograph of the alloy and measuring the areas from a print, and (2) projecting an image of the alloy on a screen and tracing the areas to be measured with a lead-pencil. The penciled areas are then measured with a planimeter.

I used a screen consisting of a piece of plate glass, $\frac{1}{4} \times 15 \times 15$ in., coated on one side about $1/32$ in. thick, with a mixture of paraffin and tallow. If now the coated glass be used with the waxed side up, in conjunction with a Bausch and Lomb photomicrographic camera, the areas can be traced in the wax with a planimeter according to Sauveur's method. The camera should be in a vertical position thus making the glass horizontal. It is best to confine the waxed portion of the glass to an area about 4 in. diam. In this way the fixed end of the planimeter can be so placed that the roller moves on the glass without danger of encountering the wax.

The wax used was obtained from candles and the amounts of paraffin and tallow were not determined. It is evident that an opaque wax is desirable since it permits the use of a thinner coating. If the coating is too thick and the areas to be measured are small, the tracing point of the planimeter has a tendency to break the wax when an area is about seven-eighths surrounded. In discussing this difficulty with Mr. F. A. Fahrenwald, 2168 Stearns Rd., Cleveland, O., he suggested mixing some white powder with paraffin to make it more opaque and thus permit the use of a thinner layer. Of several powders tried we found that starch was the best. The paraffin is first melted and then powdered starch added slowly with vigorous stirring, until the mixture is pasty. We found that glass coated with a layer of this material .008 in. thick would give a much better image than ground glass.

After a determination has been made the wax can be melted by applying the flame of an inverted bunsen burner, after which the plate is ready for another determination.

ZAY JEFFRIES.

Case School of Applied Science,
Cleveland, Ohio.

Electric Zinc Smelting

To the Editor of Metallurgical and Chemical Engineering:

Sir: Referring to the article on Electric-Zinc Smelting by Mr. Louvrier in your November number, will you kindly allow me space to disclaim the intention of imputing to him, in my paper of last August, the expression of the opinion that "the production of carbon dioxide in actual electric furnaces, was approximately equal to that formed in the Belgian retort."

On the contrary, I understood him to say that "the excessive production of blue powder in electric furnaces is due to the formation in the electric furnace of a much higher percentage of carbon dioxide than in the usual retort."

I expressed as my own opinion, that either he was mistaken in this opinion, or that the excess of oxygen came from another source; for the reaction $\text{ZnO} + \text{CO} = \text{Zn} + \text{CO}_2$ will give the same amount of CO_2 in either case.

I also ventured to express my opinion that the reaction $\text{CO}_2 + \text{C} = 2\text{CO}$ would be equally rapid in both cases.

I also stated that in my opinion, the electric current, by forming cavities in a paste, partly fused charge, might prevent the intimate contact required between the carbon dioxide formed and the hot carbon, in order that the second reaction could take place.

I offered as proof of this, that a rapid reduction of zinc could not take place unless the carbon dioxide was rapidly reduced, and suggested that some of the carbon dioxide escaping into the slag-lined passageways, might pass into the condenser without an intimate contact with carbon.

The amount of carbon dioxide formed being the same in both cases, the percentage of it which passes into the condenser in an electric furnace, may thus be much greater than in the Belgian retort. This may be the result either of a defective construction in the particular furnace under consideration, or of some inherent difficulty in reducing zinc at the temperature of an electric furnace.

At the time my article was written I had no idea that Mr. Louvrier had designed a zinc furnace, and, of course, had no idea of criticising it.

This is plainly a case of misunderstanding which should be avoided if possible, and I wish to dispel in the minds of Mr. Louvrier, and any of your readers, the thought that I have treated him unfairly.

Denver, Col.

F. L. CLERC.

Hydrometallurgy: Joys of Its Theory, Woes of Its Practice

To the Editor of Metallurgical & Chemical Engineering:

Sir: In reading Dr. Chauvenet's objections to "hydro schemes," in your September issue, I wondered why he selected for attack dry chlorination as typical of hydrometallurgical processes, for not a single "woe" that he gloats over applies to the method in its present development. He surely must have had in mind chloridization of a period much earlier than the "early attempt at Helena," or the developments at Corbin, Mont. As well take as an example of current cyanide practice, the process in its stage of development fifteen years ago, as to refer to chloridization today in terms of concentration roasting, sodium chloride electrolysis, ferric hydroxide precipitation, water washes, diluted solutions and fractional precipitation.

The prominence of dry chlorination as the most likely process in the field for the solution of the complex-ore problem is attested, not only by a score of engineers of unquestioned ability, but now by its unexpected selection as a typical example of "hydro schemes," with no specific "woe" charged against it.

If overcoming a simple problem of eliminating ferric hydroxide is to be termed "much ingenuity," "we" would bankrupt the English language in trying to express what has been done to overcome the *real* difficulties which have been encountered and successfully solved in the development of chloridization.

What engineer of real training and experience would offer as proof of the success of any hydrometallurgical process, the results of a test, whether on a gram, pound, ton or carload, if such test had not been made under commercial conditions of apparatus, reagents and time. Continuity of operation has no small bearing on the testing of a process. I do agree that engineers should know the relation of test work to actual practice. My experience has convinced me that the principal difference between a test on a few pounds and one on a ton (if both are intermittent) is the difference in the cost of making such tests.

Why eliminate as a class of material for wet treatment the available complex zinc ores which are not too low-grade, but rather too complex? I submit that in the consideration of

any scheme, its field of operation and the purpose for which it is being developed should determine largely its possibilities. In other words: Will we "get a run for our money?"

In the October issue of your journal, Mr. Bleeker defends in a general way hydrometallurgical possibilities, but he has touched upon chloridization under the so-called Malm process just enough to expose his ignorance of the subject by his reference to its theory and calcium sulphate.

I am interested in the development of chloridization solely from an established conviction that it offers the solution of the complex-ore problem. To this end I am prepared to defend it, and welcome any sincere, specific criticism devoid of repartee or hair-splitting. This will offer to Mr. Bleeker an opportunity to enlighten us further on "calcium sulphate in the Malm process," and I trust will make available these columns for a discussion of the relative merits of various proposed schemes of dry chlorination.

To establish my position, I state that my experience with chloridization has been through the gram to the ton test in as thorough an investigation as intermittent tests will permit, then to a test with commercial-size apparatus under conditions indicated by the work on a smaller scale. This latter installation was made at Corbin, Mont., and proved a failure under the methods which the preliminary tests indicated would work successfully. From this failure was evolved a method which of necessity discarded all specially constructed apparatus, and made use of standard apparatus excepting only the special construction of minor parts and the development of a simple, cheap, fool-proof electrolytic cell for the decomposition of zinc chloride.

This work covered a period of three years, during which time the weak points of the process became apparent and were overcome. The difficulties encountered were solved in such a way as to simplify the method, so that in its present development all of the "bugaboos" mentioned by Dr. Chauvenet (many of which never existed) have vanished into thin air.

I stand for a form of chloridization which, at least to myself, has proved a success in yielding a high recovery of gold, silver, copper, lead and zinc in metallic condition at a cost, under normal Colorado conditions, of 2½ cents per pound of metal recovered. In the last analysis the success of any commercial operation is determined by its net financial return.

I therefore have no proof of the success of chloridization. If there were the case the plant at Georgetown, Colo., would not be in its uncompleted state. It was to bring the process to a stage of commercial proof that the installation at Georgetown was undertaken. At present the plant is about two-thirds completed. It was designed on data obtained from the Corbin plant and from later special investigations, to treat all ores, whether sulphide, oxide or carbonate, but more particularly complex ores of gold, silver, copper, lead and zinc. The present installation bears evidence as to the comparative simplicity of apparatus and operation proposed.

During the eight years which I have given to this method of ore treatment, I have not encountered a difficulty which has not been readily solved, and no difficulty is foreseen which is any greater than those which are the regular diet of engineers who do things. Oh, yes, "the cost will finally prove its death." In that connection, let me state that the value of ores available will permit the cost to more than double and still show a substantial profit on ores which have no market value at present.

JOHN L. MALM.

Denver, Colo.

Great Falls Electrolytic Copper Refinery

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The paper on the "Great Falls Electrolytic Plant," by Willis T. Burns, presented at the Montana meeting of the American Institute of Mining Engineers and published, slightly abstracted, in your September issue, has brought up some thoughts which I herewith offer.

Re Purification of Electrolyte by means of insoluble anodes, the writer states that using insoluble anodes with a speed of flow of electrolyte of 4 liters per minute four tanks removed 99.3 per cent of the arsenic, leaving the bismuth still in solution. In December, 1910, Harold Schroeder in the *Australian Mining Standard* published details of a similar method of purification of electrolyte conducted at Lithgow, N. S. W., in 1908.

The object he concisely states under "Scheme" as "to purify the liquor and at the same time produce the least possible quantity of impure copper." To effect this, as much pure copper as possible is deposited by sending the liquor through a cascade of tanks with insoluble anodes just sufficient to ensure the lowest one giving pure copper on last cathode. In this plant five tanks were used, for the double reason that that number necessitated no alteration in the ordinary flow arrangements and that it still left the ordinary number of insoluble-anode tanks in use for keeping the copper strength of all electrolyte on works at a constant figure. The outlet from the fifth tank was used to fill tanks in which the impure copper was to be made, no circulation of liquor taking place in those tanks beyond the ordinary air-lift circulation in each tank.

The figures published show a steady and regular daily deposit of the arsenic, but the bismuth was deposited much more rapidly than the arsenic. In fact where seven days were required to remove the bulk of the arsenic, the bismuth was all removed from solution in less than half that time. This is confirmed by the analysis of the deposits, these being divided into hard cathode, metallics and fines of the powdery part, the ratio of bismuth to arsenic always being higher in the metallic portion.

Having personally supervised the whole of this experimental purification, I may say that what suggested this method of cleaning electrolyte was largely, having noted on several occasions that when the circulation of liquor was accidentally stopped or slowed down during the night in the insoluble anode tanks, the tanks at the end of the cascade produced dirty copper, and sometimes a cathode had the top 6 inches black and the lower part clean bright copper. Samples of electrolyte from the tank containing such a cathode, drawn off in zones from top to bottom of tank, showed differences as to copper, arsenic and bismuth contents. To this H. Schroeder draws attention.

Our experience on the Australian works differs completely from that given in Mr. Burns' paper.

It would be of interest to know if the removal of the first of the copper by crystallization has been found more economical than the making of pure cathode copper till the solution reached the "danger point" for such.

Re the Treatment of the Slimes and Impure Copper Produced, it would also be of interest to know whether any other method than blast-furnace treatment has been tried; further, whether the anodes cast from converters at a time when much of this impure copper was coming around, carried a higher percentage of arsenic than the average anode. It is worthy of time to evolve a method of treatment by which a minimum of the arsenic would eventually return to the liquor, whether from blast-furnace matte or the after-treatment of flue dust from the roasters or converters that expelled it from the impure copper deposit.

In another part of the paper the writer states, "all of the silver present in the cathodes is deposited mechanically from the slimes in suspension." This I find very interesting as bearing out what I have always maintained, and we have proved conclusively at the works at Lithgow by the use of sand filters and by analysis of cathodes. Details are given in E. H. Blake-more's paper on "Electrolytic Refining," read before the Australian Institute of Mining Engineers, and published in its *Proceedings*, February, 1910.

It is to be regretted that the author of the paper on the Great Falls plant makes no comment on the use of chlorine in the electrolyte, which I see was kept at 0.40° grm. per liter.

* The figure should be 0.044. The figure in the September issue was a misprint. Editor.

In Lithgow we proved it to have a decidedly bad effect in relation to the gold and silver in the cathode copper.

Gourock, Scotland.

HARRY A. B. MOTHERWELL.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Replying to Mr. Harry A. B. Motherwell's comments above on my notes on the Great Falls Electrolytic Plant reprinted in your September issue I wish to say that the bismuth in the copper treated at Great Falls is present in very small quantities, as compared with the other impurities, and as it has shown no tendency to increase in the electrolyte its elimination has not been studied.

Since the receipt of Mr. Motherwell's comments a bismuth determination has been made on samples of the inlet and outlet solutions of a group of four insoluble anode tanks employed in purifying electrolyte. No bismuth was found in either sample. Had bismuth been present in these solutions it would probably have been deposited with the arsenic.

Regarding the economy of first removing the copper from the impure electrolyte, by crystallization, before treating in the insoluble anode tanks for the removal of impurities I may say that this method has been found much more economical than attempting to obtain commercially pure copper in insoluble anode tanks operated at 35 amperes per square foot when the electrolyte is circulated at a rate of speed sufficiently slow to insure the deposition of a large amount of arsenic. Even were it possible to recover most of the copper in this way we would find the scheme impracticable as our working electrolytes would become depleted in copper as the increase in copper contents of the electrolyte, per ton of copper refined, decreases as the current density increases.

Regarding the arsenic contents of anodes produced at the smelter while the arsenical slime from the insoluble anode tanks is being treated there, it may be said that no material increase in impurities in the anodes is noted.

The purpose of adding chlorine to the electrolyte is to precipitate antimony in the form of an oxychloride. We have observed no effect of the chlorine on the silver and gold contents of the cathodes.

W. T. BURNS.

Great Falls, Montana.

Conductivity of Fluosilicic Acid and Lead Fluosilicate

To the Editor of Metallurgical and Chemical Engineering:

Sir: As there is, to my knowledge, no published data on the conductivities of fluosilicic acid and lead fluosilicate, the constituents of the electrolyte used in electrolytic lead refining, I thought the following tables might be of interest.

These results were obtained in the Electrochemical Laboratories of the Pennsylvania State College. The method employed was that familiar to anyone who has worked with conductivities, namely, the one due to Kohlrausch, which is essentially the standard Wheatstone bridge method of resistance measurement, with the exception that an alternating current is substituted for a battery and a telephone receiver for a galvanometer. Before making any measurements the slide-wire bridge was calibrated by the method of Stronhal and Barus, and all subsequent readings on the bridge were corrected from the calibration curve obtained. Two conductivity cells, having platinized-platinum electrodes, were used, one cell having a high and the other a low "cell constant."

The acid used in making up the solutions was the pure concentrated fluosilicic acid, a clear syrupy liquid, bearing the following analysis:

N. V. M. = 0.005 per cent

SO₂ = 0.002 per cent

The salt used was the crystallized lead fluosilicate of the formula PbSiF₆ (4H₂O).

All results given in the tables are the means of at least three readings.

Table I gives the conductivity and equivalent conductivity of fluosilicic acid at 25 deg. C.

TABLE I

Conductivity and equivalent conductivity of fluosilicic acid at 25 deg. C.

Normality	Per cent of acid in solution	Conductivity in reciprocal ohms per centimeter cubed	Equivalent conductivity
N	7.2	0.087	87.
N/2	3.6	0.0469	93.9
N/4	1.8	0.025	100.
N/8	0.9	0.0133	106.5
N/16	0.45	0.00708	113.2
N/32	0.23	0.00372	119.9
N/64	0.11	0.00192	123.
N/128	0.06	0.00102	130.9
N/256	0.03	0.000602	154.
N/512	0.015	0.000384	106.9
N/1024	0.007	0.000254	271.
N/2048	0.0035	0.000156	320.

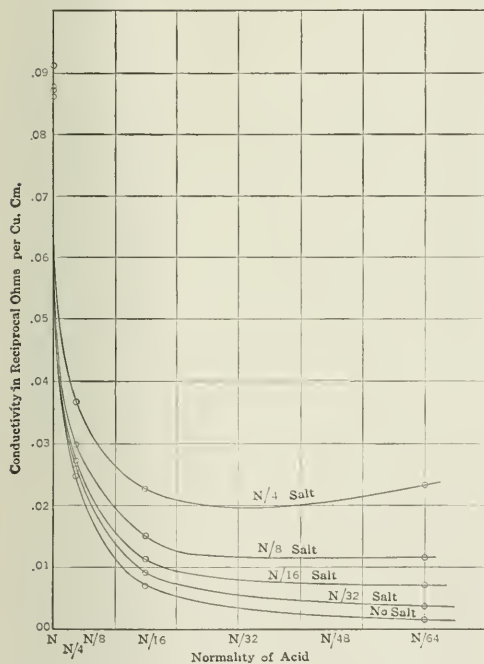


FIG. 1.—EFFECT ON THE CONDUCTIVITY OF FLUOSILICIC ACID OF DIFFERENT NORMALITIES OF LEAD FLUOSILICATE DISSOLVED IN THE ACID

TABLE II

Temperature coefficient of conductivity of normal fluosilicic acid (corresponding to 7.2 per cent free acid) from 20 to 45 deg. C.

Temperature in degrees centigrade	Conductivity in reciprocal ohms per centimeter cubed	Equivalent conductivity
20	0.0806	80.6
25	0.0806	86.6
30	0.093	93.
35	0.099	99.
40	0.104	104.
45	0.1092	109.2

Temperature coefficient of conductivity equals 0.01450 per

degree centigrade between 20 and 45 deg. C. referred to 18 deg. C.

TABLE III

Temperature coefficient of conductivity of 1/16 normal fluosilicic acid (corresponding to 0.45 per cent free acid) from 20 to 45 deg. C.

Temperature in degrees centigrade	Conductivity in reciprocal ohms per centimeter cubed	Equivalent conductivity
20	0.00669	107.
25	0.00715	114.2
30	0.00754	120.6
35	0.00792	126.9
40	0.00830	131.1
45	0.00849	135.9

Temperature coefficient of conductivity equals 0.001106 per degree centigrade between 20 and 45 deg. C. referred to 18 deg. C.

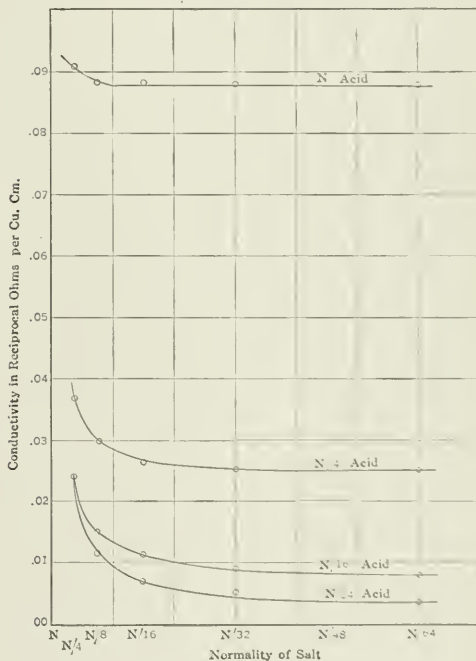


FIG. 2.—EFFECT ON THE CONDUCTIVITY OF LEAD FLUOSILICATE WHEN DISSOLVED IN DIFFERENT NORMALITIES OF FLUOSILICIC ACID

TABLE IV

Conductivity table of various mixtures of fluosilicic acid and lead fluosilicate at 25 deg. C.

	N acid	N/4 acid	N/16 acid	N/64 acid
N/4 salt	0.0068	0.0367	0.02278	0.0230
N/8 "	0.0084	0.0300	0.01510	0.01175
N/16 "	0.0084	0.0273	0.01122	0.00725
N/32 "	0.0076	0.0258	0.00932	0.00470
N/64 "	0.0076	0.0254	0.00838	0.00362
N/128 "	0.0076		0.00702	0.00265
N/256 "			0.00772	0.00224
N/512 "			0.00750	0.00200
N/1024 "			0.00750	
No salt	0.0070	0.0250	0.00708	0.00102

Table II gives the temperature coefficient of the conductivity of normal fluosilicic acid (corresponding to 7.2 per cent free acid) from 20 to 45 deg. C.

Table III gives the temperature coefficient of the conductivity of 1/16 normal fluosilicic acid (corresponding to 0.45 per cent free acid) from 20 to 45 deg. C.

Table IV is a conductivity table of various mixtures of fluosilicic acid and lead fluosilicate at 25 deg. C.

When the results obtained in Table IV are plotted, as in Figs. 1 and 2, they show, at a glance, the great effect that the free acid has in increasing the conductivity of the electrolyte.

Niagara Falls, N. Y.

T. ARTHUR PATTERSON.

The Continuous Sand Plant in Cyanide Practice on the Rand

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In your issue of July 1913 appears an article by Mr. H. N. Spicer in regard to Rand methods, and under the heading "Cone Classification of Sand and Slime" Mr. Spicer refers to rotary vacuum filter tables of local invention in use at the City & Suburban G. M. Co., and alleges certain disadvantages in connection with the use of these tables.

The tables referred to, and also the special type of cone used in conjunction with same, are the invention of Dr. W. A. Caldecott, and the method of handling sand by means of table and cone in the manner described is known as the "Caldecott continuous sand plant," and as the writer has had a great deal to do with the installation of these plants, as Dr. Caldecott's representative, he would like to comment upon Mr. Spicer's article. The alleged disadvantage, as stated, are based on faulty premises.

Loss of Head.—Mr. Spicer seemingly overlooks the fact 85 per cent of the total weight of pulp, in the shape of slime and water, overflows the diaphragm cones, and return-sand cones above the filter table, and the total loss of head of this amount of pulp is only about 2 ft., which can hardly be considered a serious matter. The remaining 15 per cent of the pulp is the only part which has any considerable loss of head, as it is this amount of the total pulp which is delivered as a thick candy pulp from the underflow of cones on to the sand filter table.

These percentages are based on ordinary Rand conditions of 5½ parts water to one of solids in the pulp from the mill, and with cone underflow containing not more than 30 per cent of moisture. The sand delivered by underflow of cone amounts to 50 per cent of the original solids; the other 50 per cent being slime which overflows cones.

The disinclination to any loss of head requiring elevation and return of a portion of the pulp in a reduction works, whatever advantages may be derived therefrom, and however advantageous it may be found elsewhere, is quite a marked feature of the older American and Mexican ore-treatment practice, and is, apparently, due to the fact that, in the early days, the side of the hill was always selected as the site for the reduction plant, and full advantage was invariably taken of the use of gravity for transferring pulp from one stage of treatment to another, and any departure from the gravity system has been adopted very slowly and cautiously, but in the more modern American and Mexican plants there is a growing tendency to the use of circuits in milling, especially in tube milling and the use of a circuit necessarily means the re-elevation of some part of the pulp. In connection with this matter the article by Herbert Lang in the July 5 number of the *Minning & Scientific Press*, page 7 is of interest as showing the trend of modern metallurgical practice.

In regard to the double treatment mentioned by Mr. Spicer, I presume he refers to the use of return sand cones, but, as stated above, the use of these cones only involves the loss of about 2 ft. of head on 85 per cent of the total pulp.

Sand Filter Tables.—The cost of operating Caldecott Tables at the City & Suburban G. M. Co., referred to in Mr. Spicer's article, expressed in American currency, is as follows:

	Per Ton of Sand:	Per Ton of Ore:
Power	2.36 Cents.	1.31 Cents.
Maintenance	0.43 "	0.23 "
Labor	0.58 "	0.33 "
Totals	3.37 Cents.	1.87 Cents.

The value of the original sand is 3.52-dwt., and the value of the residue is 0.371-dwt.

It will be seen from the above figures that the metallurgical efficiency of the system is high, and that the cost of power and maintenance is very small, which is due, principally, to the fact that the table revolves at a very slow speed, viz.: one revolution in 3 minutes. In regard to the cleaning of filter cloths, this work is performed in half-an-hour by two natives.

Necessity for Two Tables.—While in all continuous operations it is desirable to be able to cut out a unit periodically still it has been found possible, on the Princess Estate & G. M. Co., to work with one filter table and yet obtain most satisfactory results under proper supervision as can be seen by reference to Mr. H. A. White's contribution to the discussion on the Metallurgy of the Homestake Ore, which appears on p. 55 of *Bulletin No. 100* of the Institution of Mining & Metallurgy.

In Mr. White's article he shows that the sand residue at the Princess Estate, over a period of three months, had an average value of 0.23-dwt., so it can be seen that the use of one table only gave excellent results. Mr. White's article also shows the extremely low value of last drainage solution of 0.05-dwt., and this is due to the fact that, by the Caldecott system, the slime is completely eliminated, resulting in very clean sand.

At the Princess Estate the sand treatment is what is termed "single" treatment, i. e., the extraction of gold from the sand is completed in the vat into which the sand is transferred from the filter table, and the residues are so low that they would not pay for transferring into another vat for the purposes of being given a further treatment.

The advantages claimed for the Caldecott patent continuous sand plant are as follows:

- I. Efficient classification of sand and slime in tailing pulp.
- II. High percentage of slime which can be treated by cyanide at less cost than sand.
- III. Clean, leachable sand charges, permitting free aeration, and facilitating the dissolving of gold, and the washing out of gold with decreased volume of solution.
- IV. Clean sand residue, for sand filling.
- V. Low cost of plant operation and maintenance.
- VI. Lower capital expenditure for plant.
- VII. Dissolving of gold in sand by cyanide solution begins within half-an-hour of ore being crushed in battery.
- VIII. Less volume of gold-bearing solution to precipitate.

The validity of the foregoing claims is evidenced by the fact that at present about 20 per cent of the total Rand tonnage is being treated by this process, and the following is a list of companies using the Caldecott process, together with the respective ore tonnages treated *per month*.

	Tons
(a) Simmer & Jack Proprietary Mines, Ltd.,	70,700
(b) Jupiter G. M. Co., Ltd.,	42,400
(c) Simmer Deep, Ltd.,	63,000
(d) Princess Estate & G. M. Co., Ltd.,	22,800
(e) Geduld Proprietary Mines, Ltd.,	24,000
(f) East Rand Proprietary Mines, Ltd.,	170,000
(g) New Klemfontein G. M. Co., Ltd.,	45,000
(h) City & Suburban G. M. Co., Ltd.,	27,500
Total	474,400

Crushing in Water.—Mr. Spicer writes as though there is nothing in favor of the Rand practice of crushing in water, and while impartial judges readily admit the desirability of crushing ore in cyanide solution where the percentage of recovery by amalgamation is trivial—as in connection with gold telluride, or silver sulphide ores—still, they are equally ready to admit the fallacy of crushing in cyanide solution where a high percentage of the gold can be recovered by amalgamation.

It is certainly unsound, metallurgically, to crush Rand ore in cyanide solution and thus needlessly incur trouble from corroded copper-plates, liability of loss of gold-bearing solution, base bullion due to deposition of dissolved copper and zinc, and the more prolonged and expensive cyanide treatment necessitated by richer tailings. Further, crushing in cyanide solution renders it impossible to obtain accurate screen values, due to the presence of dissolved gold in solution.

The crushing of Rand ore in cyanide solution has been tried on a large scale, and abandoned for the reason stated above, and as a great deal of expense was incurred in converting mills to make them suitable for crushing in cyanide solution, there was, of course, every inducement not to revert to crushing in water, but the troubles experienced were so great that such a change was found to be absolutely necessary.

In America you have the Homestake Co., treating an ore of about the same characteristics as regards amalgamation as Rand ore, and it is interesting to note that the Homestake crushes in water.

Rand Metallurgists are becoming rather used to outside criticism, and an article appearing in the August 9th, 1913 issue of the *South African Mining Journal*, under the heading "Rand Metallurgical Methods," would doubtless interest some of your readers. It reads as follows:

"In two or three recent issues of this journal we have printed extracts from overseas critics of Rand mining and metallurgical methods. The views of gentlemen like Mr. Motherwell, of Australia, and Mr. Spicer, of America, the two other large gold producing continents of the world, have, we understand, attracted much local attention, and it is generally felt that their comments on Rand metallurgical methods are scarcely just.

"Briefly stated, most of these overseas critics, while applauding the general organization and magnitude of operations on the Witwatersrand, find fault with the metallurgical practice of the Main Reef series. One or two of them attack the methods pursued in mills and cyanide works on the ground that they are antiquated and conservative. Other technical visitors who have recorded their impressions appear to find fault with Witwatersrand metallurgists in that they have borrowed ideas from all other mining countries.

"On the face of it these statements seem contradictory to one another. If the argument that our equipments and devices are obsolete is tenable, then the view that the Rand is to be condemned because our technical experts have not been slow to appreciate the value of the ideas of other mining camps cannot very well be supported, for it obviously infers a denial of the first-named statement.

"The essential view which most of these critics seem to lose sight of is that the primary object of mining is the earning of profits for shareholders. The enunciation of this basic economic fact leads one on to a consideration of a number of logical corollaries. In the first place it should be recognised that progress in ore treatment means making more profits from the same grade of ore. It may or may not, however, involve (a) new processes or devices—a 'different' method or device is not necessarily a 'better' one; (b) a reduction in working cost; (c) higher percentage recovery.

"Critics who have machinery or processes to exploit consider that the employment of a new device or a new method is synonymous with progress. Often they do not take best results but those which favor their own purposes. Progress may involve the use of new devices or the employment of new methods, or it may consist in (a) better use of existing appliances; (b) economy through better arrangement of known devices; or (c) better organization of work.

"On the Rand all types of practice are represented, from those of the pre-war epoch to those of 1913, but no individual existing plant represents all known 'best' practice. During the last few years there has been a large accumulation of knowledge requiring to be digested and profitably applied. Local conditions make many devices and methods unsuitable here, as for instance, the large scale of operations precludes the 'small

handy machine,' of limited capacity, just as a 3s. per day Kaffer may be more economic than a mechanical device with an 8s. per day white man.

"Another important consideration is that many devices have been tried locally and have been found wanting, which are claimed to work well elsewhere. The results of the successes only are made public; experience with failures is generally kept quiet for obvious reasons. Then, too, it must not be overlooked that the low average grade of ore and large sums of capital in plants and interests involved on the Rand preclude rash changes being recommended by those who are trustees for shareholders, and gambling merely in the name of progress with shareholders' money is a breach of faith.

"Notwithstanding all this, it must be admitted by all unprejudiced critics that Rand practice is good. At present Rantek is treated with a total residue at 0.3 dwt. at a total cost from shaft to dump of 3s. 2d. per ton. In fact, Rand practice of today represents combined efforts of many able men from every part of the world for a quarter of a century; the combined result of large capital and experimental expenditure. It is essentially a process of evolution and survival of the fittest. Note, too, that there is very little scrapped machinery, representing waste of shareholders' money on these fields.

"Gradual development has usually admitted of utilization of existing plant combined with its extension on more modern lines.

"Another important aspect of progress is this: At any time and on any mine there is an 'economic limit' of extraction or residue beyond which it does not pay to go, since such extra gold costs more than it is worth. Other essential facts to remember are the simple nature of the Rand ore which does not require many appliances in use elsewhere. The large scale of operations makes organization of great importance, and the group system with specialist staff at the disposal of each mine gives these mines advantages single companies cannot afford.

"To sum up, we believe that in crushing practice, in stamp and tube milling and classification with limited supply of water the Rand is ahead of the rest of the world. When an appliance from abroad (e. g., tube mill) which is suited for use here arrives it is promptly adopted and fully utilized. There are probably more tube mills in use on the Rand today than anywhere else in the world."

It occurred to me the article quoted may interest your readers as representing the Rand side of the question.

EDW. L. BATEMAN

Johannesburg, South Africa.

The Western Metallurgical Field National Radium Institute

For over a decade the carnotite deposits of southwestern Colorado and southeastern Utah have been exploited in a desultory way. They have been regarded mainly as possible sources of vanadium and uranium, and up to two or three years ago the ores were mined and treated for those metals exclusively. Recently, however, a new phase has been put on the whole situation by the general recognition of these ores as a source of radium; and the demand for salts of the latter element has lent an added value to the deposits. Not only has this district come into prominence in the eyes of American scientists, but it has gained a world wide reputation, so that foreign capital has sought to purchase the ores mined there, or failing in that, to buy mining claims.

The vast change in the situation has been caused by a realization of the radium value of these ores; and with that realization came the feeling on the part of some of our altruistic scientists that this important resource should be conserved for the benefit of our own country; that the radium-bearing ores should be treated here instead of in Europe, and the radium placed at the disposal of American scientists. The result is the organization of the National Radium Institute, directed by Dr. James Douglas, of Phelps Dodge & Co., New York; Dr. Howard A. Kelley, of Johns Hopkins University, Baltimore.

Dr. C. F. Burnam, associate of Dr. Kelley; Mr. Archibald Douglas, nephew of James Douglas, and Mr. E. J. Maloney, of Wilmington, Del. The institute has provided capital for the purchase of carnotite claims, and the mining and treatment of the ore. A cooperative agreement exists between the institute and the national Bureau of Mines, whereby the latter carries on the technologic study of the ore treatment and supervises for a time the actual process of recovering the three valuable constituents, uranium, vanadium and radium. The institute is a philanthropic body to the extent that its capital is to be expended in the production of radium which will not be sold, but used for scientific purposes.

The radium value of American uranium ores can be best comprehended from the following facts: In 1912 the United States produced 28.8 tons of U_3O_8 , equivalent to 24.4 tons of metallic uranium. All but a few tons of this was shipped abroad. On the basis of the normal equilibrium between radium and uranium, this production represented 9.77 grams of radium chloride or 12.7 grams anhydrous radium bromide. But on the basis that the ratio of radium to uranium in carnotite is about 10 per cent below the equilibrium value, the above uranium production in 1912 contained radium equivalent to 8.8 grams radium chloride or 11.43 grams radium bromide. Contrasted with these figures we have the fact that the Austrian production of radium chloride in 1912 was only 2.23 grams. In making this comparison it must be remembered that the figures from Austria represent radium chloride actually produced, while those for the United States give only the radium content of ore shipped from the country; but allowing a recovery of 85 per cent of the radium in the United States ore, it is seen that our local production was still three times as great as that of Austria, the principal producer.

The reduction plant of the National Radium Institute will be built in Denver, where it can be under the immediate supervision of the officials of the Bureau of Mines who are charged with the technologic work on rare metals. The mill will be adapted to the application of several of the processes which have been evolved in the Bureau's Denver laboratory. None of the details will be made public until the results are officially announced through the Bureau.

With the impetus given to the radium industry by the organization of the institute, and the cooperation of the federal government, we may expect to witness a few disreputable promotions by unscrupulous parties who will prey on the public through promises of fabulous returns on a small investment. Such a development must result in inevitable loss to investors, besides reacting unfairly on a few commercial undertakings of a legitimate character. It would seem to be in order to sound a warning against promiscuous investment in "radium" companies.

Concentration in the Joplin District

Representatives of the Bureau of Mines have recently spent some time studying the methods of mining and concentrating zinc and lead ores in the Joplin district, Missouri. Their observations are contained in *Technical Paper 41*. Joplin concentrating mills have been the subject of much criticism, particularly in regard to their inefficiency in treating the finer portions of the pulp. The jig practice is regarded with more favor, and is generally conceded to give excellent results on the local ores. "In the concentration of the fine material the operators of this district have, until the last few years, been rather behind in their methods, but are beginning to realize the importance of saving the finer mineral particles formerly wasted in the discarded sands. There are still several mills in the district that have no concentrating tables or sand jigs, no attempt being made to avoid the losses mentioned." Even in those mills where tables are used, little care is exercised to get good classification.

The loss in concentrating lead ore is not as great as in the case of zinc. "The average recovery of blende from the ore in milling is about 60 to 65 per cent. When the heavy losses in smelting zinc ores are included, the total loss in the production of zinc from the ore in the mine up to the commercial

product, reaches nearly 50 per cent." Tests were made on the feed and tailings from different mills, with the following results: On ore from "sheet-ground" mines, rougher jig feed contained an average of 3.57 per cent zinc, and the tailings 1.36 per cent. From other than "sheet-ground" mines, rougher jig feed contained an average of 2.79 per cent zinc, and tailings 0.84 per cent. The average table feed in a number of mills was 5.28 per cent zinc, and tailings 1.57 per cent. The losses in different sizes of tailings are shown below:

	Dry weight grams	% of whole	Zinc assay %	Loss %
Oversize to $\frac{3}{8}$ in.	2426	15.225	0.77	12.256
$\frac{3}{8}$ in. to 3 mm.	7579	47.565	0.96	47.608
3 mm. to 1.5 mm.	3817	23.955	0.76	18.946
1.5 mm. to 0.46 mm.	1659	10.412	0.71	7.678
Undersize through 0.46 mm. ..	453	2.843	4.55	13.512

An excuse for the tendency to sacrifice efficiency for capacity in the Joplin mills and mines, is found in the system of royalties which are imposed on lessees and sub-lessees. Royalties to owners of the ground run from 10 to 15 per cent of the gross output of mineral, and this is sometimes increased by the practice of subleasing, to 20 or even 30 per cent. The system results in waste in both mining and milling.

Strike Situation in Michigan

The latest reports from the copper country of Michigan are to the effect that normal conditions are gradually returning. Some demonstration of violence is still made by the strikers, particularly toward men coming into the district in search of work. These violations of the strike injunction have caused the arrest of a large number of strikers. The mine managers have issued a statement through the Copper Country Commercial Club to the effect that striking miners will be allowed to return to work, provided they have not engaged in riots and disturbance. An 8-hour day is to be established for underground work by the first of the coming year. As a result of these overtures work is being resumed and production is increasing. The Calumet & Hecla is employing nearly its normal quota of labor, and shipments of rock to the mills are being made to the extent of 7,000 tons daily. No shortage of labor is anticipated, as many of the iron miners are now available following the closing of some of the iron mines for the winter.

Settling the Fume Question in California

A state investigation of the differences between California smelters and farmers is under way, in accordance with an act of the last legislature which provided an appropriation of \$5,000 for the use of a commission charged with the determination of damage by smelter fumes to stock and vegetation. This commission is composed of Dr. W. F. Snow, secretary of the state board of health, Dr. Charles Keane, state veterinarian, and Dr. J. A. Cook, commissioner of horticulture. The first work of the commission is being done in Shasta county, where a number of smelting plants have been closed.

Dr. J. A. Holmes, director of the Bureau of Mines, visited the affected districts in Shasta county on his recent return from Alaska. It is understood that he regards the situation as more hopeful of settlement than at any time past. General prosperity of the farmers in the smoke belt has diminished their antipathy toward the smelters, and complaints are fewer and less bitter.

Company Reports

Quarterly reports of the activities of the large copper companies for the third quarter of 1913 show no marked changes in conditions compared with the first half of the year. The Utah Copper Co. produced 23,884,467 lb. copper from 2,035,391 tons of ore averaging 1.2450 per cent copper at a cost of 9.68 cents per pound. Crediting the net miscellaneous earnings, the cost would be 8.187 cents. The low grade of the ore treated was responsible for the comparatively higher cost. Both the Magna and Arthur plants are in good condition, and handled an average of a little over 22,000 per day for the quarter.

The Ray Consolidated Copper Co. produced during the third quarter of this year 12,969,120 lb. copper from 575,190 tons of ore averaging 1.72 per cent copper. The average mill recovery was lower and the operating cost higher than for the previous quarter, owing to shortage of water during the summer months, and to some important improvements in the mill. The average cost of copper produced, after allowing for smelter deductions and applying the earnings of the Ray & Gila Valley R. R., was 10.155 cents per lb.

The Chino Copper Co. produced 15,187,003 lb. copper during July, August and September, 1913. The total amount of ore treated was 507,650 dry tons, of an average grade of 2.23 per cent copper. The extraction averaged 66.98 per cent; the average grade of concentrate produced was 14.46 per cent copper. The cost per pound of copper, after allowing for smelter deductions, but without crediting miscellaneous income, was 8.41 cents. If miscellaneous earnings are credited, the cost is only 8.08 cents.

The Nevada Consolidated Copper Company's operations the third quarter of the year resulted in the production of 15,835,563 lb. of copper. The amount of ore treated was 813,153 tons, 93 per cent of which was from the pits and 7 per cent from underground workings. The grade of the ore was 1.53 per cent copper, which is lower than that for last quarter. The cost per pound of copper produced was 10.09 cents, which is much higher than that for the preceding quarter, viz., 8.95 cents. The grade of concentrate produced is decreasing on account of the character of the ore now being milled. Experiments are being carried on looking to a cheaper and better method of treating the low-grade slimes which are produced in milling, and which reduce the grade of concentrate.

The annual report of **Camp Bird, Ltd.**, for the fiscal year ended June 30, 1913, shows that the mine at Ouray, Col., is able to supply ore for only 25 out of the 60 stamps. The gross amount of ore treated was 30,012 dry tons. The content of the ore treated averaged as follows: Gold, 0.958 oz. per ton; silver 3.83 oz. per ton; lead, 1.42 per cent; copper 0.21 per cent. The saving of the gold amounted to 93.79 per cent. Of the total value extracted, there was obtained by amalgamation 49.28 per cent; by concentration 43.91 per cent; and by cyanidation 6.81 per cent. In addition to the above tonnage of ore treated, the cyanide plant handled also 1745 tons of impounded tailing, from which it is estimated that \$4,937 was recovered.

The Camp Bird company's operations at the Santa Gertrudis property in Pachuca, Mex., resulted in bringing up the capacity of the new mill to 25,000 tons per month. During the fiscal year there were treated 263,554 dry tons of ore of an average value of \$13.08 per ton. The recovery amounted to \$11.70 per ton, or 89.41 per cent. The old Guadalupe mill was not run during the year; but 761 flasks, or 28.6 tons of quicksilver were recovered from the old patio, yielding a net profit of \$17,335.

The Non-Ferrous Metal Market

Dullness has characterized the non-ferrous metal market during the period since our last report. Most of the prices have suffered a decline, and in cases where business has been good, the competition has been keen and prices have suffered. The dullness in some metal industries in this country, and the weakness of the London standard metal market have both been reflected in the domestic market.

Copper.—This market has been decidedly weak, and competition for orders has been keen. This has resulted in open cutting of prices, until unusual concessions were made. Business in Lake copper has fallen off until quotations are strictly nominal at 16@17 cents. Electrolytic is quoted at 15.25@15.30 cents.

Tin.—Prices have declined in the face of a fair statistical position of this metal. Considerable business has been transacted in a retail way, but the general tone of the market has not been strong. November tin in the domestic market is quoted at 39½ cents.

Lead.—This market has been fairly active, with sellers competing for such business as is in sight. The last available quotations are 4.15@4.20 cents, St. Louis, and 4.30@4.32 cents, New York.

Spelter.—This market has declined during the month. Business has been light and producers have made concessions to get orders from consumers. The last quotations are 5.05@5.10 cents, St. Louis, and 5.30@5.35 cents, New York.

Other Metals.—The aluminium market has shown dullness and little business has been transacted. The New York price for No. 1 ingots is 19¼@19¼ cents. Prices for antimony range from 6.50 to 7.75 cents for various brands, with a fair business in sight. The quicksilver market is steady with no material change in prices. New York quotes \$38.50 per flask of 75 lb. The domestic price at San Francisco is the same as for New York, with special prices for export trade.

The Iron and Steel Market

November has been a month of falling prices and sharply decreasing productions, the flow of orders having undergone a great diminution.

At the close of October the steel finishing mills were operating at an average rate of 80 per cent of full capacity; at the close of November operations were at an average of not more than 60 per cent, and prospects were that the rate would drop to 50 per cent before the end of December.

The report of a month ago suggested that enough had been developed to indicate that the tariff of October 4, 1913, was not directly the chief influence in the price decline. It is now clearly established that the tariff has had practically nothing to do with iron and steel prices, which have been falling because there was insufficient demand and therefore sharp competition for orders. It may be that the tariff is in part responsible for the general slowing down in business which has extended to steel, but that is really an altogether different factor.

The iron and steel market is passing through one of its periodic liquidations, and the nature and extent of such movements is well understood. The readjustment starts by orders becoming less than shipments, until eventually the mills must seek new business in order to maintain full operation. This results in price cutting and the open market declines. The existing contracts are on a basis below the open market, and when the market declines to or below the level of a given contract the buyer refuses to take any more deliveries under it without a price readjustment. This forces the mills to seek more new tonnage and then, unless there is artificial control of prices, the market declines rapidly. In the whole history of the American steel trade there is no case on record of prices being artificially maintained for the entire time from one period of activity to another. What has occurred at times has been the artificial maintenance of prices for a period, during which contracts would thus be kept alive and mills would be content to operate upon such business as was afforded at the controlled price level.

The most variable element in these periodic liquidations and readjustments is the time element. In that which followed the panic of October, 1907, the period of controlled prices lasted about 15 months, the actual liquidation and readjustment occurring during a few months following. The next time, in 1910-11, the period of control was less clearly separated from the period of active readjustment, the one merging into the other.

The present readjustment is sharply distinguished from its predecessors by there being no artificial control. The market is proceeding to the inevitable low point, not far from the cost of production, with relative rapidity. According to theories formerly promulgated in the trade, the element of time rather than of price reduction is the controlling one in forcing a revival in demand, but those theories have fewer adherents than formerly. If such a theory is correct, the price decline will not soon be followed by recovery, with forward buying and advancing prices, but there are many who urge that the

governing element is not time, but price, and that if a low point is quickly reached, the recovery will be correspondingly early. An excellent opportunity is about to be afforded to test the validity of the latter theory.

The total decline in finished steel prices now averages about \$4 per net ton, from the high point of last winter and early spring. The decline thus far represents about 60 per cent of the advance which had occurred from the previous low point, which fell late in 1911 and was the lowest average level in the finished steel market since 1898, when prices reflected the condition of there having been an industrial depression of five years' duration, with wages and supplies very low.

Pig Iron.

Although pig iron experienced an extensive decline during the first seven months of the year, the decline proved insufficient to meet present conditions, and the recovery which began in August came to an end about the middle of October, since when the market has been falling rather rapidly. During the first seven months of the year the average decline was about \$3 per ton, while August and September showed an average advance of 30 to 35 cents. Since the middle of October there has been a further decline of nearly 75 cents, so that the market is now about \$3.35 a ton below the level at the beginning of the year, which was substantially the top level in the advancing movement of 1912.

The pig iron market is almost absolutely stagnant as to inquiries and orders, representing stagnation rather than a normal dullness. In the summer there was a moderate volume of forward buying, due to the temporary stiffening in the market, and as deliveries are still coming in, and no forward buying is contemplated, orders and inquiries are far below normal at the moment. The actual consumption appears to be fairly large, although there has undoubtedly been some decrease. The current market stands as follows, with most of the quoted prices representing the ordinary asking figure and being susceptible of shading in the event of interesting orders being negotiated: No. 2 foundry, f.o.b., Birmingham, \$10.50; No. 2X, delivered, Philadelphia, \$15.50; No. 2X, f.o.b. Buffalo furnaces, \$13.50; No. 2 foundry, delivered, Cleveland, \$14.25; No. 2 foundry, f.o.b. Chicago furnaces \$14.75; at valley furnaces (90 cents higher delivered Pittsburgh) Bessemer, \$15.00; basic, \$13.00; No. 2 foundry, \$13.50; gray forge, \$13.25; malleable, \$13.75. Ferromanganese is \$50 for English and \$49.50 for German, f.o.b. Baltimore.

Steel.

As was the case in October, the unfinished steel market has been made not by important transactions in the open market, but by the readjustment of contract prices, agreed to by the sellers rather than allow the consumers to make inquiry in the market. On this basis we quote the market roundly at \$20.50 for billets, \$21.50 for sheet bars and \$26 for rods, f.o.b. maker's mill, Pittsburgh or Youngstown.

Finished Steel.

Since last report bars, plates, shapes and sheets have declined from \$1 to \$2 a ton. Nearly all products are weak, and present quoted prices represent what would be done on ordinary small lots, while in the case of attractive orders the prices would probably be shaded quite materially. Quotations are f.o.b. Pittsburgh unless otherwise stated:

Plates, tank quality—1.25 cents.
Shades, 1.30 cents.
Steel bars, 1.25 cents, base.
Iron bars, 1.45 cents, Pittsburgh; delivered Philadelphia, 1.27 1/2 cents; f.o.b. Chicago, 1.15 cents.
Wire nails, \$1.60, base; plain wire, 1.30 cents, base.
Sheets, blue annealed, 10 gage, 1.50 cents; black, 28 gage, 1.95 cents; galvanized, 28 gage, 2.95 cents; painted corrugated, 28 gage, 2.15 cents; galvanized corrugated, 28 gage, 3.00 cents.
Merchant steel pipe, 3 1/2 to 3 in., 80 per cent off list.
Steel boiler tubes, 3 1/2 to 1 1/2 in., 60 per cent off list.

Standard railroad spikes, 1.35 cents, Pittsburgh; 1.60 cents, Chicago.

Button head structural rivets, 1.85 cents.

Cone head boiler rivets, 1.95 cents.

Cold roller shafting, 62 per cent off list.

Carborundum Litigation

The United States Supreme Court has denied the application of the Carborundum Company for a writ of a *certiorari* in the case of the Electric Smelting & Aluminum Company versus the Carborundum Company, which had been formerly decided in favor of the former company by the United States Circuit Court of Appeals for the Third Circuit. This decision was given in some detail in our issue of April, 1913, page 203. This terminates this patent law suit of many, many years.

Refractories Manufacturers' Association

The recent Philadelphia meeting of the Refractories Manufacturers' Association was a very enthusiastic affair and well attended, considering the fact that all the members had to come from a considerable distance.

This Association is doing great work in the matter of standardization. Heretofore, the brick made was almost any size that the manufacturer wanted to make them. Now all the forty-five concerns which belong to the Association, manufacture the same sizes, both in silica and clay.

Colonel H. D. Savage, of the Ashland Fire Brick Company, of Ashland, Ky., is the president of the Association; Mr. J. E. Lewis, of the Harbison-Walker Refractories Company, of Pittsburgh, Pa., is the vice-president; Mr. J. H. Cavender, of the American Refractories Co., of Chicago, is secretary-treasurer.

The executive committee consists of Mr. J. L. Green, of the Laclede-Christy Clay Products Company, of St. Louis, Mo.; Mr. C. S. Reed, of the Chicago Retort & Fire Brick Company, Chicago, Ill.; Mr. E. S. Hitchens, of the General Refractories Company, Olive Hill, Ky.; Mr. D. D. Davis, of the Davis Fire Brick Company, Oak Hill, Ohio; Mr. T. E. Thomas, of the Niles Fire Brick Company, Niles, Ohio; Mr. J. H. McFeely, of the McFeely Brick Company, Pittsburgh, Pa.; Mr. George H. Diack, of the Queens Run Fire Brick Company, Lock Haven, Pa.; and Mr. C. H. Claiborne, of the Union Mining Company, Mt. Savage, Md.

The American Blower Company, of Detroit, Mich., have purchased the entire Air Washer interests, including patent rights, of the McCreery Engineering Company, formerly of Toledo, Ohio, and later of Detroit, Mich.

Gas Engines.—The Mesta Machine Company of Pittsburgh, Pa., has issued illustrated Bulletin 1 on "Mesta Gas Engines," built in sizes from 350 hp upwards for any class of service and for any fuel gas, such as producer gas, blast-furnace gas, coke-oven gas or natural gas.

Alunite has been discovered in Arizona and Nevada and the deposits have been investigated by the Geological Survey. No prediction of the value of the deposits is given, but they are considered as warranting further prospecting and development.

Electrolytic Coating of Iron and Steel with Lead.—The London *Electrician* of September 26 states that Lead, Ltd., has acquired a process devised by Mr. Sherard Cowper-Coles for the coating of iron and steel with lead electrolytically. It is stated that by this process lead can be economically deposited up to a thickness 1/8 in., and that the method is suitable for protecting iron and steel from corrosion, and for the lining of pipes, tubes and chemical vessels for containing corrosive liquids. Steel plates coated by this process are much smoother than the ordinary tinned plate, and there is no reduction in tensile strength or ductility. The process can also be used for the coating of earthenware and wood and for the protection of ornamental ironwork. The cost is stated to be low.

Present Status and Future Prospects of the Metallurgy of Zinc.

A symposium before the New York Sections of the American Electrochemical Society and the American Institute of Mining Engineers.

The joint meeting of the New York sections of the American Institute of Mining Engineers and of the American Electrochemical Society held on the evening of November 20 in the Engineering Building was well attended, there being some 200 members present, and the carefully arranged program deserved the interest which was shown.

There were three papers. Mr. George C. Stone presented a very careful review of the improvements which have been attained in the metallurgy of zinc with the time-honored retort process; Mr. W. R. Ingalls discussed the possibilities and limitations of electric zinc smelting, while Dr. J. W. Richards dealt with electrolytic zinc, having stepped in at the last minute to cover this phase of zinc electrometallurgy, as a promised paper by Dr. Victor Englehardt, of Germany, on this subject, had not arrived in time.

Mr. Lawrence Addicks, chairman of the New York Section of the American Electrochemical Society, presided during the presentation of the three papers. Mr. L. D. Hinton, chairman of the New York Section of the American Institute of Mining Engineers, presided during the discussion.

Improvements in the Retort Process

Mr. George C. Stone, of the New Jersey Zinc Company, in reviewing in a most interesting manner the development of the retort process, emphasized that while no spectacular or radical changes had been worked in the metallurgy of zinc as in the metallurgy of iron and copper by the introduction of converters, yet the progress had been as great with zinc as with the other common metals.

Three essential facts limit the possible apparatus for the production of zinc.

First, the temperature at which zinc is reduced by carbon is much above the volatilizing point of the metal, and it is, therefore, always produced as a vapor.

Second, to obtain a merchantable product the vapor must be condensed at a temperature well above the melting point of the metals. The temperature of the condenser must be close to the boiling point of zinc (920°) nearest to the retort and very much lower at the other end, because the condensation temperature depends on the concentration of zinc in the vapor and is lower the more dilute the vapor. The vapors are rich in zinc when they enter the retort and low in zinc when they leave it.

Third, the reaction by which zinc oxide and carbon monoxide are changed into metallic zinc and carbon dioxide is reversible. The reduction of zinc proceeds only until there is a certain ratio of carbon dioxide of zinc in the atmosphere and then stops, and in order to effect complete reduction there must be an excess of carbon always present.

These three essential facts control the type of apparatus to be used. It must be a closed chamber. It must be heated externally because the gas from the heating fuel must not come in contact with the charge. As the mixture of ore and coal forming the charge is a poor conductor of heat the size of the chamber containing it must be small in at least one direction or the heating will be too slow. The charge must, at all stages, contain a large excess of carbon to reduce at once all CO_2 formed. The condensers should be as close to the charge as possible and as small as will do the work in order to make it possible to maintain the proper heat gradient in them. The condensers must be kept within proper limits of temperature, diminishing towards the exit.

Mr. Stone explained in some detail why he questioned the statement commonly made that his spelter furnace is extravagant in heating, and then summed up the progress made in recent years as follows:

Forty or fifty years ago the typical Belgian furnace contained from fifty to ninety retorts of about 0.8 of a cubic ft.

capacity each. At the present time the furnace used in the natural gas field contain from 500 to 700 retorts of about 1.4 ft. each, an increase in capacity of about 1300 per cent. Furnaces containing 1008 retorts are in operation doing satisfactory work; this is an increase of about 2400 per cent in size and capacity. The increase in capacity of the spelter furnaces is, therefore, about the same as that of the modern coke furnace making pig-iron over the old charcoal furnaces.

More increase in size, however, does not amount to much unless accompanied by increased efficiency. In this respect the zinc furnaces show well. In the iron furnaces there has been no improvement in the proportion of metal recovered the old furnaces having given almost theoretical results. With zinc the improvement has been very marked. In 1844, at the Vieille Montagne works, the recovery was about 62 per cent, and even as late as 1880 a recovery of 75 per cent was considered good work. To-day 87 per cent is only fair, and many works can show runs of long periods averaging 90 per cent and over.

The fuel consumption has been decreased largely mainly by the adoption of regenerative gas-firing. Even in direct-fired furnaces the decrease is large. Comparing furnaces only using the same kind of coal, where the hand-fired furnaces twenty years ago required 3, $3\frac{1}{2}$ and even 4 tons of coal per ton of ore, well-equipped gas furnaces using the same coal, and working the same class of ore, now take only $1\frac{1}{2}$ to 184 tons.

Labor has also been largely reduced. The old Belgian furnaces required five days' labor per charge, two men working twenty-four hours continuously, and an extra helper on the day shift only. As these furnaces worked only about a ton of ore they required five days' labor per ton. At present all twenty-four-hour work is done away with, and not over 12 days' labor is required per ton.

In 1844 fifteen to twenty days was the usual life of a retort. Now good retorts made in hydraulic processes last from thirty to forty days, and at that more are replaced because they are filled up than because they leak.

Mr. Stone believed progress would continue to be made rather on the lines of improvement in the present retort process than by the adoption of radically different ones. He finally discussed briefly three different new types of processes which have been proposed, namely, the wet, electrolytic, and electro-thermic.

One main reason for the lack of success of the wet process is the difficulty of dissolving the zinc. Zinc blende usually contains a large amount of iron and when roasted the zinc and iron combine to form zinc ferrite, which is practically insoluble in either acids or alkalis under economic conditions. The purification of the solution is also expensive and rather troublesome.

Electrolytic zinc of excellent quality has been made by one firm for several years, but they have never increased the original plant. It is generally believed that this plant is only possible in connection with other operations carried on by them. Besides other difficulties of wet processes Mr. Stone said that electrolytic processes have the trouble of large power requirements.

Electric smelting of zinc ores is seriously limited in this country by the fact that in most mining districts electric power cost is high. Further, there are condensation troubles and other difficulties.

Mr. Stone concluded that the present types of furnaces can be much improved. Fuel can be saved by better design and proportions of regenerators; better arrangement of the ports and decreasing radiation losses by more thorough lagging of the parts of the furnaces from which radiation is purely wasteful. A determination of the best size and shape of retort and condenser for different ores, and the choice of such as suit the particular ore to be worked instead of using one size and arrangement for all, will give improved results in recovery and output. A careful study and application of the principles on which ores should be mixed offer a wide field for improvement.

Electric Zinc Smelting

Mr. W. R. Ingalls' discussion of the possibilities of the electric furnace for zinc smelting was very careful and conservative. Mr. Ingalls said he was neither a pessimist nor an optimist with respect to electric zinc smelting, but an agnostic. He agreed that electric zinc smelting was no longer in the laboratory stage and referred to some enlargements now being made in the electric zinc furnace plants of Norway. He thought reckless talk had held back the development of electric zinc smelting.

At considerable length Mr. Ingalls discussed the well-known condensation troubles, blue powder being obtained instead of spelter.

The fire-smelting process is a two-stage intermittent process of heating up and distillation. The electric smelting process is intended as one continuous operation. This distinction causes important differences.

The condensation troubles originate at least partly in the furnace itself. Mr. Ingalls distinguished between two kinds of blue powder—meltable and non-meltable. The fact that the condenser is too cold may account in part for meltable blue powder, while non-meltable blue powder is due to impurities in the vapor to be condensed—if it contains carbon dioxide, volatilized lead, dust and other impurities. Oxidized lead is often found in large extent in blue powder.

As to the different electric furnace designs for zinc smelting, Mr. Ingalls could not see any great difference between them. He thought there was not much in electric zinc smelting at present beyond visions and ideas. There were few figures and the subject was attracting attention chiefly on account of its mystery.

Mr. Ingalls then compared the different items of cost of electric smelting and retort smelting. But few of the items in question were known besides the cost of electric power as against coal. He thought the consumption of 1200 kw hours per ton of ore to be plausible though hardly yet an accomplished fact. As to the comparative cost of retorts and electrodes, he maintained his former statement that nobody can yet say which will be the weaker. If the cost sheet of the Trollhättan plant was refigured by inserting American for Scandinavian prices the cost of the application of the process would be found to be absolutely prohibitive.

Mr. Ingalls concluded that electric zinc smelting was still very much in its infancy and that at the present time all we could say was that we did not know.

Electrolytic Zinc

Dr. Joseph W. Richards then discussed electrolytic zinc. In order to show the variety of processes which have been devised in this field he gave a brief outline of the contents of Guenther's German book on the subject. He then took up recent developments with reference to a lecture by Dr. Victor Engelhardt which was published in abstract in our January issue, 1913 (Volume XI, page 43).

While Dr. Engelhardt considers the mere refining of commercial zinc analogous to copper refining, to be economically absurd, Dr. Richards thought it might be possible, under favorable conditions. It would, of course, be necessary to keep the solution clear. It can be kept free from iron by bubbling air through it, whereupon it should be filtered clear. Such an electrolytic refining process might be particularly advantageous if carried out for electroplating purposes.

Engelhardt had stated with regard to direct processes using soluble anodes containing zinc that many attempts had been made since 1880 to utilize the energy due to the solution of the anode and to prepare anodes containing various zinc compounds mixed with coke or coal powder with and without pressure with various binding materials, etc. All such processes have proven a failure. Dr. Richards thought, however, that packing the ore around an insoluble anode (instead of incorporating it in it) was a possibility. He then commented briefly on Engelhardt's description of the latest Siemens & Halske process with insoluble anodes and finally discussed broadly the scheme of sulphate roasting.

Dr. Richards thought that sulphate roasting was essentially a misconception. One gets a dilute solution of sulphuric acid which must be dispensed with after all the zinc is gotten out, and the sulphuric acid produced is a source of trouble.

The correct way to work is, according to Dr. Richards, to roast the ore to oxide as nearly as practical. It is then leached with sulphuric acid and electrolyzed in a double compartment electrolytic cell, zinc being deposited on the cathode and sulphuric acid is then used for leaching another quantity of roasted oxide ore whereby the acid is neutralized. The zinc sulphate thereby obtained is again electrolyzed, etc. In this process, therefore, the sulphuric acid passes continuously through this cycle. It is formed in the electrolysis, neutralized in leaching, again formed in electrolysis, etc. The first supply of sulphuric acid is obtained from the roast furnace gases. This principle is used in a plant at Butte, Montana.

Discussion

Dr. C. F. Chandler opened the discussion with a characteristic little speech. When the chairman, Mr. Huntoon, called him up Dr. Chandler said: "My old pupil gets even with me. He knows I am not prepared, and that is why he calls me up." He then spoke briefly on two occasions in his career when he got into connection with zinc—the first time, 50 years ago, in a poison-murder case, where he had trouble to get zinc free from arsenic, and the second time, later on, in a lawsuit on electrolytic zinc.

Mr. Woolsey McA. Johnson was then asked to give a review of his work on electric zinc smelting at Hartford, Conn. With respect to Mr. Ingalls' statement that a power consumption of 1200 kw hours per ton of ore was plausible and pretty good, Mr. Johnson said this would not be the minimum, but rather the maximum consumption he would consider acceptable. With a small furnace running with 25 per cent efficiency he had usually an energy consumption of 1400 kw hours per ton of charge, but he had also done it at 900 or 1000 kw hours. With a larger furnace he was sure to get a much higher efficiency and reduce the energy consumption correspondingly.

As to the electrode consumption this was $6\frac{1}{2}$ to 7 pounds of electrode carbon per 1000 kw hours, and he figured the cost of electrodes was considerably less than the corresponding expense for zinc retorts.

Mr. Johnson thought that the field of the electric furnace was not so much for plain zinc ores as for complex sulphide ores and by-product recovery must be considered if we want to see what electric zinc smelting can do and will do.

About roasting, Mr. Ingalls had expressed the opinion that the cost would be about the same for electric smelting and for the retort process. But Mr. Johnson pointed out that with the electric furnace it is by no means necessary to dead-roast the ore. Roasting down to 4 per cent of sulphur is about right for the electric zinc furnace. Sometimes in the Hartford plant they add green ore to get a matte. They do not want it to be too metallic and, therefore, add sulphur. This point should make a big difference in the cost of roasting for electric smelting and retort smelting. There is a vast metallurgical difference between a roast to half of 1 per cent and a roast to 4 or 6 per cent, both with respect to capital cost of the roasting furnace and with respect to the cost of roasting. Nine-tenths of all roasting troubles is caused by removing the last 40 or 50 lbs. of sulphur. This point also has a distinct bearing on the sulphuric acid plant for the reason that a richer gas is made. Mr. Johnson said that they are now building a three-ton furnace at Hartford, and he would be glad to show it to visitors. "Zinc comes from Missouri, and you have to show people."

Mr. Alfred H. Cowles spoke of some experiments made by him many years ago on electrolytic zinc. One difficulty which he encountered was the high resistance of the zinc sulphate solution; this was overcome by putting the electrodes close together by the use of two drums. The zinc deposited on the wires could be easily scraped off. The process used was cyclic with respect to sulphuric acid as described in Dr. Richards' paper.

The chief difficulty which was encountered was the melting

together of the finely divided zinc deposit. They tried for half a year all kinds of fluxes, but were unsuccessful until they found that it could be quite easily done with zinc chloride. It would be worth while to try this in the blue powder difficulty. The use of zinc chloride is not patented. Mr. Cowles pointed out that in comparison with aluminium production about one-seventh of the amount of energy is required to deposit a pound of zinc as is necessary to deposit a pound of aluminium.

Mr. H. A. Wentworth asked whether any unmeltable blue powder was formed at the higher temperature before the fumes reached the condenser at all. Mr. Ingalls said that he had no evidence to that effect and thought that the blue powder was formed in the condenser.

Dr. J. W. Richards referred to Mr. Ingalls' statement that there was no great difference between different electric zinc furnaces. While this might appear to be so at first glance, Dr. Richards insisted that there was a very great and important difference if the matter was investigated more in detail and the differences in the furnace designs had a great deal to do with the condensation problem. In the DeLaval type of furnace the fumes acted as resistor, and were thereby overheated, which increased the condensation difficulties considerably. On the other hand, a furnace using a buried arc or smothered arc was better, but it had zones of high temperature, and within these zones there was liability of evaporation of materials like silica or silicon which would later cause condensation troubles. But, in the whole, this type of furnace had certain advantages over the type in which the fumes acted as resistor. Finally, in resistance furnaces of the Clerc type or the Queneau type the conditions seemed to be quite different and probably more favorable.

Mr. E. G. Spilsbury emphasized that zinc is produced in the retort furnace not so much by contact with carbon monoxide gas, but by actual contact with elemental carbon itself. He recited some unsuccessful experiments which he had made some years ago on the use of carbon monoxide gas as reducing agent.

Mr. Woolsey McA. Johnson said that they had overcome their condensation troubles largely. Their average condensation factor (the ratio of weight of zinc in spelter produced to the sum of weight of zinc in flue dust plus ladle skimmings plus blue powder plus spelter) at Hartford was 75 per cent. They produced 1 to 5 per cent of blue powder and 15 to 20 per cent of flue dust. He emphasized that a sharp distinction should be made between blue powder and flue dust. One form of blue powder was particularly troublesome, namely, that due to the reaction between zinc and SO_2 or sulphur from the matte, and referred to his Canadian patent allowed February, 1913, in which a metallurgical description of "sulfidized chemical" blue powder and ways of preventing its formation is clearly given.

After a vote of thanks, proposed by Professor Kemp and seconded by Mr. Stoughton, to the speakers of the evening and to the Committee of Arrangements, the meeting adjourned. It was followed by an enjoyable informal social gathering in the rooms of the Institute of Mining Engineers.

The Wellcome Photographic Exposure Record and Diary for 1914 has just been issued by Burroughs, Wellcome & Company, of London and New York. (Bound in red cloth, price 50 cents.) As in former editions, a very large amount of useful information is condensed in this little and neat pocket guide to photography. Exposure, development, toning, fixing, printing, the various processes of production in warm tones and colors, and the methods of dealing with errors of technique are explained, particular attention being directed in the 1914 edition to the green and blue toning, and the production of various colors by development and other methods.

The cost of copper produced by the Isle Royale company in 1912 was 11.89 cents per lb. The average yield from the ore was 15.4 lb. refined copper per ton. The ratio of concentration at the mill was 93 tons of ore into one ton of mineral which yielded 71.43 per cent fine copper.

The Passivity of Metals

A Symposium Before the British Faraday Society

For the meeting of the Faraday Society, held on November 12, 1913, in London, a symposium of highly interesting papers on the subject of the passivity of metals had been arranged.

We herewith give abstracts of the papers presented. An account of the general discussion which followed will be given in the letter of our London correspondent in our next issue.

The following abstracts are taken from advance copies of the papers kindly sent to us by the secretary of the Faraday Society, S. F. Spiers, 82 Victoria St., London, S. W.

Introduction Into the Passivity of Metals

A paper forming a general introduction to the discussion had been prepared by Dr. George Senter, who pointed out the marked divergence of opinion which exists with regard to the interpretation or explanation of the experimental results on the passivity of metals.

The more important phenomena which have to be accounted for on any theory of passivity are as follows: The passive state is best known in connection with iron, cobalt, and nickel, but most, if not all the metals may be made to assume this state under suitable conditions. Passivity may be induced by immersing the metal in an oxidizing agent such as nitric or chromic acid, and in solutions of alkali hydroxides; also by anodic polarization in a great variety of electrolytes.

When the force producing passivity is removed the metal as a rule returns to the active state, slowly at first and then more rapidly, in some cases almost instantaneously. When the surface of a metal in the passive condition is scratched or rubbed, it regains its activity at once. Treatment with acids, cathodic polarization with hydrogen, and raising the temperature all favor return to the active condition.

A metal becomes much more positive (more "noble") when it changes from the active to the passive condition, and does not dissolve when the equilibrium potential is reached. On further increasing the anodic potential the metal may dissolve as a more highly oxidizing iron (e. g. chromium), or the current may be employed mainly or entirely in discharging anions. The rise of potential on anodic polarization is usually slow at first, and then there is a sudden rise of considerable amount, the metal almost becoming insoluble.

The nature of the anion has considerable influence. Thus the ions of oxidizing acids, such as NO_3 and ClO_4 , favor passivity, while Cl and Br have the contrary effect.

Three theories of passivity have obtained a greater or less degree of recognition:

(1) The *oxide-film theory* of Faraday. (*Phil. Mag.*, 1836, vol. 9, p. 53.)

(2) The *valency theory* of Krüger-Finkelstein (*Zeit. Phys. Chem.*, 1902, vol. 39, p. 104) and W. J. Müller (*Ibid.*, 1904, vol. 48, p. 577); according to this theory passivity phenomena are due to the change of a metal to a "nobler" modification.

(3) The *reaction velocity theory* of LeBlanc (*cit. f. Elektrochem.*, 1900, vol. 6, p. 472, and 1905, vol. 11, p. 9). In its most general form this theory states that passivity phenomena are due to slow rate of change (electrochemical or purely chemical) at the anode. In this form the reaction velocity theory is little more than a statement of the facts, and several special hypotheses have been put forward as regards the mechanism of the retardation. Thus we have:

(a) The *oxygen charge hypothesis* of Fredenhagen (*Zeit. Phys. Chem.*, 1903, vol. 43, p. 1; 1908, vol. 63, p. 1) and Muthmann and Fraenkenberger (*Sitzungsber. der Kgl. Bayerischen Akad.*, 1904, vol. 34, p. 201). The cause of passivity is to be sought in the slow rate of reaction between the anode and the oxygen liberated there, with the result that the anode becomes charged with the gas, or, alternatively, a metal-oxygen alloy is formed.

(b) The *anion discharge hypothesis*, according to which the main change at the anode is not the formation of metallic ions but the discharge of anions; the slow reaction of the dis-

charged anions with the metal produces passivity. The views of Sackur are based on the assumption of a primary anion discharge.

(c) The *hydrogen activation hypothesis* of Foerster (*Abhandlungen der Bunsen Gesellschaft*, 1909, No. 2), and of Schmidt (*Zeitsch. Physikal. Chem.*, 1911, vol. 77, p. 513), according to which the normal condition of iron is passive and it only becomes active under the influence of a catalyst. According to Foerster the catalyst is hydrogen; according to Grave (*Zeit. Phys. Chem.*, 1911, vol. 77, p. 513), a pupil of Schmidt, it is hydrogen ions.

(d) The *retarded ion hydration hypothesis* of LeBlanc (*Lehrbuch der Electrochemie*, 5th edition, p. 285).

Each of these different theories is briefly considered and discussed by the author. He finally concludes that in all probability it will eventually be found that no one theory will serve to account for all the phenomena of passivity.

Thus both LeBlanc and Foerster, although in the main favoring other explanations, recognize that the formation of insoluble films on the electrodes plays a part in some cases of passivity.

The hydrogen activation theory, which is perhaps most in favor at the moment, is undoubtedly valid to some extent, as the effect of nascent hydrogen in restoring the activity of passive metals is beyond dispute.

Whether iron entirely free from hydrogen is necessarily passive, as Foerster and Schmidt assume, would, however, appear to be still open to discussion. In particular, the activating influence of the free halogens (e. g., iodine in potassium iodide) is difficult to reconcile with this view, as it would appear that the metal in the absence of hydrogen is capable under these circumstances of sending charged ions freely into solution, which is contrary to the theory.

On the other hand, it should not be difficult to devise a plausible explanation of the *sudden* changes from the active to the passive state and vice versa on the basis of the hydrogen activation theory, whereas these changes are difficult to reconcile with any of the other theories. It is not easy to understand, for instance, how such sudden changes could occur on the basis of the ion hydration hypothesis favored by LeBlanc.

Anodic and Cathodic Retardation Phenomena

A paper by Dr. G. Grube, of the Institute of Technology of Dresden, Germany, discusses anodic and cathodic retardation phenomena and their bearing upon the theory of passivity.

There used to be a general tendency to assume that both the formation of metallic ions in the anodic dissolution of a metal and the transition from the ionic into the metallic state in cathodic deposition were processes which took place at practically infinite velocity. It was the merit of M. LeBlanc to have pointed out that, with certain metals, the passivity finds its explanation in the circumstance that these metals emit their ions of lowest valency into the solution at a greatly restricted velocity.

That the fundamental view of the theory of passivity of LeBlanc is correct, that it to say, that both anodic dissolution of a metal and cathodic deposition of a metal may be retarded phenomena, has within recent years been established by many experimental researches.

The author deals especially with the second part of Foerster's passivity theory, according to which the small reaction velocity marking the anodic dissolution and the cathodic deposition of passive metals does not alone suffice to focus the whole domain of activation and passivation, and that we are hence forced to assume a co-operation of positive and negative catalysts on the electrode.

The following important question thus suggests itself as concerning the problem of passivity: In how far is the velocity of reaction, and hence the variation of the potential in anodic and cathodic processes, influenced catalytically by the occurrence of small quantities of foreign substances on the surface of the electrode?

The author gives a review of a series of experimental re-

searches carried out by Foerster and his pupils during the last years.

In reviewing their results the author accentuates the following main points:

1. The cathodic deposition of the metals—in itself a slowly progressing reaction—can still more be retarded to a considerable extent by the simultaneous deposition of zinc or of hydrogen. The cause of these retardations is probably to be found in a change of the surface of the electrode, the change being characterized by the formation of unstable intermediate products of higher electrolytic solution pressure than the pure metals possess.

2. The circumstance that cathodic processes can materially be retarded by the occurrence of small quantities of foreign substances on the electrode lends essential support to the assumption that anodic processes may likewise be retarded by the formation of oxygen alloys on the anodes. In many cases, therefore, the cause of the anodic passivity will not be a mechanical closure (covering up) of the electrode surface by an oxide film, but the formation of an alloy consisting of the material of the anode surface and of oxygen, this alloy having a lower electrolytic solution pressure than the pure metal.

3. In the electrolytic oxidation of ferropotassium cyanide to ferri-potassium cyanide a strong polarization was always observed when the electrodes were surrounded by films of ferrocyanide or of oxides. These strong polarizations were not found with platinum, gold, nickel, cobalt, and copper in alkaline solutions. It may, therefore, be concluded that, in this latter case, the passivity is caused, not by an oxide film, but by an oxygen alloy on the anode.

4. The retardations observed in the anodic discharge of the halogens are likewise to be ascribed to alloys of the platinum anodes with oxygen.

5. The anodic evolution of oxygen shows what we have to understand by the passivating oxygen alloys. We may imagine that the oxygen combines to a small extent with the anode material to form oxides, which in their turn form, together with the unchanged electrode material, solid solutions, and therefore, one-phase systems of continuously variable composition and likewise variable electrolytic solution pressure. The oxide film theory contrasts with this view, since according to that theory the oxide formed would mechanically cover and shut off the electrode surface so that oxide film and electrode would form a two-phase system. The oxygen-alloy theory of the passivity thus seeks the cause of the passivity in a change of the chemical properties of the anode material, while the oxide-film theory finds the cause in a change of the mechanical properties of the electrode surface.

The Electrode-Volume Constant-Sum Hypothesis

A paper giving a "review and interpretation of recent experiments which extend and elucidate the domain of the passivity of metals," by Dr. D. Reichinstein, of Zurich, Switzerland, develops a theory which is an extension of the views of LeBlanc that it is velocity phenomena which condition passivity. The principal question discussed is how the transition is taking place from the active state of a metal into the passive state.

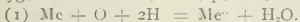
Several series of experimental facts are first discussed with respect to chemical polarization of reversible active metallic electrodes and with respect to "negative depolarization" (when a material added to the electrolyte does not chemically react with it and diminishes the current intensity instead of increasing it).

The author then passes over to a discussion of direct experiments on the passivation of metals.

"Surprise has often been expressed at the fact that a metal turning passive does not, at higher current density, show at least that anodic solubility which belongs to it at small current density. Let a passive metal be dissolved anodically at a low current density a to the extent of 50 per cent, and let it liberate oxygen, so that its dissolution takes place at $a/2$ amp./sq. cm. of metallic surface.

"When we increase the current tenfold, the dissolution will not proceed at the rate of 5a amp./sq. cm.; it does not even take place at the former rate $a/2$ amp., but at a much smaller rate. How is this to be interpreted chemico-kinetically?"

"Let us assume that a passive metal Me generates primarily oxygen. Two reactions are possible:



"The commencement of reaction (2) is designated 'passivity.' The example teaches us that passivity is not a result of two competing reactions, for both of which the velocity would increase with increasing oxygen concentration, but the passivity is caused by two reactions such that the velocity of the one reaction decreases with increasing O concentration, whilst the velocity of the second reaction regularly increases with increasing O concentration."

The author introduces the conception of the "electrode volume." The contact of an electrode with the electrode is assumed not to be formed by the solid metal, but by a solution (or alloy) of all those substances whose ions are represented in the electrolyte. The volume which this hypothetical alloy occupies in the electrode is designated "the electrode volume."

Now concerning this the following important hypothesis is introduced: "The sum of the concentrations of all the constituents of the electrode volume is always constant, whether the current flows or not."

The electrode volume is always fully saturated, and any new constituent, possibly introduced by the electric current, ejects parts of the old substances.

When a metal electrode is treated anodically the oxygen introduced pushes the metal atoms out of the electrode volume—the same will happen on the cathode by the H_2 introduced—and the chemical equilibrium prevailing before the current was flowing will be destroyed. The oxygen now begins to react with the metal atoms of the electrode volume. Let this reaction be of the form:



where Me and O are respectively an atom of the metal and of the oxygen of the hypothetical alloy, and H^+ concerns the electrolyte.

If a is the constant sum of the concentration of the constituents of the electrode volume, then the author shows that the following holds true:

When with a certain current density of the electrolysis the compensation process is so rapid that the oxygen concentration of the electrode volume does not rise above the value $a/2$ (which may be called the critical concentration), then there will be no reason to fear that another compensation process will set in, e. g., generation of oxygen (= passivity). When, however, the critical concentration is exceeded, the compensation process will necessarily be retarded, and the oxygen concentration will increase rapidly until a new compensation process sets in.

The case of solution of gold in cyanide is discussed in some detail.

Photo-Electricity and Passivity

A paper by Dr. H. Stanley Allen, of the University of London, deals with "the photo-electric behavior of iron and the theory of passivity."

The author points out it is not unusual to find that investigations in one branch of science serve to throw light on problems connected with some other subject. The author's research was undertaken in the hope that the study of the physical phenomenon of photoelectricity might assist in elucidating the century-old problem of the nature of "passivity."

The photoelectric effect here referred to is that which is known as the Hertz-Hallwachs effect. It consists in the loss of a negative electric charge or the acquisition of a positive electric charge by an insulated metal when the surface of the metal is eliminated.

The author's researches show that "iron which is chemically active shows large photo-electric activity, while processes

which render iron passive greatly reduce this activity. These facts are in good agreement with the theory which as well as the oxidation theory must be attributed to Faraday that the cause of passivity is to be found in the oxidation of the gaseous layer at the surface of the metal."

Hydrogen Theory of Passivity

A paper by Professor Gerhard C. Schmidt, of the University of Muenster, Germany, contrasts the oxide theory according to which metals are normally passive and passivity is due to a coating of oxide or oxygen on the metal, with the hydrogen theory according to which the metals in question are normally passive and become active only in the presence of dissolved hydrogen acting as a catalytic agent.

The latter view has recently been strongly supported by the author's pupils Grave and Adler on the basis of experimental investigations, but Flade has criticized the result as non-conclusive. The author now describes further experiments made by W. Rathart.

These experiments deal with the effect of a hydrogen charge on the potential and activity of a metal and further give the proof that a metal is rendered active by diffusion of hydrogen ions. This proof is given both for iron and chromium.

In this proof advantage was taken of the fact that when hydrogen is liberated electrolytically at one side of a piece of iron foil it rapidly diffuses through and alters the e. m. f. on the other side which is under test.

The author's views of the phenomenon of passivity agree with those of Grave. "Just as water or other liquids do not boil, even when their vapor pressure is equal to that of the superincumbent pressure, unless a catalyst such as air is present, so the metals which can be passivated dissolve rapidly only in the presence of a catalyst. The most important of such catalysts is hydrogen, since it is dissolved readily and in large amount by metals. Whether other gases are also catalytically active cannot at present be stated, since they are only absorbed to a very small extent by metals. Just as relatively small amounts of air initiate rapid evaporation in large quantities of liquid, so a small quantity of hydrogen can activate large amounts of iron, nickel, and chromium. It might be supposed that the hydrogen, like air in the case of boiling, forms a nucleus around which the ions of metal collect. Further investigation must decide whether this hypothesis is in accord with the facts."

Passivity and Reaction Velocity

In a paper on "the phenomena of passivity" by Professor Max LeBlanc, at the University of Leipzig, Germany, the author first reviews the oxide theory of passivity and enumerates the objections which have been raised against it. He concludes that it is evident that the hypothesis of an impervious layer, which covers the surface of the metal, thereby protecting it from contact with the electrolyte, is open to the most serious objection.

Professor LeBlanc then takes up the study of polarization measurements, when employing a system, metal-a nitro dilute solution of a metallic salt—metal (e. g., copper-copper sulphate solution—copper). It was expected that no appreciable polarization would take place in the case of short-time electrolysis with moderate current density. While this expectation was fulfilled in some cases it was not fulfilled in others. How can this be explained?

In considering the action at the anode, it is conceivable that primarily no metallic ions are formed, as is generally supposed, but that the electrical process consists of a discharge of negative ions and the chemical process of an interaction between the non-electrified radical and the metal. The potential difference would, according to this view, depend upon not only the concentration of the negative ions, but upon that of the separated non-electrified radical, and consequently upon the velocity with which the latter reacts upon the metal.

In a similar manner the cathodic polarization can be explained. When the current passes, the free ions separate, which

must, in the state of equilibrium, according to the law of mass-action, always be present in addition to the hydrated ones. Consequently a diminution in concentration and then polarization follows, owing to the insufficiently great speed of the formation of free from hydrated ions.

An analogous explanation may be made for the phenomena of passivity. With passive metals ionic hydration occurs so slowly at the anode (a certain amount of solution nearly always takes place), that the concentration of free ions and, therefore, the potential difference, becomes so great in a short time that visible separation of the negative radical or of oxygen can result.

From this point of view we must henceforth understand by passivity phenomena all such as result from chemical polarization. Passivity is not the exception, but the rule, and it is not connected with oxygen-containing anions, as was supposed, but it takes place also with chlorides and other solvents, and especially it can also take place at the cathode.

Anodic Oxygen and Oxide Films

A paper by Professor E. P. Schoch, of the University of Texas, discusses "the mechanism of anodic reactions and the behavior of iron and nickel anodes."

The mechanism of any anodic reaction is considered to be the direct attack of the anode by the anions. From Guenther Schulze's experimental work on the electrolysis of phosphate, borate, etc., solutions with an aluminum anode (*Ann. d. Phys.*, vol. 28, p. 789) the conclusion is drawn that a film, or cover, over the surface of an anode will hinder or prevent the direct access to the surface of the metal of complex anions and will bring about the formation of oxygen ions from them. The latter pass on to be discharged on the surface of the metal.

This gives all the facts necessary for the explanation of the chemical change of any anode in any electrolyte. Briefly this explanation may be stated as follows:

"With a clean anode surface the anions come in direct contact with the metal, react with it, and give up their electrons; the rapidity of the reaction and the final products obtained will vary greatly with different metals and different anions.

"The product may be so rapidly formed and so rapidly dissolved that other anions may have unhindered access to the surface. Again the reaction may be slow, or the product may dissolve slowly, or the latter may be insoluble, or it may be hydrolyzed into an insoluble compound, so that other ions may find their progress to the anode more or less obstructed—a condition that may naturally be brought about in almost any case by a sufficiently great current density.

"Many complex ions which are prevented by the obstruction from reaching the surface of the metal will dissociate into an equivalent number of oxygen ions, which reach the surface of the metal more readily.

"Upon discharge these oxygen ions may form the oxide of a metal (as with copper in sulphate electrolytes), or they may form partly an oxide and partly oxygen (as with platinum in sulphate electrolytes), or they may react so slowly that the discharged oxygen still in contact with the metal becomes gradually more condensed as more oxygen ions are forced into the same area with a greater potential difference until finally gaseous oxygen, or perhaps some higher oxide of the metal is formed.

"In the absence of any oxide skeleton the layer of oxygen, on thickening, will soon allow bubbles of gas to escape—as is probably the case with platinum, iridium, etc., in which the oxide formed is within the body of the metal.

"But when a non-conducting oxide skeleton is formed the layer of gas may accumulate to a relatively great thickness as the oxide skeleton grows, and this may prevent more or less completely the passage of any ions to the electrode.

"Finally, if the oxide is oxidizable to a conducting (per-) oxide then a suitable concentration of oxygen produced by its discharge under a sufficiently great potential will produce this peroxide and oxygen gas will then be evolved from the new surface of the peroxide.

"The actual existence of this latter state of affairs has been beautifully demonstrated by E. Mueller (*Zeit. f. Electrochem.*, vol. 13, p. 133) who showed that a copper anode in sodium hydroxide solutions forms hydrated cuprous oxide as long as the current density is small enough; but with a larger current density the anodic potential becomes more noble, copper peroxide is visibly formed on the anode, and then oxygen is evolved from the surface of the latter. It appears that the "oxygen-producing" discharge of ions on peroxide anodes is a rapid reaction, because no great hindrance is observed even with great current densities.

"This beautiful demonstration by E. Mueller of the relations between active and passive copper indicates that the phenomena which characterize the attainment of passivity by some metals in certain electrolytes appears to be nothing but the regular behavior of anodes under the existing conditions."

Professor Schoch has reached the same conclusion from his study of the behavior of iron and nickel anodes in various electrolytes. (*Zeit. f. Phys. Chem.*, vol. 58, p. 301; *Am. Chem. Journ.*, vol. 41, pp. 203 and 235; *Jour. Phys. Chem.*, vol. 14, p. 739.) A review is given of the experimental facts which are then explained by the general theory of anodic reactions outlined above.

"No special attempt was made to state in what particular form the oxygen obtained by the discharge of oxygen ions is present on the surface of iron or nickel anodes, that is, whether it is present as free oxygen or as oxide. To the writer there do not appear to be any facts which establish one view over the other. The assumption of a free oxygen film is perhaps a trifle simpler than the assumption of a formation of an oxide, but the possibility of the presence of an oxide has not been disproven."

Professor Schoch finally discusses briefly the hydrogen catalysis theory.

"Possibly the best experimental evidence for a decision between the hydrogen catalysis theory and the theoretical views we have here advanced in the following.

"A chromium anode that has become passive with reference to the formation of bivalent ions does not become active again to form trivalent ions, but does become active again to form chromation.

"An iron anode that has been made passive with reference to the formation of bivalent ions does not become active again to form trivalent ions. However, when electrolyzed in acetate, or oxalate solutions the electrode becomes active again (that is, the metal dissolves again extensively) at potentials considerably below those (more noble!) at which it has ceased to be active before and at which it is usually passive in other electrolytes. The same thing is true of nickel (*Jour. Phys. Chem.*, vol. 14, p. 719; *Jour. American Chem. Soc.*, vol. 30, p. 1737)

"In all three instances above the activity at the lower potential is due to a reaction which consumes oxygen and thus lays the metal surface bare. This is plainly the case in the formation of chromations. In the other examples the activity of the iron or nickel reappears when the potential is low enough to oxidize the acetate or oxalate, and when the oxygen is removed to oxidize these substances, then the metal surface is exposed and is subject to the attack of the anions."

Passivity and Electrolytic Valve Action

A paper by Dr. Günther Schulze, of the Reichsanstalt, deals with the relation of passivity to the phenomenon of electrolytic valve action. The author emphasizes the remarkable analogy between the phenomena of valve action and passivity which follows from the assumption that valve action is caused by an insulating film and consequently a gas layer while passivity takes place if the film shows metallic (electronic) conductivity together with a limited or negligible solubility in the electrolyte.

The consequence of the metallic conductivity of the film is that it cannot grow much above molecular size and must be almost undetectable by optical or mechanical means. The oxygen developed at its outer surface covers it more or less and protects it against the electrolyte when it has been formed

Thus the current may be decreased below the critical forming value without restoring activity.

The behavior of iron in H_2SO_4 of different concentration is of special interest. In dilute sulphuric acid it shows passivity. In concentrated acid valve action. In intermediate concentrations an unstable valve action is succeeded by stable passivity. In dilute solution the anions SO_4 may form an oxide of iron by the aid of the water. In concentrated solutions the small amount of water present is not available for this reaction, as it is bound to the sulphuric acid itself. Therefore, ferric sulphate must be formed. This insulates and causes valve action.

"Therefore I hold: Valve action is caused by an insulating skin of ultra molecular thickness with a thin gas layer within it; passivity by a conducting skin of molecular thickness with a molecular gas layer upon it."

Electric Furnaces, Their Design, Characteristics and Commercial Application

By Woolsey McA. Johnson and George N. Sieger
Buildings for Electric Furnace Plant

The character of building depends entirely on the stability and certainty of operation. For a small experimental plant a wooden shack 16 ft. x 32 ft., 10 ft. to eaves, 16 ft. to ridgepole, costing \$140, is good enough.

When the work has a reasonable certainty of continuing two or three years an edifice like the new furnace building of the Continuous Zinc Furnace Company, 32 ft. x 60 ft., built of heavy braced wooden framework, with a roof supported on strong wooden trusses, sides, ends and roof covered with No. 26 galvanized sheet steel, is neat and serviceable. This building was designed by F. E. Waterman, architect, of Hartford. The total cost without architect's fee was \$608. This did not include concrete floor, which was afterward laid on a floor of ashes and slag.

For a more permanent structure, a structural steel framework covered with galvanized sheet steel is to be recommended. This will cost nearly double what a similar structure of spruce framework costs. Its main advantage lies in the fact that it is fireproof.

For a permanent structure designed to last ten to fifteen years, a structural steel framework with curtain walls of red brick or hollow tile with a concrete roof covered with tar and gravel or covered with tar and gravel on a felt covering is ideal, but this will cost about 2.5 times what a galvanized-iron structure costs.

Rough figures per cubic foot for average conditions in the United States (and these are not far from wrong for any place as where lumber is cheap, supplies are apt to be dear, and where labor is low it is apt to be inefficient) are given below:

Kind of Structure.	Cost Per Moderate Height Per Cubic Foot
Wood shack	\$0.02 to \$0.025
Galvanized sheet on wooden frame- work	0.30 to 0.35
Galvanized sheet on steel frame- work	0.00 to 0.075
Hollow tile walls steel framework	0.10 to 0.150
Concrete roof covered with tar gravel.	

FIRE PROTECTION

In any electric-furnace plant, as in any metallurgical plant, while the building is in course of erection, it is good business to install a fire hose in a corner or a number of them in corners. These hoses should have a large pipe connection with a quick-turning shut-off valve to city main or system that has good water pressure back of it.

A safe rule is if one puts in 1-in. fire hose to connect it to an 1½-in. pipe. If one wishes to exhibit a great pretence of accuracy and scientific engineering, multiply the diameter of fire hose by $\sqrt{2}$ or 1.414 and then add 10 per cent, guessing at the final proper size for pipe connection. The main point is

to have the pipe connection considerably bigger than the fire hose.

The fire hose with nozzle is coiled on a framework and placed in a box made of double walls of sheet steel or asbestos-wood," and having space between inner and outer walls filled with mineral wool. In the bottom of these boxes is placed an electric heater of 500 watts capacity. If the door shuts tightly, and the heater is on, the fire hose will never freeze up. The nozzle should be securely fastened to the hose so that it cannot be taken off to furnish water for the masons or for mixing concrete.

Around the walls as a further fire protection, small chemical extinguishers using acid and soda ash or "squirt guns," using carbon tetra-chloride, may be provided. These are first rate for small incipient blazes. In case of a fire the first thing for the superintendent to do is to see that high-tension switches are thrown, so that water can be played without men getting "shocked," and then to save any note books and measuring instruments from damage by water or fire. The men will use hoses with zest, energy and dispatch, and the superintendent can smoke his pipe.

The Continuous Zinc Furnace Company has had twenty-seven fires in the past six years and none of them has caused damage amounting to more than \$25. If it had not been for efficient fire protection the amount of damage undoubtedly would have been serious. With such a protection the only damage from fire can come when nobody is at the plant, as during the night when no runs are going.

FOUNDATIONS FOR ELECTRIC FURNACES

In view of the hazardous and uncertain element in electric-furnace work, especially at its inception, emphasis must be laid on the necessity of having foundations which are above all question. Test pits should be dug and a bearing value of ground measured by well-known engineering methods. Then a large factor of safety should be employed.

For a small furnace, weighing from 1 ton to 5 tons, a bearing value of 1000 lb. per square foot is a conservative assumption. In this case a hole 30 in. deep and 48 in. square can be dug, filled with concrete (a 1:1:2.55 mixture is all right). This will hold up 16,000 lb., unless the ground is soft on one side and hard on the other.

A cheaper way is to put in large blocks of masonry or of concrete from demolished buildings and pouring a wet mixture of sharp sand and cement (1 to 1). Old wire, barrel hoops, old pieces of pipe thrown in tie the piece together and make a monolith.

When a larger furnace is to be supported an excavation 48 in. to 72 in. deep can be made. In such a case old girder rail-tied together with corrugated iron bars in regular fashion and a whole mass of 1:3:6 concrete tamped in. Where special strain comes a mixture 1:2:4 can be used in the monolith.

This scheme is good engineering where a total load of 125 tons to 175 tons is to be held up. Fig. 1 shows this.

But when loads exceed 175 tons, there is a risk of uneven settling and piles must be used. A pile will hold up from 10 tons to 40 tons, according to size of pile and according to force of blow that pile-driver used can deliver. The pile should be capped with a steel hoop or "hat" of cast iron. Above the pile special reinforcing should distribute the upward forces in the form of an inverted pyramid. On top of this should come a platform of reinforced concrete.

The reason why piles are advisable is that the certainty of their holding up the load is as absolute as anything can be.

POWER FACILITIES

The authors believe that in general the power facilities should have 20 per cent. capacity in excess of the normal rating of furnace. There always comes a time in the operation of a furnace when it pays to force the furnace, for the reason that the furnace is cold and working slow, or because the ore is especially easy-working and a little forcing will increase daily tonnage. With this reserve power capacity above normal furnace demands the furnace will be operating all the while at its maximum commercial efficiency.

The great tendency in design of electric furnaces is to underrate them, either because of the fact that thermal efficiencies are usually estimated at too high a figure or because heat-balance sheets are made that are too optimistic. Moreover, the power-factor or load-factor rarely comes up to what is hoped for or the drop in potential from switchboard to electrode is greater than was originally estimated.

It may seem paradoxical with a 200-kw furnace to have a 250-k.v.a. transforming or generating apparatus, in view of the fact that any well-designed electrical apparatus is capable of

(2) Raising the voltage up or down by varying field voltage of electric generator or by cutting in or out resistance in field circuit.

(3) Where an electric transformer is used, use may be made of cutting out number of turns in primary or secondary winding or by changing either of these from series to multiple or to semi-multiple, or increasing or decreasing the magnetic reluctance of circuit.

(4) By using a water-rheostat or other variable resistance in series with the electric furnace.

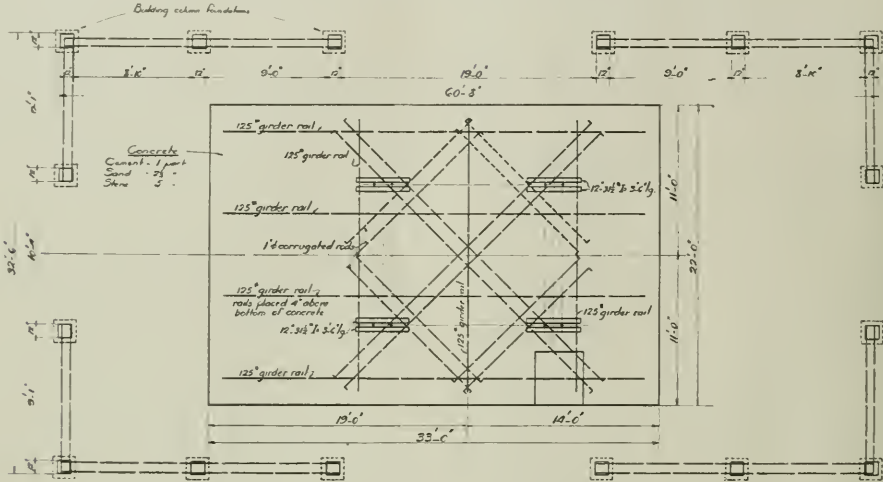


FIG. 1.—FOUNDATION PLAN FOR ELECTRIC FURNACE

being overloaded for short periods tremendously. But if we apply first principles of commercial science we will find that the following reasoning applies:

(1) Electric furnace processes if successful at all have a large margin of profit.

(2) That which makes them successful at all is the powerful metallurgical effect of the electric current.

(3) Therefore, the supply of this should be ample to carry operation along at a maximum rate of daily tonnage at will of the manager.

VOLTAGE REGULATION

There have been few large electric furnaces built that operate at a constant voltage. The reason for this is the fact that the voltage depends on the resistance of the furnace and that this resistance depends greatly on the temperature. Substances that do not conduct cold at all conduct well at high temperature. To use a striking example, a 100,000-volt modern porcelain insulator would break down at 25 volts at 1400 deg. C. With this tremendous change from a perfect insulator to a good conductor, it is evident that the regulation of voltage must be under good control, for temperature is sure to vary from time to time.

Now, one way to do this is to have some automatic regulation of the length of circuit through which the electric current plays. The best example of this is seen in the classical steel furnace of Siemens. As lately practiced by Girod and Héroult in the modern electric steel-refining furnace, the voltage is kept nearly constant by moving the electrode automatically up or down by a relay-controlled motor. The relay is actuated by a device which depends for its motion on some function of the electric current.

When no such internal regulation is used, external voltage regulation is employed. This can come from the following scheme of design

(1) Speeding up or down the prime-mover such as steam engine. Such a process can only be used well when one furnace is operated tied electrically to a generating set.

When doing electric zinc furnace work for the Lanyon Zinc Company in 1903 and 1904, one of the authors used this electrical layout.

An old 75-kw. Westinghouse alternator was belted to the line shaft of the power plant. The field of this was excited by an old Edison direct-current generator. The field current of the former could be controlled by one rheostat and field current of the latter by another rheostat so that any voltage from 300 to 1000 could be obtained from the alternator at a maximum of 125 kw. with a minimum of 45 kw.

This high-tension circuit was put directly on the electric zinc furnace which was of ore resistance type making an infusible residue, when a furnace was started up from the cold.

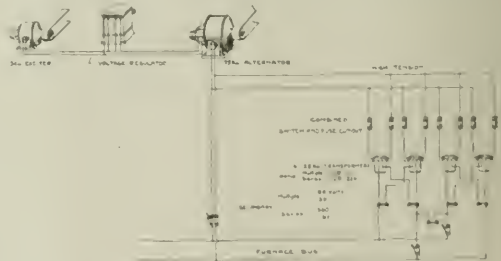


FIG. 2.—WIRING DIAGRAM FOR ELECTRIC ZINC FURNACE

As the voltage decreased this current was transformed by means of four 25-hp transformers (ratios 1100 to 220 and 1100 to 110). So a minimum voltage of 30 volts at 45 kw was at hand. The entire set gave thus a minimum of 45 kw at any voltage from 30 volts to 1800.

This scheme of electrical connections is shown in Fig. 2. It is a good layout for experimental work where plenty of surplus mechanical power is at hand.

ELECTRIC FURNACE TRANSFORMERS LOW CAPITAL COST THE ESSENTIAL

Electric-furnace transformers are a special department of electrical engineering. In general cases a number of taps are brought out from the primary winding and by some switching device these can be cut in or cut out so as to make a varying transforming ratio and a variable voltage on furnace circuit.

A heavy double-throw switch on the secondary puts the secondary coils in series or in multiple, thereby changing the transforming ratio 100 per cent in one operation. This is the usual design.

Special devices such as choke coils in series or in shunt in the primary or some device that will change the magnetic reluctance of the transformer core may be employed.

The main consideration in designing an electric-furnace transformer is to make a practical machine and *one of low capital cost*. It is evident that with power at 2 mills to 4 mills per kw-hour little money is saved by increasing the electrical efficiency from 96 to 99 per cent.

Rather should one pay a few more dollars on the power bill and cut down hundreds of dollars on the interest and depreciation account, for in a new industry this figure the authors place at 20 to 35 per cent per year.

Good commercial design of an electric-furnace transformer has this commercial fact incorporated in itself.

The subject of electric-furnace transformer design is an important one and will be made the subject of a forthcoming separate article of this series.

The transformer plant should be either outside on a platform or in a separate building away from dust and smoke.

The design of a transformer station of an experimental plant, of course, is quite different from that of a regular furnace plant. Here the main point is to buy as little as possible and it is good business to waste power seemingly quite recklessly but really scientifically.

Any experimental plant should have on the high-tension circuit either an oil-switch or a "disconnecting" switch on pole-better both. In time of trouble these can be "pulled" and "juice cut off" so that repairs can be made in safety.

Fuses should be eliminated or of large capacity so that nothing but a prolonged short-circuit will blow them.

High-tension circuit breakers made so as to break the current instantaneously and placed so that all breaking of the furnace current is done on the high-tension side are the right things to install.

At the plant of the Continuous Zinc Furnace Company there

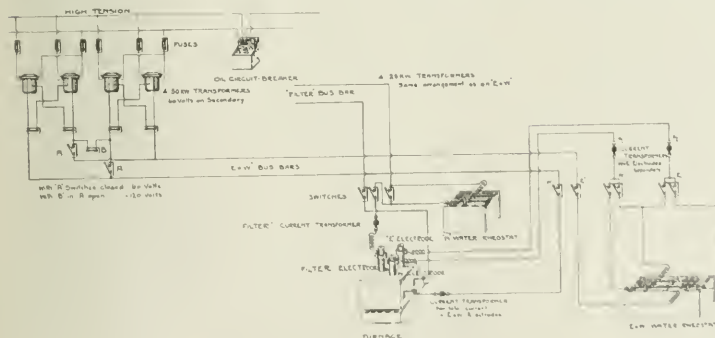


FIG. 3. DIAGRAM FOR ELECTRICAL CONNECTIONS

are four 50-kva transformers on each phase. The voltage ratio is 2400 to 240 to 120 on each transformer. Each phase has the transformers arranged in two sets of two transformers in series.

The various voltages on the bus-bars are secured by connections made both on the primary side and on the secondary side.

Provision is made for connecting the sets together in parallel when 60 volts are desired, or in series when 120 volts is wanted. The two transformers of each set are connected in parallel 240-volt connection can be made by disconnecting the bus bars and putting the primaries of the four transformers of each group in multiple, and the secondaries of each set in series.

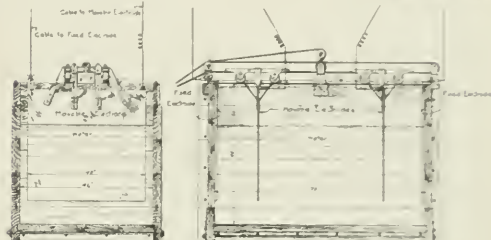


FIG. 4.—WATER RHEOSTAT

Intermediate voltages between zero and 60, or 60 and 120, 120 to 240, are obtained by the use of water rheostats.

Recently there has been installed an arrangement of disconnecting switches on the primary side so that the primary of each of the four transformers can be connected in series, whereby 30 volts on the secondary is obtainable.

Thus there is at disposal out of standard apparatus that will give 125 kva as a minimum at any voltage from 30 volts to 240 by imperceptible gradations.

Fig. 3 shows this.

WATER RHEOSTATS

A strong, durable and efficient water rheostat has recently been constructed at the Continuous Zinc Furnace Company's plant. The vat is 72 in. x 46 in. x 48 in. and is made of cypress wood 3 in. thick. All joints were splined and white-leaded. The whole is securely held together by the use of 5/8-in. rods running vertically and horizontally. The vat is designed to regulate the voltage of two furnace electrodes on the same phase.

Regulation is obtained by the use of three electrodes made of 3/8-in. perforated boiler plate 40 in. x 42 in. One of these, the neutral, is located in the center and is stationary. The other two are movable. The movable plates are attached to No. 1 Standard Fairbanks trucks, the tracks for same being 2-in. tee bars running across the top of the vat. The trucks are easily moved in either direction by means of ropes and pulleys.

Electrical connection is made to the boiler plates by first cleaning a small area of the plate and copper-plating it. A small piece of copper bar is then soldered to it and the connection is given mechanical strength by the use of a bolt. Fig. 4 shows this.

BARREL RHEOSTAT

An ordinary strong barrel (gun or whiskey) is used to an advantage in the construction of a cheap and serviceable water rheostat. In rheostats of this type used at Hartford, the fixed lower electrode consisted of a coil of 1000 copper wire about 3 ft. in length

on which was placed an iron disk, 15 in. in diameter. A water-tight connection was made through the barrel by tightly screwing a brass nipple in the barrel, running the copper wire through this, solder being used for the seal. The copper wire was tinned and the inside of the brass nipple was tinned before pouring in the seal of solder.

The movable electrode was made of a perforated disk 15 in.

thick and 15 in. in diameter, connected to a 1-in. pipe as shown. To the upper end of the pipe a four-way was attached.

This served the double purpose of a means of support and for electrical connection.

The whole was suspended from a pulley by a $\frac{3}{4}$ -in. hemp rope to the four-way.

Electrical connection to movable disc was effected by using a brass nipple and a flexible cable which was soldered to the brass nipple.

The idea of having copper wire enter through the bottom in a perfect seal, so as to avoid troubles of internal short-circuit, was suggested first by Mr. Cole, electrical engineer of the Johns-Pratt Company.

All rheostats should be placed outside, using soda ash, which is non-corrosive during all months save December, January and February, when common salt, which is slowly corrosive but which will not freeze, should be used as the solute to give requisite conductivity.

The big water rheostats will absorb without trouble 40 kw

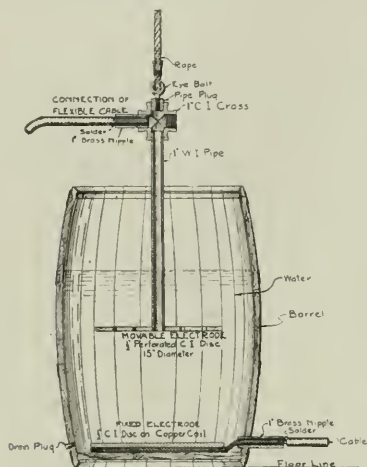


FIG. 5.—BARREL WATER RHEOSTAT

each. A barrel rheostat will absorb in constant service only about 5 kw.

Fig. 5 shows the Continuous Zinc Furnace Company's barrel rheostat.

BUS-BAR AND CABLES

Undoubtedly the cheapest way to carry electric-furnace current is by bus-bars. These sell at 2 cents to 4 cents above the price of ingot copper and will carry at least 25 per cent more than the same amount of copper in a bare cable and nearly double the amount of copper in an insulated cable. They can be bolted or clamped together. At the Hartford plant of the Continuous Zinc Furnace Company bus-bars are used for almost all switchboard work. We have a lot of right-angle bends and other short pieces, which are used just the way a plumber uses T's, L's, etc.

These are fastened together with malleable iron carpenters' clamps 4 in. to 6 in. in size. It is surprising how serviceable these ungainly circuits are, especially if care has been employed to see that the contacts are bright and smooth as all copper contacts should be. Bus-bar circuits are interlaced to cut down the self-induction and to raise the "power factor."

To the furnace, of course, flexible cables lead to electrode connection-clamps. The lugs for these should be of high-conductivity bronze or forged copper. Where possible the cable should be pressed or forged into place or brazed into lug. Soldering is not the right way to do it, although it can be used. A special electric furnace cable with a core of asbestos rope of about perfect properties and characteristics is made by the General Electric Company.

Intercrystalline Cohesion of Metals

A second paper on this subject was presented by Dr. W. Rosenhain and Mr. D. Ewen at the recent meeting of the (British) Institute of Metals in Ghent, Belgium.

This paper is a continuation of a research undertaken in order to test by experiment the theory that the crystals of a metal are held or "cemented" together by thin films of the same metal in the amorphous or undercooled liquid condition.

In their previous paper the authors obtained evidence in favor of this theory from the behavior of metals when heated in a high vacuum. Since then the theory has received further support from the researches of Rosenhain and Humfrey on the tenacity and fracture of mild steel at high temperatures (Iron and Steel Inst., May, 1913). There it was shown that at high temperatures the type of deformation and fracture obtained by any but very rapid strains was entirely different from the type found at low temperatures. In the latter type both deformation and fracture occur principally within the crystals, while the boundaries act as stiffening and strengthening ribs; at high temperatures, on the other hand, the crystals remain almost undeformed while movement and ultimate fracture occur principally in the intercrystalline boundaries.

Detailed consideration of the observed facts shows that they can be readily explained by the amorphous cement theory—the cement being strong and hard at low temperatures, but, like all very viscous fluids, becoming soft and mobile at high temperatures.

An extension of this idea shows that at a sufficiently high temperature in all pure metals the "cement" must become so fluid as to possess no appreciable strength, while the crystals still retain a definite degree of strength and hardness. It follows that at such temperatures the crystals, even of the most ductile metals, should be capable of being pulled apart without undergoing deformation and the metal should exhibit extreme intercrystalline brittleness.

This has been experimentally verified in the case of lead, tin, aluminium and bismuth, photographs of the strikingly intercrystalline fractures thus obtained being given in the paper—the brittle fracture of lead at 323 deg. C. (melting point 327 deg. C.) having an appearance like that of a brittle casting. These experiments show that the well-known brittleness of metals near their melting points is due to loosening of intercrystalline cohesion as indicated by the amorphous cement theory.

The authors deal with the suggestion that impurities concentrated in the crystal boundaries may affect these results and show that this objection cannot apply to their experiments and they strengthen this view by showing that a pure eutectic alloy (that of lead and tin) exhibits analogous phenomena.

Microchemistry of Corrosion

A paper by Dr. Cecil H. Desch and Mr. Samuel Whyte, entitled "The Microchemistry of Corrosion. 1: Some Copper-Zinc Alloys," was read at the recent meeting of the (British) Institute of Metals.

Experiments are described in which uniform areas of various alloys are corroded for a definite time under the influence of an electric current. The material dissolved by the electrolyte is analyzed, as well as the adherent metallic deposit. The surface of the alloy is also examined microscopically before and after the removal of this deposit. Only a few milligrams of material have to be used for each full analysis.

The alloys used in the first series of experiments are beta alloys of copper and zinc with and without the addition of tin or iron. Quenching from above the transformation point (470°) increases the corrosion, although the action is slightly retarded at first. Iron hastens corrosion, while tin retards it. The influence of the latter element, however, is chiefly mechanical, by means of the formation of a tough, adherent layer of basic salts, which serves as a protective coating.

The microscopic appearance of the corroded surfaces is characteristic.

The Iron Blast Furnace and the Characteristics of Its Fuels

Fundamental Conditions—Definition of Blast Furnace—Classification of Fuels Used—Essentials of Plant

By J. E. Johnson, Jr.

Iron is one of the most common elements in nature. Geologists state that it comprises 7 per cent. of the earth's crust and that there is strong proof that the whole interior of the earth below a depth of a few miles is composed of iron. The vast segregations which have gone on throughout geological time have resulted in concentration of iron to all degrees from the lowest up to 72 per cent. The concentration never exceeds this degree for the reason that iron is unlike most other of the useful metals in one important particular. It is seldom or never found native on the earth, except in the form of meteorites which have fallen from space.

This is in consequence of the great affinity of iron for oxygen, with which it will unite under almost all conditions of temperature and chemical environment found on the earth. This brings us to the fundamental fact in the metallurgy of iron which distinguishes it from all the other useful metals.

Its only ores are its oxides. The sulphides which are so important in the occurrence and metallurgy of other metals are not regarded as ores of iron, but only as ores of sulphur.

When the sulphur has been burnt off for conversion to sulphuric acid the iron takes up oxygen to replace the sulphur and becomes an oxide. These artificial oxides then become iron ores under certain conditions and with certain limitations which will be considered later.

But the use of iron sulphide as iron ore either by burning the sulphur off to waste, or directly in the furnace, as is so commonly done with the sulphides of other useful metals, is entirely unknown in the metallurgy of iron.

It is true that carbonates are important sources of iron in Europe, but their carbonic acid is not strongly held and before they are charged into the furnace are driven off by an oxidizing roast which converts the resulting ferrous oxide FeO to ferric oxide Fe_2O_3 . This reaction takes place so easily that it will occur slowly under ordinary atmospheric conditions, but with the slight modification in that case that the ferric oxide takes up water of combination and becomes limonite.

More than 90 per cent. of the world's production of pig iron is derived from ores which occur in nature as oxides and the remainder comes from the roasted carbonates and the ferric oxide or "pyrite cinder" derived from the sulphides.

From the point of view of the blast-furnace, therefore, it is correct to say that the only ores of iron are the oxides.

The removal of the final traces of oxygen from iron is so difficult that it can commonly only be done with an expensive gas like hydrogen or methane or by solid incandescent carbon and as the latter is commercially so much more available we may say that industrially iron is reduced only with solid carbonaceous fuel.

Fundamentally then the metallurgy of iron consists of the reduction of the oxides through the use of solid carbonaceous fuel. This operation is carried on to the extent of a few hundred tons per year in the Catalan forge and to the extent of some sixty or seventy million tons in the blast furnace which, therefore, constitutes the foundation of the iron industry.

The blast-furnace is a shaft furnace, the "high-furnace" of France and Germany, and takes its English name from the fact that the air to support combustion must be forced into it under pressure because of the resistance to the passage of the combustion gases offered by its high shaft filled with material. It became, therefore, a "blast" furnace as opposed to a draft furnace, obtaining its air through chimney draft.

The blast-furnace is historically the outgrowth of an attempt to utilize the waste heat of a "low furnace" or Catalan forge so as to reduce the fuel required for the latter and is probably the first industrial application of what is called the counter-current principle.

This may be defined as a method of transmitting the properties of one mass of matter to another by causing the two masses to travel past one another in opposite directions in intimate contact. This process leads naturally also to continuous as opposed to intermittent operation, one of the great desiderata of industrial processes.

This practice has so long and so universally prevailed in the iron industry that it has caused us to overlook one of the most important facts concerning the blast furnace and one to which we shall have occasion to revert many times; namely, that it is an apparatus in which we have merged two distinct processes. First, the reduction of iron oxide to metallic iron. Second, the melting of the reduced material.

The deep fire necessary for melting the "iron sponge" or reduced iron in a reducing atmosphere (without which it would instantly become oxidized again) was not unlike a gas producer in its construction and operation, and consequently discharged a hot gas rich in CO .

The utilization of this heat was the object sought by those early unknown pioneers who added the shaft above the low hearth or Catalan forge, increased the blast pressure to suit and so transformed it to the blast-furnace.

The fuel economy desired was obtained to such an extent that the low hearth declined to a position of insignificance, although the change brought many other changes not contemplated by the pioneers, some of which were not to their liking.

Most of these must await discussion in a subsequent article, but one was vastly important because it made a radical alteration in the character of the product, and in the process. The original product was tough and pasty, could not be melted but could be "wrought" directly under the hammer, while the product of the improved furnace could not be worked under the hammer, either hot or cold, passing immediately from the brittle solid state to the liquid state in which it could be "cast" or poured. Hence the names of the two principal varieties of iron, "cast" and "wrought."

The production of the iron in the liquid condition contributed further to the continuity of the process since the iron could be allowed to gather in a pool at the bottom of the furnace instead of requiring to be worked out in lumps or "lumps" with fire tools, when ready; and the subsequent introduction of a fresh charge of ore.

The shaft-furnace along with these advantages brought a requirement of far-reaching importance. The column of stock must be penetrable by a considerable gas volume at a reasonable pressure. This requirement is just as important to-day as it was in the early day of which I have spoken, though it must be confessed that our ideas of a "reasonable pressure" have undergone great changes.

This penetrability of the stock column may be seriously if not fatally impaired by either of two conditions.

First, the presence of tarry or sticky matter which would fill up the interstices of the charge and make it practically gas-tight.

Second, great fineness of the charge, either of the ore or of the fuel. Moreover, the fuel is subject to a certain solvent action by the gas as will be described later, and this is more active in proportion to the fineness of the fuel so that serious loss occurs if the latter be too fine.

The two principal fuels available, wood and bituminous coal, in their native condition both contain tarry material, but in very varying quantities. The vast majority of their varieties are, therefore, not adapted to use as furnace fuels without preliminary treatment to drive off the tar, so that this condition is industrially of paramount importance.

We can now combine all these necessary conditions to form a fundamental definition as follows:

The blast-furnace is a vertical shaft-furnace blown with air under pressure for reducing iron from the oxides and delivering it in a liquid condition. Its fuel must be not only solid but in lumps of considerable size and must be sufficiently free from tarry components to permit the passage of the gas resulting from the combustion in its hearth.

In addition to these fundamental requirements of a blast-furnace fuel there are others of nearly equal importance.

The three characteristics next in importance to those mentioned are first, purity, particularly freedom from sulphur and phosphorus. Second, porosity or ratio of surface to mass. Third, physical strength or resistance to crushing.

The reason for the importance of *freedom from sulphur and phosphorus* is that next to carbon these two elements exert the most important influence of any of the common constituents in steel both being destructive of quality if present to an extent greater than one-tenth of one per cent. If present in the pig iron of which steel is made they cannot be eliminated at all when certain processes are used and in the others only with increased difficulty and expense. In foundry irons phosphorus is not objectionable within certain limits, but sulphur is as much so as in steel.

The blast-furnace may be so operated as to remove the sulphur in the charge but only with increased cost for fuel and flux and with decreased output and in some cases with reduced quality.

This is not the case with phosphorus, virtually all of which present in the charge goes into the pig iron, in normal operation.

For these reasons sulphur in fuel is always bad and progressively worse as its amount increases, although its removal is possible.

Phosphorus in the fuel, on the other hand, may not be objectionable in any given quantity for certain iron, but if objectionable is prohibitive of the use of that material because its removal in the blast-furnace is impossible.

The importance of *porosity* of the fuel comes from the fact that the velocity of combustion of carbon in air is not infinite but limited. Only a limited amount of carbon per unit of surface exposed can combine with oxygen no matter how favorable the conditions and in order to secure the rapid combustion necessary to produce high output from a given size of furnace, much surface must be exposed by the fuel.

This cannot be done by reducing its size and so increasing the ratio of surface to volume and mass except to a very limited degree for reasons already explained, and high combustion velocities can, therefore, only be obtained with porous fuels.

The porosity also has a direct bearing on the volume of given weight of fuel, and this also will be shown subsequently to have an important influence on the operation of the blast-furnace.

Physical strength is important because without it no matter how large the lumps of fuel may be initially they are broken by the superincumbent weight of the charge column as the coke descends into the furnace and we have that condition of fineness of fuel which is so objectionable.

On account of the fundamental effects on the operation of the furnace and the quality of the iron caused by these chemical and physical variations in the fuel used, furnaces and the irons they produce are commonly and quite properly classified by the fuels used. These are in the order of their importance as follows: Coke, charcoal, anthracite, raw coal.

Coke

This subject is so important that large volumes have been written about it to which those in search of profound information are referred. It is only possible to give a bare outline here.

Coke is a light, very porous fuel produced by the distillation of mineral coal containing sufficient bituminous matter to cement together the whole mass at a certain temperature.

The distillation drives off nearly all the volatile matter and good coke consists almost entirely of fixed carbon and ash. Coke is made in four ways, of which only two are metallurgically important.

First, in *heaps or mellers* on the ground. These heaps are covered with turf and dirt to prevent the access of sufficient air for complete combustion. The limited combustion permitted generates heat enough to drive off the volatile matter from the

remainder of the coal and coke results, but the yield of coke per ton of coal is very small, the cost is high, and the valuable by-products are of course completely wasted in this process.

For these reasons this process is not in industrial use so far as known to me, though it is sometimes used to determine whether a given coal can be coked, and once during the last boom in the iron business it is said to have been used for a short time to secure a supply of coke for a furnace which could not get one otherwise, but the broad statement that the process is not in industrial use stands.

Second, *gas retorts*. One process of supplying illuminating gas for cities consists in distilling a high volatile coal in airtight retorts with external heat. The coke is purely a by-product and is used in part for heating the retorts and in part for domestic and other non-metallurgical uses.

Its quality is very poor, being soft and friable owing in part to the kind of coal from which it is made and in part to the process of manufacture; for this reason it is not used in blast furnaces at all, its interest comes from the fact that the gas retort was one of the progenitors of the by-product oven.

Third, *bee-hive ovens*. These now produce about 75% of all the coke made in this country, and are, therefore, of vast commercial importance.

They consist of a bee-hive shaped structure of firebrick generally from 9 to 13 feet in diameter approximately hemispherical in shape. They have a floor of fire brick, a doorway about 30" wide by 36" high, in one side, at the bottom and a charging hole and smoke outlet combined in the center of the dome known as the "eye." These ovens are always built in batteries and frequently in double batteries with the doors in one row on the opposite side to those in the other. The space between them is filled with dirt level with their tops and a track runs over the top of each row to carry a "larry car" which delivers the coal to them. In the case of a double battery the track may be between the rows and the larry have a spout on each side to deliver the coal to either row desired.

The track for the larry is in good practice, always supported on brick pillars coming up from below the base of the ovens so as to eliminate the jar of the passage of the larry cars from the oven proper, as the life of the latter is seriously curtailed when it gets this continual pounding.

In operation the floor and walls of the oven are heated by maintaining a fire on the floor for several days. Then coal which will fill the oven to about 50 per cent. of its capacity is charged through the "eye" in the dome; and leveled off. The doorway has previously been built up with brick and clay almost to the spring of its arch which prevents the coal from running out. After the coal is leveled off, the rest of the door is built in the same manner; airtight except that two or three draft holes are left.

In a short time the heat of the oven begins to distill off the volatile matter which ignites and burns in the free space in the dome. This further heats up the coal and causes more volatile matter to distill off and burn in its turn. The whole mass becomes heated and fuses together in a plastic condition, its temperature being a dull red. The gas from the lower portion of the coal forces its way out through the plastic mass with the result that its structure is cellular to a high degree. John Fulton in his work on coke estimates that good cokes have a cell-space of about 45 per cent. of their total volume.

As a result of the decreasing combustion when the gas evolution diminishes, the temperature begins to fall and the mass of coke develops vertical cracks.

The process is not finished until the volatile matter is reduced to 1 or 2 per cent. As the amount of volatile matter coming off diminishes by degrees, the supply of air through the draft holes is cut off so that it may be, as nearly as possible, only as much as is needed for the combustion of this gas without leaving any air free to attack the coke itself. The air supply is then shut off altogether and after a few hours a spray-pipe is introduced through the top of the doorway and the surface of the coke thoroughly watered down. As little air as possible is admitted during this operation so as to

prevent loss of coke by combustion. After the oven is watered down the brick door is removed and the coke pulled out with hooks, rakes and other appropriate tools on to the platform in front of the oven where the watering-down process is further carried on if any of the coke is hot enough to require it.

As soon as possible after the oven is empty the brick door is reset and the oven charged again. This time the heat of the floor and walls of the oven is sufficient to ignite the coal in a short time without any preliminary firing.

The process is then carried on continuously, charging and drawing as rapidly as possible, because after the oven gets too cold it will not ignite the incoming charge of coal at all, or if the charge becomes ignited more of it has to be burned to waste to make up the heat lost by the oven by radiation while standing idle. The work of drawing is laborious and very hot. For this reason oven pullers are among the most difficult of labor forces to keep in regular supply in an industrial organization.

On account of the severity of the labor it is not customary to draw ovens on Sunday and this, with the necessity for having an oven charged immediately after drawing, necessitates two periods of burning. These are commonly 48 and 72 hours. This is regulated in large part by the size of the charge and in part by the supply of air admitted for combustion. The yield of coke in percentage of the coal charge is larger in good practice with 72-hour coke than with 48-hour, but the yield of coke per oven per day is considerably smaller with 72-hour than with 48-hour coke.

The coke made in the two periods differs to some extent in its physical characteristics. Seventy-two-hour coke is considered harder and, therefore, more desirable for foundry purposes.

The operation of coke ovens on this plan is so simple that it can be done by almost anyone, but in order to get the best results much skill and careful oversight are required. It is not impossible by superior management to increase the yield of coke per ton of a given coal from 60 per cent to 70 per cent of the coal charged in the same ovens. The matter of drafting is particularly important and a large percentage of the coke in the front of the oven may be burned up by improper draft.

It is considered by some coke-oven authorities very objectionable for all the air to be admitted at the relatively low elevation of the top of the doorway and all in one place because, owing to the delay of the air in becoming thoroughly mixed with the volatile matter as the latter distills, the air has an opportunity to attack the coke.

Owing to these considerations ovens have been built with an annular passage around the dome above the level of the doorway, this passage communicating with the oven proper by narrow radial slits in the brick work around the circumference of the oven and communicating with the outside air through a draft door. There is little doubt that better results can be secured with this construction if proper attention be paid to it, but the annular passage is likely to become filled with dirt and the narrow ports stopped up. These are difficult to clean out and when this is neglected the oven is no better than the plain oven which costs rather less. With first-class supervision the annular air-passage is probably a valuable feature, but where the operation of the ovens is left to careless and untrained men its benefit would be short-lived.

The volatile matter which distills off from the coal contains a considerable quantity of tar and during the operation this breaks up at the surface of the mass of coke, presumably, owing to the intensity of the radiation in that region, depositing fixed carbon of a beautiful silver-like luster on the face of the coke. If the watering-down is done without admission of much air, this will be cooled below its ignition point before air reaches it and the coke when drawn from the oven will have the silver luster by which its quality has been so widely judged in the past and is even yet to a considerable extent.

Fourth, *by-product ovens*. Even with best practice there is considerable loss of fixed carbon in coking in beehive ovens.

Moreover the coal contains valuable by-products, especially tar and ammonia, which cannot be recovered in ordinary practice with the beehive oven. Also, the heat-value of the volatile matter is much in excess of that required for the coking operation. This is plainly indicated by the considerable flame rising from the "eye" of every oven in operation.

In the distillation of coal in gas retorts and its coking in coke ovens we have exactly the same fundamental operation carried on for two different purposes and for two different products. In the case of the retort the gas is the desired product, the ammonia and tar are recovered from it and sold. The coke is of little value.

In the beehive oven, on the other hand, the gas is distilled off from the coal much as in the retort, but the gas itself is thrown away or burned up, as are its valuable contents of tar and ammonia, the coke itself being the only product.

It was natural that in the course of industrial development attempts should be made to consolidate these two processes and to make simultaneously coke for metallurgical purposes and gas useful for heating and lighting, with the recovery of tar and ammonia. Such attempts began in Europe more than 50 years ago and owing to the greater cost of coal was the lower cost of technical supervision and labor the result. Resulting from these attempts have almost completely supplanted the beehive oven. In America the opposite conditions prevailing, beehive coke still constitutes three-fourths of the total output, but the by-product oven is making rapid strides and it is only a question of time when it will have displaced the beehive oven as completely as it has done in Europe.

In its essentials the modern *by-product* or *retort-oven* consists of a firebrick chamber about 18 inches wide, 6 feet high and 18 feet long, with flues on each side in which gas and air are burned. The ends are closed by metal doors lined with brick set against metal frames built into the brickwork of the oven. Instead of one opening for charging in the top there are three or more, and instead of the laborious operation of drawing by hand the finished charge is pushed out of the oven by a mechanically actuated ram carrying a sort of rectangular piston of the same general shape as the oven, but somewhat smaller. This is so powerful that it forces the whole mass of coke out at the opposite end of the oven.

There it is caught in a suitable grating or rack, and quenched with a heavy shower of water as quickly as possible. But no matter how quickly this may be done it has had access to the air and enough combustion results to destroy the silver luster, characteristics of good beehive coke. This makes the coke look black and lusterless. For this reason furnacemen in the past have frequently been prejudiced against it and in fact it was a matter of years before the builders of by-product ovens were able to produce coke of metallurgical quality equal to the best beehive. But that point has now undoubtedly been reached.

By-product ovens are built in long batteries parallel to one another and only separated by the air and gas flues on each side. The modified "larry" for delivering the coal to them runs overhead as in the beehive oven, but in handling the gas a vast difference exists. No air whatever is admitted to the ovens, the greatest pains being taken to preserve them absolutely air-tight. The gas is taken off through pipes which deliver to a common gas main, and all the gas is sent to a central purification plant where the tar and ammonia are removed and a portion of the gas is then returned to supply the heat needed for the coking process. The rest of the gas is available for either city lighting or industrial purposes.

In order to maintain the high temperature necessary to force the heat for coking through the firebrick walls of the ovens regenerators are commonly used to preheat the air for combustion and where regenerators are not used recuperators are substituted for them. The results obtained with these latter are not so economical of gas as those obtainable with the regenerators and the best practice tends more and more strongly to the use of the regenerator oven.

These regenerators are commonly built as the substructure

of the oven itself. Sometimes each oven has its own separate regenerator, and in other cases the regenerators are common to a whole battery of ovens.

The first two types of ovens to achieve commercial success in America were the Semet-Solvay and the Otto-Hoffman. The former designed uses horizontal flues, the latter vertical. In more recent years the Koppers oven or ovens of which it is the prototype have taken the lead over the other two. These have vertical flues like the Otto Hoffman oven of an earlier date, which they have largely replaced.

The yield of coke with the by-product oven is always higher than with the beehive. In fact the quantity of fixed carbon in the coke is greater in good practice than the fixed carbon in the original coal.

This apparent paradox comes from the fact that during the operation the tar and volatile matter of the coal are broken up by heat as described for the beehive oven, but unlike the latter the by-product oven does not permit the combustion of any appreciable fraction of the coke produced. The consequence is that the fixed carbon of the coal is augmented by the fixed carbon resulting from the breaking up of the bituminous matter without any loss through combustion.

A wider range of coals can be successfully coked in the by-product oven than in the beehive, though, on the other hand, some coals that can be successfully coked in the beehive oven swell to such a vast extent during the distillation of their volatile matter that it is impossible to push them from the by-product oven. These, however, are few in number.

Much attention has been paid to the best percentage of volatile matter for coking purposes in by-product ovens and its line of commercial development has been very different from that of the beehive oven. The latter has very generally been built as close to the mouth of the coal mine as possible with the object of avoiding transportation charges to the point of consumption of the volatile portions of the coal, and, as the yield from such ovens is relatively small, this is an important consideration.

With the by-product oven, on the other hand, the yield of coke is very much larger and moreover the gas is of great value, as above mentioned. In addition to this the by-product oven will do its best work on coal of a certain content of volatile matter approximately 25 per cent. This is naturally not always found in nature.

The volatile matter in coking coal covers a wide range. It is only from 18 to 20 per cent in the excellent coals of the New River and Pocahontas field and as high as 40 per cent in some other coals. The coal of the Connellsville district contains from 27 to 30 per cent.

In order to secure the advantage of coal the nearest possible to the exact composition desired, recourse is often had to mixtures of different varieties of coals. Mixtures of two are common and mixtures of three or more are sometimes used. Such mixtures enable the best results to be obtained in yield and quality of coke, in quantity of gas and reduction of coking time.

The advantage of making mixtures operates in the same direction as the value of the gas and the yield of coke in making it very desirable to place the ovens at the point of consumption of the coke rather than at the mouth of the mine. These considerations are so powerful that by-product ovens are almost always so located.

Coke as a Furnace Fuel

Coke has many of the most desirable characteristics for a blast-furnace fuel. Well-made coke from good coals is always in large lumps thereby supplying one of the fundamental requirements of a blast-furnace fuel. It has sufficient strength to resist breaking during the necessary handling which it undergoes both while it is being charged and during its descent through the furnace. It has a highly cellular structure which exposes the largest possible amount of surface per unit of weight. Consequently a very large amount of coke may be burned in a given area per unit of time, one of the necessities for rapid operation and high output of furnace.

The tarry constituents in the volatile matter of the original coal are completely destroyed by the coking operation, and there is, therefore, no tar to fill up the interstices in the charge and cut off the passage of the gases of combustion.

The worst objection to coke as a furnace fuel is its sulphur content. This varies from 0.6 per cent in the very best coke up to 2 per cent or more in inferior cokes. But when this limit is much exceeded the fuel ceases to be suitable for the blast-furnace no matter how desirable it may be in other respects. In ordinary good cokes the sulphur will vary from 0.8 to 1.5 per cent.

The second objection is its content of ash derived from the original coal; this varies from about 6 per cent in extraordinarily pure cokes to 18 or 20 per cent in poorer ones. Ordinary furnace fuels range in ash from 8 to 12 per cent.

Phosphorus is ordinarily low, generally under 0.05 per cent, but its amount is important because it all goes into the iron and 0.03 per cent is the maximum amount allowed in some special grades of iron. As some phosphorus is always present in the ore and some in the flux the amount in the coke is of great importance in such cases.

The sulphur can be removed to a very large extent by proper use of fluxes, but the melting of the slag resulting from the use of these fluxes requires more coke and therefore increases the cost of iron made from high-sulphur fuel. Similarly the ash which generally consists in large parts of silica has to be fluxed and this is objectionable for the same reason.

It is, of course, obvious that the more ash the less fixed carbon and the fixed carbon is the only useful component of the coke. This ranges from 80 to 92 per cent, but good furnace cokes ordinarily contain from 85 to 90 per cent of fixed carbon.

Charcoal

It is an interesting fact that the development of the manufacture of charcoal has been along a similar line to that of coke there being three principal methods. Pits or meillers, beehived shaped ovens (ordinarily called kilns when used for charcoal), and retorts. There is no equivalent in the case of wood for the process in which distillation is performed for gas only and the resulting charcoal practically disregarded, as in the case of coal.

Charcoal is in general made from cord-wood, trunks and limbs of trees cut 4 feet long and in good practice when more than 10 inches in diameter, split down to a size equivalent to an 8-inch stick. There are two general classes of woods recognized in charcoal manufacture known as "hard" and "soft." The principal hard woods are hickory, hard maple, beech, yellow birch and several varieties of oak, while the soft woods embrace principally the resinous woods, very commonly the yellow pine and allied species.

The production of soft-wood charcoal is smaller and less important than that of hard-wood charcoal and the value of the coal produced therefrom is less for furnace purposes, it being lighter and softer.

Pits.—In the production of charcoal by this method a space of ground about 30 feet in diameter is leveled off and on this wood is stacked up on end in the form of a truncated cone two lengths or 8 feet high, leaving a chimney through the center. The entire structure is then covered with earth and leaves and fired at the bottom of the chimney. A supply of air is admitted insufficient for complete combustion. The volatile matter is distilled off and burns, supplying the heat to keep the operation going. This continues for a week or ten days, when the wood is all charred and the process completed except the cooling off of the charcoal by standing covered.

There is, of course, some loss of charcoal through excessive air supply and as a result the yield of charcoal per cord of wood in this process is poor, nor is there any recovery of by-products. As a consequence of these disadvantages this process has gone out of use almost completely.

Kilns.—These constitute the next step in the development of charcoal production. They are built of red brick because only low temperatures are needed for the charring of wood and

are very much larger than coke ovens, about 30 feet in diameter at the bottom tapering to about 24 feet at a height of 18 to 20 feet, the whole covered with a flat dome rising about 6 feet further. They are provided with three rows of draft holes at regular intervals, one immediately at the bottom, one about 3 feet higher, and the third about 3 feet above the second. An opening through one side of the dome, about 5 feet square, is provided through which they are charged, and a door about 5 feet wide and 6 feet high at the bottom through which the charcoal is removed, also an opening at the bottom on one side which communicates by a short chimney with a smoke main. They are also provided with a small hole about one foot in diameter in the center of the dome.

These kilns are filled with wood to the top, the wood being piled as uniformly and smoothly as possible with a stack up through the center. Fire is started at the base of the stack and the large top and side openings are then closed with thin steel plate doors and luted air-tight. The fire is permitted to burn very slowly with natural draft through the hole in the top of the dome for a few hours. This opening is then closed and the kiln is connected with the smoke main, on which suction is maintained by a system of fans. The draft is carefully regulated, the bottom row of holes being used first and the higher ones later in the operation. At the end of about 10 days most of the volatile matter is distilled off, then the kiln is disconnected from the main, the draft holes are sealed up airtight and the kiln is allowed to stand for a period of 10 to 20 days in order to cool the charcoal well below the ignition point.

Charcoal is extremely inflammable. It will ignite spontaneously even at atmospheric temperature under certain conditions; presumably, on account of its very great power of absorbing gases, the action would seem to be not unlike that of spongy platinum in igniting hydrogen.

The situation is further complicated by the fact that the use of water to extinguish fires in this fuel is highly objectionable. Charcoal absorbs vast quantities of water which appear not to be driven off except at a very high temperature so that the charcoal seems to descend almost to the hearth of the furnace still containing water and the effect of such water is exactly the same as if introduced at the tuyeres.

On this account charcoal must be cooled with great thoroughness before exposure to the air.

After the coal has been properly cooled all the doors are opened and the buggies which are to deliver the charcoal to the furnace are taken into the kiln and there loaded.

The shrinkage during the charring operation is very great, about 50 per cent. so that while the kilns are filled with wood they are only about half full of charcoal when opened.

The smoke drawn off through the mains by the fans as above described is subjected to cooling and scrubbing whereby its volatile constituents are removed and recovered, but in a very dilute condition because of the great quantity of water vapor in the smoke which comes from the water of the wood, because thoroughly seasoned wood contains about 50 per cent water. Of the other 50 per cent about one-half is charcoal and one-half volatile matter. Unseasoned wood contains much more water than this.

The valuable by-products in the case of wood are methyl alcohol, acetic acid, and wood tar. The latter is not so valuable industrially as coal tar and for that reason is commonly used only as a fuel to aid in the distillation process. The wood alcohol is obtained, first, in the form of a 2½ per cent solution. This is rectified by distillation up to 95 per cent or more. It is perfectly possible to make it chemically pure by proper distillation and a little chemical treatment.

The acetic acid cannot be recovered by distillation because its boiling point is too close to that of water and the two substances dissolve one another thoroughly in all proportions. The acetic acid is therefore recovered by neutralizing it with a mineral base, commonly lime, which forms a soluble but non-volatile acetate. This is recovered by evaporation of the water.

The Retort Charcoal Process.—This process is very similar

to the by-product coking process, but on account of the lower temperature the retort is built of steel instead of fire brick. It consists of a steel chamber, roughly, 6 feet wide by 8 feet high and 50 feet long, surrounded with a brick setting and provided with a fire box under each end. A stack is provided to draw the products of combustion from the fire boxes around the retort. These retorts are charged with eight cords of wood on four-wheel trucks built of steel slats. The retort is closed at both ends by air-tight steel doors and is provided with an outlet for the volatile products of distillation.

After charging, the doors at both ends are closed and fires built in the fire-boxes. The gas outlet is connected to a water-cooled condenser and all the volatile products are condensed therein. The non-volatile gases are highly combustible and are led back under the retort to assist in supplying the heat it requires. In about 20 to 24 hours the process is complete, the doors are opened, the trucks containing the charcoal are withdrawn and immediately replaced by others containing a fresh charge of wood. The doors are then closed again and the process repeated.

The trucks containing the charcoal are drawn as rapidly as possible into the coolers which are steel chambers exactly like the retorts in general shape and size, but more lightly constructed and set out of doors so as to get the maximum cooling effect. There are two of these to each retort set in line with it and separated from it and from each other by a short open space. The charcoal is put first into the nearest one and there allowed to cool for twenty-four hours, and at the next drawing of the retort it is moved into the second one, the fresh charge being put into the first cooler.

On account of the rapidity with which charcoal burns on exposure to the air, the transfers, particularly the one from the retort to the first cooler when the charcoal is very hot, are made as quickly as possible and the air-tight doors of the coolers are immediately closed so that no air can reach the charcoal. Even with these precautions and with an additional cooling of 24 hours in open air, fires by spontaneous combustion of the coal are not infrequent.

This process gives much better yields of by-products than the kiln process. The latter in good practice yields four to five gallons of refined methyl alcohol and 85 to 100 pounds of acetate of lime per cord of wood. The retort process yields in good practice nine to ten gallons of refined alcohol and 170 to 220 pounds of acetate of lime.

The reason that this great superiority in the yield of valuable by-products has not forced the introduction of the retort more rapidly is that it suffers under two disadvantages. First the cost of plant per cord of wood carbonized is about three times that of the kiln plant and for large capacity it runs into such a heavy expenditure as to put these plants out of the reach of any but relatively large and wealthy corporations. Second, the maintenance charges of the retorts and plant generally is very heavy, the bottoms are liable to be burned out by a few hours careless firing and the retort as a whole may be warped so as to be worthless without heavy repairs unless it is managed with great skill.

Those who have installed and managed retort plants with skill and intelligent care think that they constitute the only method of wood distillation worthy of consideration, while those who have operated them carelessly and ignorantly declare with equal positiveness that from a commercial point of view they are inferior to kilns, in spite of the much lower yields of the latter, which are undoubtedly very much easier to manage.

The yield of charcoal in the two cases is not very different, about 50 bushels of 20 pounds to the bushel per cord of wood.

It is probable, however, that this comparison is not quite fair to the retort because it can and does drive off the volatile products more thoroughly than the kiln can do without waste of coal by combustion, so that while the weight of charcoal is about the same in two cases the fixed carbon in the retort coal is greater than that of the kiln coal.

This is partly compensated in the opinion of some operators

by the fact that the retort coal is more brittle and therefore breaks up worse in the furnace than the kiln coal. Such personal experience as I have had tends to contradict this view as I have found that better fuel economy was obtained with retort than with kiln coal.

In a large-scale charcoal operation the difficulty of obtaining an adequate and steady supply of cord wood is very great. The yield of cord wood per acre of timber land is from five to ten cords when only the material unsuitable for saw timber is taken and 40 or 50 cords per acre when all the timber is taken. The former condition is the more common and for this reason areas of several square miles must be cut over per year to supply a large charcoal operation. This makes many transportation difficulties.

In the earlier days of the industry small batteries of kilns were built close to the timber supply and the charcoal was shipped from these to the furnace, no attempt being made to recover the by-products. When commercial conditions forced the abandonment of this wasteful procedure the small batteries of kilns in the woods were abandoned and one large central battery was built preferably at the furnace, because the labor of handling the finished charcoal and its consequent breakage were reduced to a minimum, and, what is equally important, the waste gases of the furnace became available to supply steam for the great amount of distillation involved in the recovery of the by-products.

In those cases in which retorts are installed the same considerations hold and such plants are located at sites where timber may be shipped to them by rail from a large territory. Very frequently these also are located at the furnaces which consume their coal, but not necessarily because owing to the greater profit on their by-products they can sell their coal more cheaply than can the kiln plants and can, therefore, deliver it over considerable distances in competition with kilns.

Methyl alcohol is worth at the plant from 25 cents to about 40 cents per gallon, depending on the location and the perfection of the rectification.

Acetate of lime is worth from 2 to 2½ cents per pound at the plant, depending upon the state of the market.

The cost of recovery with properly designed plants is low and therefore the income received from these by-products has a very important commercial bearing on charcoal production. In fact, the charcoal is the real by-product at many of these plants and the furnace is built simply as the most convenient means to consume this fuel.

Charcoal as a Furnace Fuel

Charcoal is almost an ideal furnace fuel. It is nearly free from sulphur, having only a few hundredths of 1 per cent as against approximately 1 per cent in coke and about 1 per cent of ash against about 10 per cent for coke. Moreover, this ash consists very largely of lime and alkalis, so that it supplies a part of the flux required for the gangue of the ore instead of being largely silica and requiring to be fluxed with additional bases as is the case with coke.

Charcoal has a highly porous structure and can therefore be burned at a rapid rate per square foot of hearth area. This fuel, however, is subject to one drawback. It has not quite the physical strength of coke and would perhaps be unable to stand the conditions existing in a large furnace. On the other hand, owing to the difficulties of wood supply above mentioned, charcoal furnaces are always small as compared with coke furnaces, and therefore the necessity of its having to withstand the pressures customary with large furnaces does not arise.

Charcoal has one bad characteristic not widely appreciated. In spite of its low ash (about 1 per cent) its percentage in fixed carbon is very low, only about 60 per cent, or less in kiln coal. The remainder, about 30 per cent, is volatile matter which is not driven off at any heat which it is permissible to use in charcoal distillation. This volatile matter is about one-half hydrogen and of the remainder a large portion consists of gases of high reducing power, methane and CO among the number.

Because of these valuable characteristics, the fuel per ton of iron produced is less with charcoal than with coke. These matters will be more fully discussed in a subsequent article.

Anthracite Coal

Anthracite coal has about the same chemical analysis as good coke for the reason that it is simply bituminous coal whose volatile matter has been distilled off under conditions of heat and pressure arising from the geological movements of the earth's crust. The pressure has closed up the cells as they were formed and anthracite instead of being cellular like coke is in fact the most close grained of any variety of coal. It only occurs in quantities of commercial importance in a small district in eastern Pennsylvania in the United States, though small deposits are known elsewhere, and deposits of much importance occur in Wales in Great Britain.

Owing to its abundance and cheapness in the early days of the industrial development of the United States it was the principal fuel used in the blast furnace until about 1860, when the development of the coking process began to furnish a rival fuel. As the magnetite mines of New Jersey and the limonite mines of eastern Pennsylvania are in close proximity to the anthracite coal fields, a great iron industry was developed based upon these materials.

But the increasing use of anthracite as a domestic fuel enhanced its value, while the development of the blast furnace proved coke to be the better fuel of the two for its purpose. The anthracite iron industry, therefore, declined, and there are only a few small furnaces using this fuel exclusively, though in the eastern Pennsylvania district, where it is cheaper than good coke, a certain amount of it continues to be used in admixture with coke to lower the cost of the fuel charge.

Anthracite as a Furnace Fuel

Anthracite has three disadvantages as a blast-furnace fuel when compared with coke.

First, it is entirely lacking in cellular structure and is simultaneously of much greater density, so that the ratio of surface exposed per unit of weight is only a small fraction of what it is in the case of coke and only a proportionately small amount can be burned per square foot of hearth area. This means that the output of a given furnace is necessarily much smaller when supplied with anthracite fuel than with coke.

Second, it has a tendency to spall or decrepitate under the action of heat. As a consequence of this spalling action the interstices through the fuel which form the principal passageway for the gases are greatly obstructed and the pressure required to drive the gases through the furnace is much increased. Moreover, this spalling seems to produce a tendency to scaffold the furnace, which will be further discussed in a later chapter.

Third, the much greater density of the fuel as compared with coke reduces the volume of the fuel charge very greatly. This increases the density of the charge as a whole and permits a longer time for the passage of the ore through a furnace of given size. But, for the reason already mentioned above this advantage is useless in increasing the output. On the other hand, the increased density of the charge means increased resistance to the blast.

As a consequence of these three conditions anthracite furnaces are characterized by slow driving and small outputs, high pressure, and a strong tendency to become scaffolded and work irregularly. These are the technical conditions, which, joined with the commercial considerations above outlined, have caused the decline of anthracite as a blast-furnace fuel.

Raw Coal

Certain coals, called "bituminous" for lack of a more descriptive name, while they contain a normal amount of volatile matter, are largely free from tarry ingredients. These are known as "dry" bituminous coals. On account of their relative freedom from tarry matter these fuels fall within the fundamental requirements of a blast-furnace fuel and in the past were quite extensively used as such.

But the development of the coke blast furnace has caused a retrogression in the use of this fuel very similar to that in the case of anthracite. In certain districts where these coals may be obtained cheaply and for the production of certain special irons they are still used in admixture with coke. But so far as known to me there are no furnaces now running in the United States on raw coal fuel exclusively.

Raw Coal as a Furnace Fuel

A lack of porosity and the consequent slow rate of combustion of these coals are presumably the principal reasons for the decline in the use of this fuel. Moreover, though these coals are relatively dry they are not entirely free from tarry matter, and as but a small quantity of tar is required to obstruct the passage of the gas, this also has tended to reduce the rate of driving possible with this fuel. Moreover, the tendency for the furnace to scaffold and work irregularly even with a small amount of tarry matter present in these coals is pronounced. Their use is, therefore, generally confined to a district in southern Ohio, where they occur and are applied in admixture with coke to the production of an iron containing 15 to 20 per cent of silicon and known as ferrosilicon.

The characteristics of these fuels have modified furnace practice, and this in turn has modified equipment and product, so that the classification of iron by the fuels used in their production is fully justified. But as will be seen from what has already been said the only units in the classification which have preserved their individuality unchanged are coke and charcoal, the only two varieties of iron now in common commercial use.

Additional Requirements

We have given a definition of the fundamental requirements of the blast furnace. To these may be added two more, which are necessities in the case of more than 99 per cent of the iron produced.

First, the slag must be sufficiently fusible to run freely from the hearth at a temperature easily attainable.

Second, the blast must be warmed before being delivered to the furnace.

The single exception to these two conditions occurs in the case of furnaces making cold-blast charcoal iron of which probably not over twenty or twenty-five thousand tons per year are produced out of a total of some thirty-five million tons in the United States. In this case the temperature attainable is very low and in consequence the slag is not heated hot enough to run freely and has to be removed from the furnace with the assistance of mechanical means such as slag-hooks, etc., or, if it comes from the furnace without assistance, it chills before going more than a few feet and is accordingly troublesome to remove.

Leaving out of account this very small percentage, all blast furnaces are blown with blast heated from 500 deg. to 1400 deg. F. and their slag is heated to a degree sufficient to enable it to run from the furnace without assistance and far enough to reach some convenient means of ultimate disposal. The heated blast was introduced by the Englishman, Neilson, about 100 years ago, and it has made possible the modern development of the blast-furnace because the velocity of combustion of the fuel is so greatly augmented by preheating the blast, and the continuity of the operation has so increased by raising the molten iron and slag well above their fusion points, that outputs have been increased almost a hundred fold since.

The iron blast furnace is now probably the largest industrial apparatus in use, not only in size of plant, but in the quantity of raw materials consumed and output. Furnaces have often been built which produced 600 tons of iron per day, but modern practice tends to slightly smaller units, furnaces now being built only for an output of about 500 tons per day; but this is maintained as an average output for months and years together.

To produce a ton of iron under favorable conditions requires about 2 tons of ore, $\frac{1}{2}$ ton of flux, 1 ton of fuel and about 4 tons of blast; greater quantities are required when the con-

ditions are less favorable. With each ton of iron is also produced about 6 tons of waste gases and about 1 ton of slag, this, as well as the iron, being handled in the liquid condition. For a 500-ton furnace we have, therefore, to handle every twenty-four hours about 1700 tons of solids, 750 tons of white-hot liquid and 5000 tons of hot gases (blast and waste gases together).

To do this on an industrial scale and with a labor cost amounting in good practice to less than 75 cents per ton a great plant is necessary. Such a plant must contain the following parts: First: The shaft furnace itself through which the entire mass of material just mentioned passes, the solids passing down to the liquid condition at the bottom and the gases upward through them. Second: Means for handling the raw materials, ore, fuel, flux, proportioning them properly and delivering them in certain accurate and positive relations into the top of the furnace. Third: Means for raising the atmospheric air to a pressure sufficient to force it through the furnace in the quantity desired. Fourth: Means for heating this supply of air uniformly and regularly to the temperature desired. Fifth: Means for handling the molten iron on its discharge from the furnace and turning it into a commercially salable product. Sixth: Means for removing the slag from the vicinity of the furnace. Seventh: Means for removing the gases from the top of the furnace and (as these gases are a valuable fuel) for leading them to those portions of the plant where they can supply the energy for compressing the blast and heating it. There is an eighth requirement not embodied in our fundamental definition, but which has become an absolute necessity notwithstanding; a supply of water for cooling certain parts of the furnace and preventing the intense heat inside from destroying the structure.

Available Alkali in Lime

The South African Engineering Standards Committee, supported by six South African scientific and engineering societies, has just published Pamphlet No. 3 of its Chemical Sub-committee, entitled: "Lime, Standardization of Methods of Sampling and Determination of Available Alkalinity" (Price 1s. net, South African Engineering Standards Committee, P. O. Box 1176, Johannesburg, S. Africa).

The pamphlet deals with the uses of lime in the manufacture, transport, and handling, describes methods of sampling unslaked lump lime, crushed lime, and slaked lime, and finally gives the following standard method for the determinations of "available alkalinity" (CaO).

"The sample as delivered for analysis is contained in an air-tight vessel, having been passed through a 30-mesh sieve. It is crushed with a Wedgwood mortar and pestle and the whole passed through a 60-mesh sieve, this operation being performed as quickly as possible. It is then placed in a clean, dry, wide-mouthed bottle fitted with a tight fitting dry glass stopper so as to prevent any access of air. Two grams of this are carefully weighed out and agitated with 1 liter of a 2 per cent cane sugar solution (or 1 gram with $\frac{1}{2}$ liter of 2 per cent sugar solution). If a shaking machine be available, 2 hours' continuous agitation should be given; if not 6 hours' intermittent agitation, every care being taken to prevent coagulation of the lime, in order that the lime and solution may be brought into the most intimate contact during this period. When the agitation is finished the solution is filtered as quickly as possible and aliquot portions titrated with N to or N₂ acid, using rosolic acid as indicator, avoiding delay so as to obviate undue exposure to the atmosphere.

"The distilled water used in the above determination must be made neutral to rosolic acid to counteract the presence of dissolved CO₂."

The metal production of Alaska for 1912 was valued at more than \$22,000,000, divided approximately as follows: Gold, \$17,000,000; silver, \$317,000; copper, \$4,800,000. The development of the coal still awaits a settled policy on the part of the government.

The Radium Institute St. Joachimsthal

By Joh. Hårdén

The medical treatment of diseases by means of radium and radioactive substances has now gained such importance that the great general interest shown both by laymen and scientists seems quite justified. It will be remembered that the Curie's discovered the new element radium in a mineral called pitchblende, which they obtained from St. Joachimsthal about 18 years ago.

During a recent visit to Bohemia the author took the opportunity to visit the quaint old town in the Bohemian mountains, and in the following lines a short report of its important Radium Institute and Factory will be given.

St. Joachimsthal is an ancient township, situated some 20 miles to the northwest of Carlsbad; it was once famous for its silver-bearing mines and smelting works, records of which are in existence as far back as 1515, when the town counted "over 1200 houses" and the smelting works are doubtless still older. By the way, it may be of interest to record, that it is assumed that the name "Thaler" the former German coin-unit has its origin from the silver gained in St. Joachimsthal.

As the mines were exhausted as regards the silver during the following 300 years, the importance of the town as a mining and smelting center diminished; also the war times proved disastrous to the industry, and to-day there are scarcely even ruins of the old silver smelting works left, while the population has dwindled down to only a couple of thousand, barely thriving on lace making, paper box manufacturing, and some revival of mining.

As the yield of silver diminished, the leading men naturally turned their attention to other sources of income from the minerals available. This part of Bohemia is remarkable for its great variety in various minerals; so, for instance, at Schlaggenwald, in about equal distance southwest from Carlsbad, the author found a tin and tungsten mine in full swing on the remainders of an old tin smelting industry and on the old dump heaps a veritable treasure for the geological student was in open daylight. The chemist stated that no less than 95 different minerals, more or less interesting, could be gathered here. Indeed, a large railway viaduct, built from the material taken from the old dump heap, showed a curious admixture of colors, copper pyrites, turmalin, small amethyst and garnet crystals appeared on the fracture of the hewn stone.

In the year 1853, the chemist Paterna, appointed by the Government to investigate the possibilities of St. Joachimsthal, succeeded in finding a practical method for the manufacture of uranium dyes, an industry which proved very remunerative; for instance, in 1886 the factory sold over 11,000 kilograms of these dyes for about half a million marks.

The element uranium was, according to records, discovered by Klaproth in the year 1789 and the metal itself was isolated by Peligot, 1840, both from the uranium-pitchblende from St. Joachimsthal. This mineral consists in the main of uranium oxyduloxyd, together with quartz, iron, cobalt, lead, bismuth, arsenic, calcium, magnesium, copper, vanadium, selenium, and in particular the radioactive substances "radium," polonium and actinium.

The discovery of these radioactive elements by Mr. and Mrs. Curie, in the laboratory of Professor Becquerel in Paris in 1896 is too well known to be further described here. The author had the privilege of being present when the late Professor Curie held his lecture on his far-reaching discovery, assisted by his skilled wife, Madame Curie, on the occasion of their receiving in joint the Nobel Prize in Stockholm in 1905, and it is safe to say that the unostentatious lecture, accompanied by a number of startling experiments, showing the apparent miracle of the birth of a new element, made a lasting impression upon those present.

Mr. and Mrs. Curie had, upon request of Professor Becquerel, commenced the investigation of radioactivity of uranium salts derived from Joachimsthal, and as they soon found that a new and unknown element was present, they obtained a larger

quantity of the slimes remaining from the manufacture of these salts. In this they found a more considerable source of the costly substances, and this slime is even to-day the chief raw material for the recovery of the radium.

The isolation of the new element is rather simple in principle, being a process of dissolving and repeated fractional crystallization, until, at least in Joachimsthal, the pure radium carbonate remains, which is subsequently transformed into bromide or chloride as the demand calls for. But the practical side of the problem is less simple, inasmuch as the element is so rare, that several tons of radium-bearing slimes must be treated in order to extract only a few milligrams, and this, of course, makes the process rather costly.

In addition to the radium salts are also the radioactive waters to be counted. These waters contain the so-called emanation, which is assumed to be a disintegration substance of radium and as such is found to be highly radioactive.

A unit for the radioactivity has been agreed on by the scientists, similar to the units for electric potential, the volt, and the ampere for the current. The unit for radioactivity is called "Mache" after Prof. Heinrich Mache in Vienna, who together with S. Meyer carried out important work on this subject. They soon found that some kind of unit was necessary, and recommended the use of the units of the "Absolute" or the "C. G. S." system. This unit is represented by the strength of the saturation current, which is caused to pass through by the emanation arising from one liter of the water to be tested; it is expressed in electrostatic units. As this, however, would give a too low figure for practical use, it is multiplied by 1000 or 10^3 , and it is this quantity which is generally called "One Mache."

To give an idea of the strength of radium emanation in various springs, counted in Mache units per liter of water, the following table may serve as a comparison:

St. Joachimsthal.....	600 units.
Gastein	54 to 155 units.
Baden-Baden	24 to 126 "
Bath (uncertain).....	20 to 86 "
Carlsbad	1.4 to 50.2 "
Muenster St.....	13.5 to 23.4 "
Teplitz	3 to 8.7 "

These waters are either used internally, or chiefly, as baths, according to the prescription of the doctors. It is, of course, outside the scope of this report to enter into a description of the various cases of which these curative waters of high strength are prescribed; may it be sufficient to mention that very excellent results have been obtained in cases of nervous breakdowns, ischiatic, etc., while the direct application of radium, as well known, is rendering important relief in cases of cancer, ulcers, etc.

The active waters at St. Joachimsthal are conducted direct from the mine, at great depths, into the curehouse, so as to render their curative action as soon as possible to the place of application.

The waters may also be stored and sent to distant places in suitable vessels; it should be borne in mind, however, that the emanation is gradually giving out when so stored, the waters losing their strength at the rate of one-half in 3½ days and this goes on regularly, so that a water containing originally 600 units, has only a strength of 300 after 3½ days, 150 units after 7½ days, 75 units after about 11 days and so forth.

The emanation substance itself is a material of greatest interest physically: it appears in form of a gas which may be isolated and weighed; it is soluble in water, and is capable of passing through porous and gaseous substances. It is also constantly developing energy in form of heat, in fact, one kilogram (2.2 lbs.) of emanation is capable of developing 20,000 horse powers in one hour, if the whole of its energy could be spent accordingly. It may be condensed to a liquid at low temperatures, about 150° C, which shows all the characteristics of the gas. The spectrum is very distinct, though the gas itself is not visible, and non-inflammable. Boiling drives it out of its solution, also shaking, or if a current of air is driven through,

Glass and gems are slightly colored by the action of the emanation, water is slowly decomposed into hydrogen and oxygen, photograph plates are darkened; if strongly concentrated, it is observed to glow in the dark and to render other substances phosphorescent, it also ionizes the air, so as to conduct electricity. It is inert against practically all chemical reactions.

The emanation was discovered through its property of sending off "alpha" rays, the first range of rays given off by radium itself. It seems that the emanation is an instable substance, which as it is emanating the alpha rays, is disintegrating into a substance of lower atomic weight. The precipitate, which is accumulating with somewhat greater velocity on bodies charged with negative electricity, is again an instable substance, which is in its turn deteriorating, thereby giving off both alpha, beta and gamma rays. A product of this substance, called radium D has been specially studied by Dr. Gudzent and found to be of great importance for the medical treatment of some diseases.

Apart from this, the emanation is also splitting up into the gas helium, a well-known element, and this seems to be the lowest step of disintegration.

This gas was discovered by Jansen and Loker (according to Rutherford) in the sun's spectrum, in the year 1868. Nearly 30 years passed before this gas was identified as an earth element, and then it was only found occluded in uranium-bearing minerals, a coincidence which was rather puzzling, as the connection between uranium, radium and helium was at that time unknown; to-day, however, we understand quite well that the disintegration of the radium present in these uranium and thorium minerals had most likely been the source of the occluded helium. R. J. Moos found, when pitchblende residues from St. Joachimsthal were pulverized and treated in vacuum at high temperature, that 1.17% of helium could be extracted.

The emanation, however, is found in smaller or greater quantities, practically all over the earth, since radium is present almost everywhere, though in minimal quantities. But only a few places can boast of such riches in radioactive materials as St. Joachimsthal; the only other source of importance in Europe is the Trenwith mine at St. Ives, Cornwall, where considerable quantities of radium are mined daily, for subsequent purification for the Radium Institute in London.

Reports are reaching us from time to time claiming discoveries of new deposits both in the United States and in Australia; but to the author's knowledge, the bulk of the world's supply of this valuable agent for fighting disease is still prepared from the minerals from St. Ives and St. Joachimsthal. The daily papers are just now mentioning a project, according to which the rich mines at St. Joachimsthal are to be bought up by an English concern, for the stated sum of £700,000. This price may appear high, at a first glance, but if we consider the extraordinarily high price of the radium, amounting to many pounds sterling per milligram (the prices fluctuating with the market) caused by the rarity of the substance and the tedious labor of extracting it from the ores, one wagonload of crude ore only yielding a few milligrams of radium, it is easier to understand why the price should be so high.

The radium factory at St. Joachimsthal is only a small one, compared with the world's demand, but it is no doubt capable of considerable developments, similar to those of the British Radium Institute; given time and means of carrying out all modern improvements.

Luton, England.

JOH. HÄRDÉN.

The Engineers' Society of Western Pennsylvania has organized a metallurgical and mining section, the membership including chiefly the metallurgists of steel mills and allied industries in the Pittsburgh district and mining engineers throughout the Western Pennsylvania, West Virginia, and Ohio coal mining districts. At the first meeting of the section Mr. B. G. Klugh lectured on sintering processes for iron-bearing materials. Besides the metallurgical and mining section, the Society has a mechanical and a structural section.

The Art of Enameling or the Coating of Steel and Iron with Glass*

By Raymond F. Nailler

Before considering the technology of any particular type of enamel it will be well to consider the meaning conveyed by the word enamel. If the present-day enamel be briefly and scientifically designated as a borosodium-potassium-aluminum silicate generally colored by metallic oxides, then the following definition given by Popelin deserves to be quoted since this, notwithstanding its length, may be described as most clear and comprehensive: "Enamel is a glass fusible at a low temperature and usually compounded of borates and silicates. This mixture, originally colorless, combines with the greatest ease with metallic oxides under the influence of a pyrotechnic operation, thereby acquiring various colors according to the nature of the oxide which the enameler can vary at will."

The uses to which the general class of ceramic compounds known as enamel is put are varied. We have noted that enamels are used for artistic or decorative purposes. Parallel to this use we may classify a far more important type of enamels as "commercial," and it is with this type of enamel that we are to concern ourselves.

The first class of commercial enamels is that used on cast iron, and this classification includes the field of sanitary equipment, together with various forms of cast-iron kettles and similar pieces of engineering apparatus.

The enameling of sheet steel may be taken as the second class of commercial enamels. The most familiar division of this class includes cooking-ware enamels. The various forms of so-called agate and granite ware as well as the single colors are made by enameling sheet-steel forms.

The second class of steel enameling which has more recently come into industrial importance is the manufacture of enamel signs, and the third field includes the manufacture of heavy equipment for large-scale food preparation, the dairy industries and general chemical operations. The operations in this last case may take the form of tanks, kettles, evaporators, pipes, etc.

In outlining the technology of the enameling industries as a whole we may include to a certain extent all of the above commercial classes, certain details of which are varied to meet the conditions of cast-iron and various types of sheet-steel products. The outline following, however, is characteristic of all classes.

The first step to be considered is the preparation of the enamel. The purity of the raw materials to be used in the compounding of an enamel must be certain, and in the cases where materials can vary in strength the actual analysis of the substance must be known to assure proper results. In many cases the secret of an enamel lies in its formula and so the compounding of the batches is very carefully guarded, only a person of responsibility having charge of weighing the ingredients. The various materials are generally kept in bins which are numbered, the person in charge of the mixing having the formula stated in terms of these numbers. The properly weighed batch is thoroughly mixed, this being accomplished either by shoveling carefully on a specially prepared floor or by mechanical mixing by means of a rotating agitator.

The product of the mixing room is taken to the smelter in which the various ingredients are fused together in the form of a mass having the characteristics of glass. The furnace in which this operation takes place is a special reverberatory furnace similar to that used in the puddling process in the manufacture of wrought iron. The coal is placed on a grate at the front end of the furnace and the burning gases pass over the bridge wall, strike the roof, and are deflected against the batch. The first change which may be noted in the smelter is the driving off of the water from the borax which produces a swelling up of the mixture. The fluxing materials then melt down and as the heating is continued, dissolve the

*Extracts from a paper published in the October issue of the *Journal of the American Society of Mechanical Engineers*. With the exception of the historical introduction the paper is here given practically in full.

more refractory constituents. If the batch is properly stirred and the temperature of the furnace carefully regulated, the product of this operation is a clear transparent glass containing no particles of unfused material. When a test of the fusion comes up to this standard the furnace is tapped and as the liquid glass flows into a tank of cold water it is broken up by the chilling action of the water, the result being a thorough granulation of the product. The material thus prepared is known as "frit."

The next step in the preparation of the enamel involves grinding the frit to a fineness at which it can be applied to the surface of the metal. The mill here used is the ordinary pebble mill lined with porcelain brick and containing very hard flint pebbles. In the cast-iron industry two general processes are used, the dry and the wet, in sheet-steel work only the wet. In the first case the frit is ground dry, the pulverized enamel being sprinkled on the hot cast-iron piece. In the wet process a certain percentage of pure white clay is placed in the mill with the frit and a certain definite amount of distilled water. When certain grades of enamel are to be manufactured various compounds are also added here to aid in producing desired colors, gloss and opacity. An instance of this is the addition of tin oxide in the mill in the production of white cooking utensil enamels. The fineness to which the enamel is ground depends upon the particular use to which it is to be put, manner of application and other factors. In the wet process the essential feature is that it be fine enough to remain suspended in the water assisted by the clay and the so-called "vehicles" mentioned below.

At this point we may well take up the consideration of the construction and preparation of the material to which the enamel is to be applied. Cast-iron enamels are applied to castings of the desired shape. In the case of cooking ware enamels the shapes are constructed by pressing and spinning. For heavier equipment such as jacketed kettles the apparatus is constructed by riveting or, preferably, welding the sheet steel in the desired form. The selection of the steel with a view to its chemical analysis is of prime importance. A reliable specification reads as follows:

Sulphur	below	0.040
Phosphorous	below	0.030
Manganese	about	0.40
Silicon	about	0.010
Carbon	about	0.10

The steel must necessarily be free from laminations or other mechanical imperfections.

With reference to the construction of heavy apparatus the sheet ranges from 3/16 inch to 3/4 inch in thickness. Two general methods of construction are in use: The first involves the formation of unit sections fitted with flanges. These sections are enameled separately, bolted together at the flanges and the desired apparatus so constructed. The preferable practice, however, is to construct the apparatus in one piece by means of autogenous welding, thus avoiding the use of gaskets or other packing materials in erection.

That the enamel may properly adhere to the surface of the metal the latter must be free from dirt or scale, and in the case of welded joints the welds must have a preliminary grinding to reduce the roughness. The entire surface of the apparatus is then cleaned by pickling or sand blasting, the latter process being altogether used in cleaning large apparatus. As the crude ware leaves the sand blast it has a roughened, clean, metallic surface and is in the proper condition to receive the enamel.

Before applying the enamel to the metallic surface it is prepared by a process known as "setting up." This involves the addition of certain chemicals to the enamel as taken from the mill the function of this addition being to assist the clay in holding the enamel particles in suspension. Substances so added are termed "vehicles." At this state the enamel must be diluted with distilled water to the proper consistency for application.

In applying the enamel to the metallic surface three general methods are in use: The first, applicable to small pieces only, is known as "dipping," the piece being dipped into the enamel, the excess of which is shaken off, leaving a thin coating on the metal. The second method, known as "slushing," involves pouring the prepared enamel over the surface and allowing it to drain. The third method, which is the principal one used on larger apparatus, involves spraying the finely ground enamel on the metallic surface by means of the compressed air atomizer.

Preliminary to the consideration of firing the enamel we may well review the types of furnaces in use. The first type is known as the muffle furnace and involves the use of a large fire clay oven externally heated by means of coal, gas or other fuel. The apparatus to be fired is placed on suitable supports in this muffle. The other type of furnace is known as the direct fire furnace, in which the heat from the fire is taken up by the walls of the firing chamber and radiated to the apparatus placed within the chamber on suitable racks. This general class of furnace has two special divisions in that, on the one hand, the piece is rotated within the furnace, while on the other hand, the piece is allowed to remain stationary. For small work the muffle is in general use, but for the production of large apparatus the direct-fire furnace is necessary. At first thought, the muffle furnace may be considered to have an advantage in that the products of combustion together with the dust from the fire cannot come in contact with the enamel. Again it may seem that a more even heat can be realized in the muffle. On the other hand, with a properly designed direct-fire furnace in which the combustion is complete before the gases reach the firing chamber no trouble is experienced, due to their presence or to the dust from the fire. The use of natural gas further does away with this latter possibility.

In a furnace for firing smaller ware the charging mechanism is a fairly simple matter, it being necessary merely to place the material in the furnace by means of a small fork operated by hand or mechanically. But in the manufacture of engineering apparatus where a single piece may weigh 3000 to 4000 pounds, it is necessary to have a large mechanical charging machine on which the piece may be placed outside the furnace, the arm of the machine then properly placing it in the furnace. The general design of such a machine suggests the charger used in open-hearth practice. The apparatus to which the enamel has been properly applied is placed in the furnace, which is maintained at the proper temperature. This temperature varies with the nature of the enamel and in cases of high-silicon acid-proof enamels reaches in the neighborhood of 2500 deg. F. The control of the burning is made possible by the changes which occur in appearance of the enameled surface as fusion takes place. At first the fine particles of enamel begin to fuse together and a general blister condition exists, giving the surface a very dull appearance. But as the enamel matures this dull appearance gives way to a bright glass, which, when properly developed over the inside surface, is indication that the piece should be withdrawn from the furnace. The time required to burn a piece properly depends upon the temperature of furnace, thickness of metal, and nature of enamel.

Nothing has been said so far as to the composition of the enamel or of the number of coats applied. In general there are two kinds of enamel, known as ground coats and cover coats. The former serves as a bond between the enamel and the steel, and the latter serves to build up the body of the enamel and presents the finished surface.

In the ground coat color is no object. Its composition is such as to render it adherent and strong. In addition to the ordinary components cobalt oxide seems to be essential to the production of adherence. The explanation of this is debatable.

The cover coat is the one which forms the major part of the enamel and if definite color, opacity, etc., are objects, the necessary ingredients for their production are introduced here, assisted by mill additions as noted above.

In the manufacture of acid-proof enamel the cover coat is essentially a high silicate and must be free from any metallic oxides, such as oxide of tin, lead, iron, etc.

The piece to be enamelled receives one ground coat which is burnt well into the steel at a high temperature. The cover coats may be one or two in number for ordinary enameling, but should be at least triple for acid-proof work. In the production of acid-proof apparatus the use of a ground coat or cover coat is now eliminated and the same material is employed for both which in place of being an enamel, as commonly termed, is in reality a boro-silicon glass and without the use of any metallic oxides in its compounding.

It is very interesting to note the chemical changes which take place in the various stages of the production of the finished enamel. Avoiding as far as possible deep technicality, they may be summed up as below. As a starting point, let us select a cover coat formula used in the production of a "dark blue" cooking ware enamel.

Feldspar, lb.	120	Salt peter, lb.	7
Quartz, lb.	72	Oxide of Cobalt, lb.	7 1/2
Borax, lb.	80	Oxide of Manganese, lb.	1
Cryolite, lb.	30	Clay in the mill, per cent.	1

First considering the behavior of each of the constituents when heated we can later clearly note the general reactions which take place when they are fused together in the smelter.

Feldspar varies greatly in composition but as commonly used in enamels is an aluminium-sodium-potassium silicate of approximately the following analysis:

	Per Cent
Silica (SiO ₂)	70
Alumina (Al ₂ O ₃)	17
Soda (Na ₂ O)	7
Potash (K ₂ O)	6

On smelting, none of these constituents vaporize, so we may consider them all later in the smelting of the above enamel.

Quartz introduced as a pure glass sand contains practically 100 per cent silica (SiO₂).

Borax is chemically known as sodium tetra-borate, and in the crystalline form as used has a certain definite amount of so-called water of crystallization which must be taken into account. The formula Na₂B₄O₇·10H₂O is resolved into Na₂O·2B₂O₃·10H₂O. Without going through the calculations assume this to correspond approximately to the following analysis:

	Per Cent
Water (H ₂ O)	16.0
Boric Oxide (B ₂ O ₃)	37.0
Soda (Na ₂ O)	47.0

Of these the last two do not vaporize on smelting, but the water is evaporated, hence 100 lb. of borax smelts to 84 lb. of the remaining oxides.

Cryolite is a double fluoride of sodium and aluminium, the formula of which may be written Na₃AlF₆. When smelted the sodium and aluminium appear as oxides and from 100 lb. of cryolite we realize about 24 lb. of alumina (Al₂O₃), 44 lb. of soda (Na₂O) and 54 lb. of fluorine (F₂). There is some dispute as to whether or not the fluorine is vaporized. In the present discussion it matters not, hence we shall consider that the third of these three compounds is lost in smelting.

Salt peter is potassium nitrate KNO₃. When heated under the conditions of smelting it may be considered to break up into potash (K₂O) and nitrogen pentoxide (N₂O₅). The reaction is $2KNO_3 = K_2O + N_2O_5$. By calculations based on this reaction 100 lb. of salt peter yields about 47 lb. of potash (K₂O) and 53 lb. of nitrogen oxide (N₂O₅). The latter may be considered as completely vaporized.

Oxide of cobalt (CoO) may be taken as non-volatile and pure. The same may be assumed for the oxide of manganese (MnO₂).

To sum up the above as an outline of the reactions taking place in the smelter

120 lb. Feldspar gives $120 \times 0.70 = 84.0$ lb. Silica (SiO₂)

		120 × 0.17 = 20.4 lb. Alumina (Al ₂ O ₃)
		120 × 0.07 = 8.4 lb. Soda (Na ₂ O)
		120 × 0.06 = 7.2 lb. Potash (K ₂ O)
72 lb. Quartz	gives	72 × 1.00 = 72.0 lb. Silica (SiO ₂)
80 lb. Borax	gives	80 × 0.19 = 15.2 lb. Water (H ₂ O) (Vaporized)
		80 × 0.37 = 29.6 lb. Boric Oxide (B ₂ O ₃)
		80 × 0.47 = 37.6 lb. Soda (Na ₂ O)
30 lb. Cryolite	gives	30 × 0.44 = 13.2 lb. Soda (Na ₂ O)
		30 × 0.24 = 7.2 lb. Alumina (Al ₂ O ₃)
		30 × 0.54 = 16.2 lb. Fluorine (F ₂) (Vaporized)
7 lb. Salt peter	gives	7 × 0.47 = 3.3 lb. Potash (K ₂ O)
		7 × 0.53 = 3.7 lb. Nitrogen Oxide (N ₂ O ₅) (Vaporized)
7 1/2 lb. Cobalt Oxide		
	gives	7 1/2 × 1.00 = 7 1/2 lb. Cobalt Oxide (CoO)
1 lb. Manganese Oxide	gives	1 × 1.00 = 1 lb. Manganese Oxide (MnO ₂)

We shall consider that of the above the H₂O, N₂O₅ and F₂ are vaporized. This leaves for the constituents of the frit (totals of the above)

	Lb.	Per Cent
Silica (SiO ₂)	159	53.5
Alumina (Al ₂ O ₃)	27.6	7.5
Soda (Na ₂ O)	59.2	20.3
Potash (K ₂ O)	10.5	3.6
Boric Oxide (B ₂ O ₃)	29.6	10.2
Cobalt Oxide (CoO)	7.5	2.6
Manganese Oxide (MnO ₂)	1	.3

Total 291.4

The loss, theoretically, on smelting is plainly $317.5 - 291.4 = 26.1$

26.1 lb. or $\frac{26.1}{317.5} = 8.2$ per cent. (317.5 lb. original weight of batch).

The mill additions together with the agents used in "setting up" are fused with the frit as the piece is fired. By a process similar to the above we could compute the final composition of the enamel by taking these additions into consideration. The actual amount of material added in this case, however, is so slight and of such a nature that the change resulting therefrom is not sufficient to affect materially the composition of the enamel. When an addition of about 12 per cent tin oxide accompanies the clay a very significant change in the composition of the enamel is produced.

It may be well to note in passing some of the means by which various colors are produced in the enameling industries. It will be impossible to enter into great detail without taking too much time, but the mention of certain compounds in connection with the colors produced by their use will serve our purpose.

The production of a good white enamel either for casting or sheet-steel work may be said to depend, at the present time, upon the use of tin oxide. Great have been the efforts to substitute less expensive substances, such as compounds of antimony and lead. But an antimony white which looks good alone is plainly seen to be off-color when compared with a good tin oxide white.

Going to the other extreme of color, black, we encounter difficulties. There is any number of formulae for black enamels, but when the results are closely compared we find that the colors range widely through brown blacks, blue blacks, purple blacks, etc. Certain compounds of manganese and iron used together give a color approaching black. Other formulae call for the combined use of oxides of manganese, cobalt and copper. Again we find oxide of nickel added to the above three oxides.

A color much seen in enamels is blue and the use of cobalt is very satisfactory in the production of this color in various

grades of intensity. Manganese alone produces purples and violets, and in combination with cobalt gives various shades of purple-blue.

Green enamels are chiefly produced by the use of chromium oxide and copper oxide, while in some cases a mixture of copper and cobalt oxide is used.

Reds of various shades are produced by the use of red oxide of iron. In connection with it we find that tin oxide aids greatly in giving capacity and bringing out the color. In the production of brown enamels we may use ferrous chromate. Various yellows are produced by salts of cadmium, chromium, and uranium.

The more delicate shades of rose and purple are produced by the use of gold compounds. So-called "pink rose" is used in the manufacture of certain artistic enamels. Perhaps the best known gold compound used in enamel coloring is "purple of Cassius." The exact composition of this product is a question. It is made by the combined use of auric, stannous, and stannic chlorides. The color produced is also commonly called "purple of Cassius."

Before drawing this paper to a close, attention is invited to a general consideration of the future of the enameling industry. Neglecting art enameling and sign making we come to the field of steel enamels. So far as the cooking ware industry is concerned, the field is practically constant. Granting that the demand for that class of article is increasing, as the public becomes accustomed and educated to its use, there is an opposing tendency in the rapidly increasing use of aluminium ware. Exactly how these and other factors now balance would be difficult to ascertain. But aluminium is a metal and its metallic properties cannot be denied. Under certain conditions it is attacked by various substances used in the culinary arts and a contamination of the preparation is inevitable. No doubt the time will come when a high silica enamel known to be free from tin and other poisonous compounds will enter the cooking ware field. The government is becoming more and more careful in protecting the public from foods of injurious nature, and it is not too much to expect that soon it will establish more rigid restrictions relative to the ingredients entering into the manufacture of apparatus in which food is to be prepared. At such a time an enamel coming up to requirement will be free from injurious compounds and will come into great demand.

In the preparation of foods on a factory scale we find an enormous and constantly increasing demand for larger pieces of enameled steel apparatus in the form of pans, kettles, tanks, pipe, etc. There are many lines of pressure being brought to bear both by the government and public opinion which lead to the conclusion that the increasing demand for this style of apparatus is without limit. Canning and preserving factories and dairy establishments have found a large use for copper and tin in the construction of containers, vacuum pans, etc. The acids of fruit juices and vegetable pulps have a very marked action on these metals and the resulting contamination of the product is known to be of danger to the consumer. The use of an enamel containing tin, lead or other metallic oxides is but the first step in the right direction. The presence of these metallic oxides in the enamel renders it corrodible and contamination results. The solution is the use of an acid-proof enamel free from all such poisonous substances. In the milk industry a similar line of reasoning applies. Further compare the ease of maintaining sanitary conditions in a one-piece enamel-lined unit with the trouble experienced in the use of a metal container or even an enameled article made up of composite parts between which are gaskets.

Another consideration relates to the preparation of chemicals later used in food products, for instance, baking powder. Many operations connected with the manufacture of such products have been carried on in lead or other metallic pans and the resulting contamination has given no end of trouble. Acid-proof enamel is rapidly solving this problem also.

Finally consider the chemical manufacturing processes now carried on in apparatus of lead, wood and earthenware, neces-

sitated by the corrosive actions of the liquors and gases involved. This includes the pharmaceutical field which alone is a matter of great importance. In all these and many other lines the use of acid-proof enameled apparatus is rapidly finding and filling a great demand.

Not only does steel apparatus meet the demands of modern industries, but in case a cheaper product is desired and at the same time a heavier construction is permissible, acid-proof cast-iron apparatus has its field. The possibility for size and variety of construction is, of course, more limited than in the case of sheet-steel apparatus.

In view of these considerations and many others which these have called to mind, we cannot but conclude that the use of enameled apparatus has just begun and with this extension of the long known art of metal enameling, a field of great industrial possibilities both for manufacturer and user has been opened. We may not be criticised as being over optimistic when we predict that in their ultimate stage of development the enamel industries will be ranked among the greatest of commercial activities. At such a degree of development the enameling industry will in no way deserve classification among the lost arts.

The Glucose and Starch Industry in the United States

Final statistics of the manufacture of glucose and starch in the United States for 1909 are given in detail in a bulletin soon to be issued by Director Harris of the Bureau of the Census, Department of Commerce. It was prepared under the direction of W. M. Steuart, chief statistician for manufactures.

Of the 118 establishments canvassed in 1909, 93.2 per cent were engaged primarily in the manufacture of starch, but the value of products of these establishments formed only 32.5 per cent. of the total value of products reported for the combined industry. The eight establishments engaged primarily in the manufacture of glucose formed only 6.8 per cent of the total number of establishments, but the value of their products represented 67.5 per cent. of the total value of products for the industry.

The establishments in the industry as a whole in 1909 gave employment to an average of 5,827 persons, of whom 4,773 were wage earners, and paid \$4,079,722 in salaries and wages.

The capital invested amounted to \$38,866,410. The cost of materials used in the industry as a whole in 1909 was \$36,898,771, and the total value of products was \$48,790,311.

A striking feature of the industry, as indicated by the figures, lies in the fact that while the number of establishments was less in 1909 than in 1879, the value of products was more than four times as large, thus showing a greatly increased productive power for the average establishment.

As measured by the value of products, Illinois was the most important state in the industry in 1909, followed by Iowa, New Jersey and Indiana, in the order named. In the five-year period, 1904-1909, there was an increase of only two per cent. in the number of wage earners employed in the industry. On the other hand, the value of products increased 49.5 per cent. in five years. The wide discrepancy between the relative advance in value of products and in number of wage earners employed is doubtless indicative of improved processes used in manufacturing the products.

The average number of persons engaged in the glucose and starch industry during 1909 was 5,827, of whom 4,773, or 81.9 per cent., were wage earners. Nearly all the wage earners (98.1 per cent.) reported for the industry as a whole were employed in establishments where the prevailing hours of labor were sixty or more per week.

Form of Ownership

In 1909 of the total number of establishments 46.6 per cent. were under corporate ownership, as compared with 40.7 per cent. in 1904. In 1909 the value of products of these establishments represented 96.2 per cent. of the total, and in 1904 96.8 per cent.

Eleven per cent. of the establishments in the industry as a whole in 1909 manufactured products valued at \$1,000,000 or over, such establishments returning 89 per cent. of the total value of products, and 18.6 per cent. of the establishments reported products valued at \$100,000 or over, as compared with 15.7 per cent. in 1904. The average value of products per establishment in the industry as a whole increased from \$233,213 in 1904 to \$413,553 in 1909. The average number of wage earners per establishment shows an increase from 33.4 in 1904 to 40.4 in 1909. In the glucose branch of the industry the average value of products per establishment in 1909 was \$4,116,305, and the average number of wage earners 350. In the starch branch of the industry the average value of products was \$144,258, and the average number of wage earners 18.

The total expenses in the glucose and starch industry in 1909 were \$43,973,558, distributed as follows: Cost of materials, \$36,898,771, or 83.9 per cent.; wages, \$2,606,483, or 6.1 per cent.; salaries, \$1,413,239, or 3.2 per cent., and miscellaneous expenses, made up of advertising, ordinary repairs of buildings and machinery, insurance, traveling expenses and other sundry expenses, \$2,995,065, or 6.8 per cent. The unusually high proportion of the total expenses formed by cost of materials and the low proportion formed by wages is explained by the fact that the various processes in this industry are almost entirely mechanical.

Materials and Products

There was an absolute increase of \$16,149,475, or a relative increase of 49.5 per cent., from 1904 to 1909 in the total value of products for the industry as a whole, which was \$48,799,316. Corn and potatoes are the most important materials used in the manufacture of glucose and starch. Computed on the basis of 50 pounds to the bushel, the 2,240,508,915 pounds of corn used in 1909 was equivalent to 40,009,088 bushels. Computed on the basis of 60 pounds to the bushel, the 210,608,127 pounds of potatoes used was equivalent to 3,510,135 bushels. Wheat flour is also an important material in the industry. The quantity and cost of arrow root and other roots used as material are combined with that of wheat, the total for these materials in 1909 amounting to 1,940,000 pounds, valued at \$21,435.

Glucose (including glucose syrups) is the most important product when measured by value. The value of this product as reported for 1909 was \$17,922,514, or 36.7 per cent. of the total value of products for the combined industry. The increase in the value of glucose from 1904 to 1909 amounted to \$5,509,898, or 45.1 per cent. Glucose was manufactured in only four states—Illinois, Indiana, Iowa and New Jersey. Corn oil, a product obtained chiefly in the process of manufacturing glucose, increased in value \$1,638,302, or 140.7 per cent. The increase in the value of stock feed amounted to \$1,567,480, or 35.3 per cent.

A total of 677,535,047 pounds of starch of all kinds, valued at \$17,514,823, was manufactured in the United States in 1909, as compared with 356,695,335 pounds, valued at \$10,027,538 produced in 1904. The production of corn starch increased 327,684,552 pounds, or 105.3 per cent., in quantity, and \$7,084,466 or 79.8 per cent in value in the five-year period. This large increase was due entirely to increased domestic consumption. Starch made from wheat and roots decreased 4,008,255 pounds or 22.5 per cent in quantity, and \$395,724, or 35.2 per cent, in value in the five years.

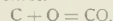
The quantity of potato starch decreased from 27,700,400 pounds in 1904 to 24,873,415 pounds in 1909, or 10.2 per cent., while the value decreased \$101,457, or 11 per cent. Potato starch was manufactured in 1909 in Colorado, Maine, Minnesota and Wisconsin, but the principal point of activity was Aroostook County, Maine, where 61 factories reported the consumption of 171,283,746 pounds, or 81.3 per cent. of the total of 210,608,127 pounds of potatoes used in the United States in the manufacture of starch, and the manufacture of 20,514,277 pounds, or 82.5 per cent. of the total quantity of potato starch reported for the industry. Starch products are used for food, for laundering and sizing, for finishing calico, for thickening colors, and for many other purposes.

Investigations of Blast Furnace Operations*

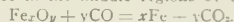
In a paper before the May 4, 1913, convention of the Verein Deutscher Eisenhüttenleute, Prof. W. Mathesius, of Berlin, developed a method of analyzing blast-furnace operations. The method was then applied to records obtained from twenty-eight different blast furnaces, and charts were drawn that show at a glance the interrelations of the most important factors in blast-furnace practice. Following the theoretical analyses, the results of experiments upon the relative reducibility of briquetted, sintered, and unroasted ores, were given.

In developing the method of analysis, it was assumed that all reactions between the charge and the gases except four could be discarded. These four reactions are as follows:

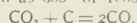
1—The oxidation of the carbon in the coke, in the hearth by the blast to carbon monoxide.



2—The reduction of iron oxides by the carbon monoxide which takes place in the middle regions of the furnace.



3—The reduction of carbon dioxide by the carbon in the coke. The former may come either from the charge or be generated by a reaction between carbon monoxide and ferric oxide. This reaction can take place only at temperatures considerably above the well-known Boudouard limit of stability: that is, at least as high as 900° or 1000° C. The equation is:



4—The catalytic dissociation of carbon monoxide which takes place in the uppermost regions of the furnace



This last reaction can only take place when the substances are considerably above the previously mentioned stability conditions, both as to quantity and temperature; that is, the temperature must be from 400° to 600° C. Above 600° C. the speed of the reaction becomes so slow that it has little practical bearing upon the blast-furnace operation.

Of the four reactions mentioned above, the first, second and fourth are useful, while the third is detrimental to the economic working of the furnace.

Reaction 4 is favorable to economical operation on account of the restoration of carbon that would otherwise be lost and works in direct opposition to the harmful reaction 3. In considering the performance of the furnace as a whole, reactions 3 and 4 can be assumed under favorable conditions to compensate each other and, therefore, in analyzing the performance of a blast furnace, reaction 4 can be left out of consideration entirely, while reaction 3 is only considered under conditions where it is not completely compensated.

It is possible for reaction 4 to become harmful under certain unfavorable conditions; for instance, if it takes place to an abnormal degree, the amount of carbon dust may become so great that it seriously increases the resistance of the charge to the passage of the gases.

Below we give the equations used in the mathematical analyses of data received from the various furnaces:

LIST OF SYMBOLS

$m = \frac{(CO_2)_g}{(CO)_g}$ = the weight ratio of the carbon dioxide to carbon monoxide in the waste gases.

$m' = \frac{(CO_2)_a}{(CO)_a}$ = the volume ratio of carbon dioxide to carbon monoxide in the waste gases.

Further, in kg. per kg. of pig iron,

C = the total carbon from the coke.

C_x = the carbon of the coke consumed by harmful reactions (the dissociation of CO_2 in the shaft and the reduction of Fe_2O_3 in the hearth which in this analysis are considered as identical because they produce the same effects upon the consumption of coke and the heat performance of the furnace).

C_{cz} = the carbon in the CO_2 of the ore and the flux

C_{ci} = the carbon taken up by the iron.

*Translated in abstract from Stahl und Eisen, Sept. 4 and 11, 1913

C_{mn} , Si, P = the carbon taken up by the reduction of MnO , SiO_2 , P_2O_5 in the hearth.

O_e = the oxygen of the oxide of iron in the ore.

O_l = the oxygen brought by the blast in the hearth.

O_b = oxygen from the charge contained in the waste gases.

$(CO_2)_g$ = the carbon dioxide in the waste gases.

$(CO_2)_{ez}$ = the carbon dioxide of the ores and the flux.

$(CO_2)_d$ = the carbon dioxide formed by reduction of the ore with CO, which is again dissociated by C.

$(CO_2)_y$ = the carbon dioxide formed by reduction of the ore with CO.

$(CO_2)_{ezu}$ = that portion of $(CO_2)_{ez}$ which passes up the shaft intact.

$(CO_2)_{ezx}$ = the $(CO_2)_{ez}$ which is decomposed in the shaft.

$(CO)_g$ = the carbon monoxide in the waste gas.

$(CO)_{fexos}$ = the carbon monoxide produced by the direct reduction of iron ore by C.

The general equations used in the analysis are given in full below:

Carbon Dioxide Equation.—The amount of CO_2 in the waste gases, not including that which comes from the ore mixture and is not reduced by the carbon from the coke in its passage upward, must equal the amount of CO_2 formed from CO by the reduction of ore, less that amount which is again dissociated by the carbon of the coke according to reaction 3.

$$(1) \quad (CO_2)_y - (CO_2)_{ezu} = (CO_2)_y - (CO_2)_d$$

The carbon dioxide of the ore mixture is split up as follows:

$$(2) \quad (CO_2)_{ez} = (CO_2)_{ezu} + (CO_2)_{ezx}$$

Carbon Equation.—The carbon from the coke, less that which has gone over to the iron as carbide and plus that which comes from the ore mixture, must be equal to the amount of carbon present in the carbon dioxide and the carbon monoxide of the waste gases.

$$(3) \quad C - C_{fe} + C_{ez} = 3/7 (CO)_g + 3/11 (CO_2)_g$$

Oxygen Equation.—The oxygen of the ore forms CO_2 with CO in the shaft of the furnace, or CO with C in the hearth.

$$(4) \quad O_e = 4/11 (CO_2)_y + 4/7 (CO)_{fexos}$$

The carbon which enters into the harmful reactions must equal the carbon which is gasified according to $(CO_2)_d + (CO_2)_{ezx}$, together with the carbon which reacts in the hearth forming $(CO)_{fexos}$.

$$(5) \quad C_x = 3/11 [(CO_2)_d + (CO_2)_{ezx}] + 3/7 (CO)_{fexos}$$

The expression $(CO_2)_{ezx}$ may be worked out from equations 1 and 2. Subtracting equation 1 from equation 2, we have by proper transposition:

$$(CO_2)_{ezx} = (CO_2)_{ez} - (CO_2)_g - (CO_2)_l + (CO_2)_y$$

This value when substituted in equation 5 gives:

$$(5^a) \quad C_x = \frac{3}{11} [(CO_2)_{ez} - (CO_2)_g - (CO_2)_l + (CO_2)_y] + \frac{3}{7} (CO)_{fexos}$$

For developing the values $(CO_2)_g$ and $(CO_2)_y$ the equation is written as follows:

$$(5^b) \quad C_x = \frac{3}{11} (CO_2)_{ez} - \frac{3}{11} (CO_2)_g + \frac{3}{11} (CO_2)_y - \frac{3}{7} (CO)_{fexos}$$

For obtaining $\frac{3}{11} (CO_2)_g$ we have from equation 3:

$$C - C_{fe} + C_{ez} = \frac{3}{7} (CO)_g + \frac{3}{11} (CO_2)_g$$

By substituting $(CO)_g = \frac{7}{11} (CO_2)_g$ we find:

$$C - C_{fe} + C_{ez} = \frac{3}{22 \cdot 11} (CO_2)_g (11 + 7m)$$

hence

$$(6) \quad \frac{3}{11} (CO_2)_g = 7m \left(\frac{C - C_{fe} + C_{ez}}{11 + 7m} \right)$$

$\frac{1}{11} (CO)_y$ is worked out from equation 4 thus:

$$O_e = \frac{4}{11} (CO_2)_y + \frac{4}{7} (CO)_{fexos}$$

hence

$$(7) \quad \frac{1}{11} (CO_2)_y = \frac{1}{4} O_e - \frac{1}{7} (CO)_{fexos}$$

Substituting equations 6 and 7 in equation 5^b, we have:

$$(5^c) \quad C_x = \frac{3}{11} (CO_2)_{ez} - \frac{7m}{11 + 7m} (C - C_{fe} + C_{ez}) + \frac{3}{4} O_e$$

According to the molecular ratio $CO_2 = \frac{11}{3} C$, hence

$$\frac{3}{11} (CO_2)_{ez} = C_{ez}$$

This, substituted in equation 5^c, gives:

$$(5^d) \quad C_x = \frac{3}{4} O_e - \frac{7m}{11 + 7m} (C - C_{fe}) + \frac{11}{11 + 7m} C_{ez}$$

It should be noted that the above equations are purely theoretical, and possess a perfectly general applicability, since no assumptions were made to fit any special local conditions.

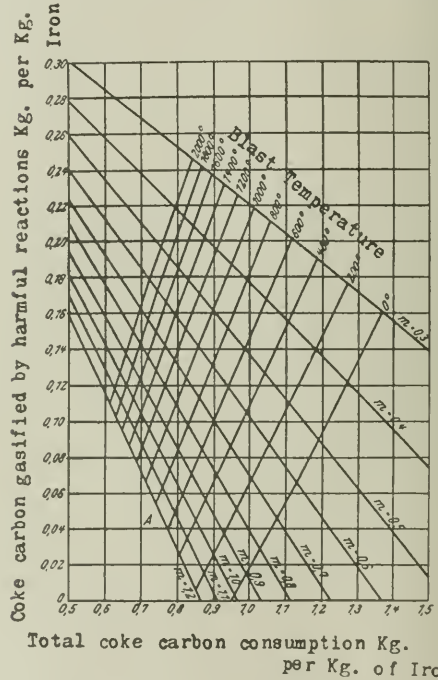


FIG. 1—DIAGRAM OF BLAST FURNACE OPERATIONS

If the CO_2 and CO are expressed in volumetric percentages instead of weight, the equation 5^d would be written

$$(5^e) \quad C_x = \frac{3}{4} O_e - \frac{1}{1 + 11m} [m' (C - C_{fe}) - C_{ez}]$$

The equation for C_x make it possible, wherever O_e , C_{fe} and C_{ez} are known, to construct a chart which gives a perfectly clear picture of the relations between C , m and C_x in the operation of any blast furnace. By assuming different values for m Fig. 1 has been thus constructed for operation with basic pig iron which has the following average composition:

C	3.70 %
Mn	1.734 %
P	1.713 %
Si	0.41 %
Fe	92.704 %
	100.00 %

Assuming a blast temperature of 600° C., and calculating the heat balance for the process, the total loss through cooling

water and radiation is found to amount to 24.48 per cent of the total heat supplied.

Now, if it is assumed that the loss remain practically constant for different points in the chart, the blast temperature may be inserted as unknown in the heat balance equation and determined for different values of m and C , the results being represented by lines drawn across the first ones.

The completed diagram or chart in Fig. 1 illustrates the intimate connection existing between coke consumption, blast temperature and the ratio of CO_2 to CO in the waste gases. Furthermore, it shows that with a constant blast temperature the changes in furnace operation must take place along that particular temperature line, thus determining the relations between the other variables and plainly indicates the great effect of the m or " CO_2 to CO ratio" upon the coke consumption, because the other variables remain fairly constant.

A cursory examination of this chart reveals the great importance of keeping continuous records of the m ratio in the operation of a blast furnace. The chart further shows that when working at the maximum m ratio (of CO_2 to CO) the lowest coke consumption is reached, and it may be interesting to trace the reason for this effect.

Naturally the amount of CO_2 in the waste gases is higher, the easier the reduction of the ore; that is, the greater the amount of reduction that takes place in the furnace by action of the gases, or conversely the ratio of CO_2 to CO is smaller, the greater the amount of ore which is directly reduced in the hearth by action of the carbon.

The value of the ratio of CO_2 to CO is also dependent in a very marked manner upon the regularity of working and uniformity of the furnace charge which for best conditions should offer a uniform resistance of the passage of the gases, thus making a uniform distribution and flow.

The point A in Fig. 1 indicates the conditions of practical operation in the particular case which was investigated. This furnace was charged with friable ores, and, therefore, had to be driven slowly in order to avoid trouble, which is indicated by a study of the diagram. On the one hand, the CO_2 to CO ratio is extraordinarily high (1.07), while, on the other hand, the carbon C available for compensation of the harmful reaction is relatively small. From these two facts it is evident why, with a yield of only 40 per cent, it is possible to get along with a coke carbon consumption of only 800 kg. per metric ton of iron. The carefully obtained heat balance sheet showed, as mentioned before, a loss of 24.48 per cent in cooling water and radiation.

Realizing that the cooling water and radiation losses for a given temperature vary only slightly for a given kind of iron, whether the operation is conducted fast or slow, it is evident that a furnace of given dimensions and operating conditions which produced double the amount of iron per ton of raw material will have half the cooling water and radiation losses per ton that the same furnace would ordinarily have. Therefore, it follows that blast furnaces should be operated at as high a speed as practicable.

Practical experience in different places, as well as these experiments, have shown that attempts to speed up the furnace operation always lead to the production of white cold pig, and this is true because of the small amount of C carbon available for counteracting the disturbances due to non-uniform settling of a friable charge and to the quantities of ore arriving un-reduced in the hearth of the furnace.

The operation of a furnace under these conditions is extremely difficult because the varying quantities of un-reduced ore which enter the hearth cause large variations in the hearth temperature.

In the case discussed above it appears difficult to determine the best conditions for operation because the furnace must be driven slowly on account of the friable nature of the charge, and this method of operation entails relatively large heat losses, a small coke consumption, and also small amounts of C carbon available for harmful reactions in the hearth.

The possibility of choosing more favorable conditions is

shown in Fig. 2, which is constructed under the assumption that a large portion, especially of the friable portion of the charge, is briquetted. The effect of this, as shown by experience, is to permit a considerable acceleration of the process without introducing irregularities into the working of the furnace and therefore the heat losses are reduced. In Fig. 2, at point B , it is assumed that they amount to 13 per cent. This sets free considerable heat which is available to counteract the harmful reactions. Due to this, the C carbon exceeds 0.14 per kg. of iron, which causes a change in the operating point of the diagram corresponding to a reduction of the CO_2 to CO ratio which goes down to 0.6. This reduction, however, makes it possible to use a hotter blast and thus to bring about a considerable saving of coke, for by raising the blast temperature to 800°C . the operating point may be transferred to C , as shown in Fig. 2.

This method of analysis also shows that increasing the speed

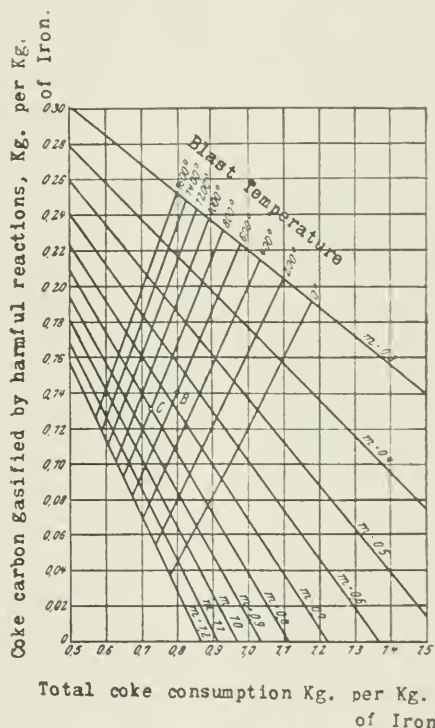


FIG. 2.—DIAGRAM OF BLAST FURNACE RELATIONS

of operation has the beneficial effect of considerably reducing the amount of blast required per ton of iron. In equation (3) the oxygen required for operation was given. If we substitute in this the values $(\text{CO})_0$ and $(\text{CO}_2)_0$, as given in equations (10) and (10), we obtain by simple transposition,

$$O_1 = \frac{4(11 + 14m)}{3(11 + 7h)} (C - C_{Fe} - C_2) - O_0$$

Substituting in this equation the value $m = 0.1$, we have the oxygen required as

$$O_1 = 1.41 (C - C_{Fe} - C_2) - O_0$$

If we substitute $m = 1$ the oxygen required is

$$O_1 = 1.85185 (C - C_{Fe} - C_2) - O_0$$

The ratio for the two cases works out

$$\frac{O_1}{O_2} = \frac{1.41}{1.851} = 0.702$$

In studying the literature of the subject it was found that

widely varying values were assigned to the losses of blast-furnace operation. Indeed, in some cases heat balances were given in which there was more heat on the outgoing side than on the incoming. Consequently, in order to obtain a reliable basis for the assumption for radiation and cooling water losses, it was necessary to calculate a very large number of heat balances for different blast-furnace operating conditions and for this purpose the author obtained suitable data from twenty-eight different furnaces and subjected them to calculations, the results of which are set forth in tabular form. The sum and substance of a careful analysis of these calculations shows that the economy of operation could be greatly increased if the ore could be introduced into the furnace in such a condition as to facilitate its rapid reduction; that is, in briquetted form.

On the other hand, it is very important to note that the total carbon consumption per ton of iron under high-speed operating conditions is not greater than in cases where the operating speed is less and the ratio of CO_2 to CO much more favorable. This is because in furnaces operating at higher speed there is a marked reduction in the radiation losses which offsets a higher coke consumption as compared with the slower furnaces.

In order to obtain a clear idea of the relation between the speed of operation of a furnace and the heat losses a diagram shown in Fig. 3 was developed. In this diagram it was assumed

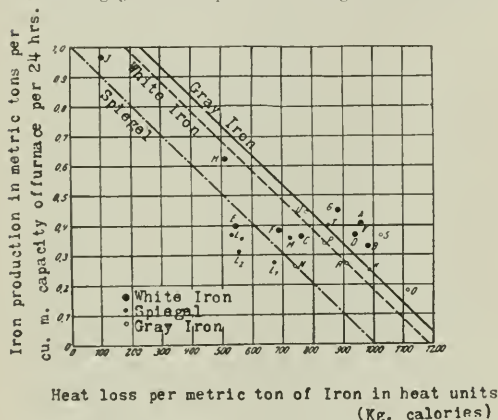


FIG. 3.—DIAGRAM SHOWING HEAT LOSS BY RADIATION

that the heat losses are inversely proportional to the speed of operation. The points of the diagram were taken from actual calculations. The diagram in Fig. 3 shows that the speeds used in current practice to-day correspond to a loss of approximately 1000 kg. calories per metric ton, and that by speeding up the operation they might be reduced to at least 50 per cent of their present value.

Unfortunately, at present there are only a few cases where briquets have been used for a sufficiently long time under proper conditions of operation to enable the actual saving in fuel to be determined. Consequently, there is not much data from actual practice available.

However, the author was able to present figures obtained in practice by Gustav Gröndal from his experience in Swedish iron works, where by the introduction of briquetted ore he was able in one instance to reduce the charcoal consumption from 1314 kg. per ton to 1000 kg. per ton by using a charge composed of 50 per cent briquets, and in this instance the iron content of the charge was increased from 56 or 57 per cent up to about 62 per cent.

In another case which Mr. Gröndal relates, the charge was made up entirely of briquets, and in this instance the charcoal consumption was only from 560 to 585 kg. per ton of iron.

Such favorable results from the use of briquets in the operation of blast furnaces can only be realized by careful attention to all the conditions that are involved, and operating the fur-

nace in a uniform manner and in such a way that by far the greater portion of the charge is reduced to metal by the gases.

Exhaustive experiments bearing upon the reduction of iron oxides by gases have been conducted at the Technische Hochschule in Charlottenburg, and the detailed results of these investigations will form the subject of a special thesis. However, the nature of the results is plainly shown in the accompanying diagrams.

In these experiments the briquets were tested in quantities of a few hundred grams, each broken into pieces of about nut size. These were heated to different temperatures in an electric furnace and subjected to the action of a stream of illuminating gas for several hours. Under this treatment, depending upon the nature of the briquet, varying amounts of the ore were reduced to metal or ferrous oxide.

The results are shown in Figs. 4 to 18, from which it is seen that briquetting makes it possible to transform practically all of the iron oxides, contained in the ore, into metal. On the other hand, while sintered material must be subjected to the action of the gases for a much longer period, in order to reduce the ferric oxide to metallic iron, the ferrous oxide cannot be reduced by the gases. In some cases the ferric oxide in the sintered ore was only partially reduced to metal, while the rest was transformed to ferrous oxide.

It is noteworthy that the briquetted material gave more favorable results than the lump ore, the cause of which probably lies in the physical structure of the briquet, which, being more porous, is more susceptible to the action of the gases. Figs. 8 and 18 demonstrate this very plainly. Fig. 8 represents a piece of solid magnetite, while in Fig. 18 the same ore was pulverized and briquetted by the Scoria process.

These results show the great importance of using briquetted material in blast furnace operations. By substituting briquets for friable ores, and ores which are difficult to reduce, such as magnetite, it is possible to operate a given furnace at a considerable higher speed, thus increasing its daily capacity and decreasing the percentage losses in cooling water and radiations, the result of which is a considerable saving in coke.

* * *

In discussing Professor Mathesius's paper, Professor Simmersbach, of Breslau, made the following remarks:

The results of Professor Mathesius's investigations show that sintered ores are not well adapted to indirect reduction. I fully agree with his conclusions, yet it is nothing new. About ten years ago I brought out these same facts in connection with studies made by me upon sintered iron ore containing manganese. It was shown that with the same manganese content in the sintered ore charge there was less manganese reduced and entering the iron and that for a certain percentage of manganese in the iron additional manganese had to be added to the burden, which naturally caused an increase in coke consumption.

The speaker cannot agree with Professor Mathesius in his assumption that reactions (3) and (4) may be considered as practically offsetting one another. Theoretically this might be permissible, but practically considered, both of these reactions must be taken into account. According to the first reaction, CO_2 is taken from the carbon of the coke, while in the second one carbon is returned; however, not to the coke, but to the ore. The effect of this absorption is to cause the ore to swell, become more porous and thus increase its reductibility by the gases, and this is most important in indirect reduction. According to tests made by Laudig at Buffalo, the carbon absorption by the different ores is as follows:

Mesabi ores		21.61 per cent of the ore weight
Soft hematite ores		13.82 per cent of the ore weight
Hard hematite ores		7.52 per cent of the ore weight
Hard crystalline hematite ores		3.68 per cent of the ore weight
Iron slags		0.38 per cent of the ore weight
Magnetites		0.00 per cent of the ore weight

In absorbing this amount of carbon the Mesabi ores swelled to double their normal size and became very much more porous. I have here a piece of magnetite taken out of the hearth

of a blast furnace, which 2 mm. below its surface is still in its original unaltered condition. This incomplete reduction is due to the density of the magnetite.

In the mathematical analysis of blast furnace operations as to the deposition of carbon, it must be borne in mind that the burning of coke according to reaction (4) produces 50 per cent more heat than direct burning; this is demonstrated in the following calculations based on furnace gas containing 1 CO₂ to 2.25 CO.

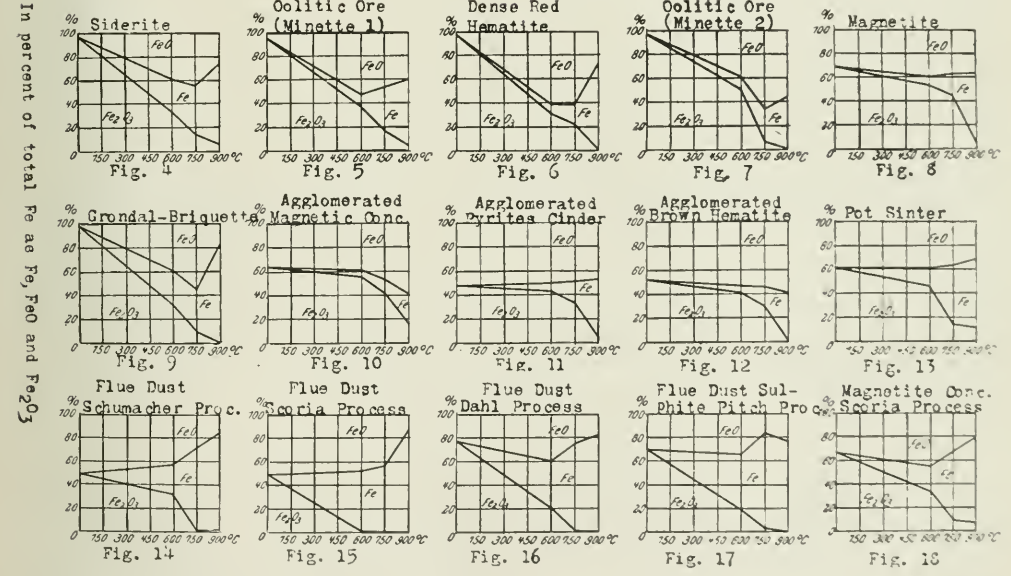
(a) Direct Burning. (C + O = CO and CO + O = CO ₂).	
1C develops by burning to CO ₂	8080 heat units
2.25C develops by burning to CO	5557.5 heat units
3.25C develops by burning.....	13637.5 heat units
1C develops by burning.....	4196.5 heat units
(b) According to Reaction (4).	
(C + O) = CO and 2CO = CO ₂ + C).	
2C develops by burning to 2 CO	4940 heat units
2CO = CO ₂ + C	
1CO develops by burning to CO ₂	5610 heat units
1CO absorbs by the reduction to C...	2470 heat units
	—
	3140 heat units
1C gives according to direct burning.	4196 heat units
2C develops	12276 heat units
1C develops	6138 heat units

Pb	1.21
Cu	0.51
Al ₂ O ₃	0.84
CaO	0.91
MgO	0.94
Sulphur as sulphide	2.27
P	0.114

The balance mostly oxygen.

In the briquetting of flue dust a number of important factors are to be considered, namely, the variable chemical composition, the quantities of raw, sintered and melted particles composing the dust and also the coke fines therein contained. The age of the dust and the place from which it was obtained is also important, as fresh dust always contains hydraulic binders. If left for any length of time in the open or in flues distant from the furnace, subjected to the influence of moisture or carbon dioxide, the binding constituents will become hydrates and carbonates. Dr. Schumacher first discovered the practical significance of the last named phenomenon, and thereby developed his briquetting process.

When briquets are tried out on the furnace, it is customary to replace a certain amount of ore by as large a quantity of briquets as possible. If there is an improvement in the operation of the furnace, the briquets are credited with the same. If, on the contrary, the operation of the furnace is impaired,



FIGS. 4 TO 18.—REDUCTION OF ORES SINTERED AND BRIQUETTED MATERIAL. FIGS. 4 TO 8.—ORES. FIGS. 9 TO 13.—SINTERED ORES. FIGS. 14 TO 18.—BRIQUETTED ORES AND FLUE DUST

Because of the lack of time, I refrain from further criticism as to the author's method of calculation, since his conclusions have for years been found correct in actual practice.

Referring to the results of experiments as plotted in Figs. 4 to 18, it should be noted that the various processes cannot be compared with each other unless the same raw material is used in all cases. Considering particularly a flue dust briquet and its reducibility the following analysis was obtained from a partly reduced Schumacher briquet containing 25 per cent metallic iron:

Total carbon	7.40
of which coke, graphite.....	4.88
SiO ₂	0.55
Fe	61.00
Mn	0.40
Zn	6.70

it is a question whether this is directly chargeable to briquets, or only indirectly. When using briquets on a furnace the blast is of primary importance, and the pressure, the amount and their temperature must carefully be considered. Although sintered material is more difficult to reduce than briquets, and therefore less advantageous, yet there may be unusual conditions under which sintered ores, with correspondingly high temperatures and certain ore mixtures, will give better results than briquetted material.

In the author's experiments better results were obtained from briquets than from lump ores. A reason for this, besides the difference in density, is due to the coke amounting to from 10 to 25 per cent, and which is distributed evenly in the briquets. It is my opinion that the coke in the briquet should be figured as coke in the calculated charge, as to the coke consumption.

Professor Osann made the following remarks in the discussion:

The author's temperatures used in test are too low to be of greatest practical value, as they do not indicate what takes place in the hottest part of a blast furnace.

Professor Osann takes issue with the author on the point that the most easily reducible ore mixtures are always the most desirable ones. Practice does not, in all cases, bear this out. Ores that are difficult to reduce are often added to the charge in order to improve the performance of a furnace by counteracting certain difficulties that occur when easily reducible ore mixtures are used.

It is well known that ores that are difficult to reduce are very desirable when first-class iron for casting is wanted. The very easily reducible Lake Superior ores are reduced in American furnaces with no less an amount of fuel than are the ore mixtures of the lower Rhine, which are rich in magnetite. The reason of this is that easily reducible ores pass so quickly through the furnace that the saving in fuel is offset.

The blast furnaces in America use at least as much fuel as the furnaces in Germany under similar conditions, and probably the American furnacemen would be well pleased if their ores did not reduce so easily.

It is also well known that in Siegerland and in Steiermark red spar is not used alone but about one-third of raw spar is added. By this means the disturbances caused by the easily reducible ore are practically eliminated.

The cause of these phenomena we do not exactly know at the present time. We do know, however, that in blast furnace operation all the different factors work together and that no one of them can be considered by itself.

As far as deciding which briquetting process should be used, no other way is open except experimentation, and these experiments must be continued over considerable length of time. Professor Osann advises that, without any prejudice, experiment should be conducted with both sintering and briquetting processes and decision should be made entirely in conformity with results obtained.

Some processes should be dropped out at once as being of too little advantage or of too great a cost.

Sinter is often compared with slag because of its resistance to reduction; however, this is not entirely permissible, because slag consists principally of silicates while sintered material is produced by gangue going into solution in iron oxides without the necessary melting.

When flue dust briquets, made by the use of a binder, sink into the furnace, they soon behave in some similar way to sintered material.

Professor Osann recommends a program for exhaustive experiments to prove the value of briquetting and then closes by criticizing the use of temperatures below 900° C. in the tests and the assumption of a constant heat loss in the analysis. He hopes, at some future time, to be able to discuss in detail the effect of speed of operation upon heat losses.

Condenser Tube Corrosion

Past and Future Work of the Corrosion Committee of the British Institute of Metals

By Sir Gerard A. Muntz, Bart., and Professor H. C. H. Carpenter, M.A., Ph.D.

The (British) Institute of Metals Corrosion Committee, which consists of the following members: Sir Gerard A. Muntz, Bart., chairman; Professor H. C. H. Carpenter, M.A., Ph.D., Hon. Secretary; Dr. G. D. Bengough, M.A., Hon. Investigator; Mr. L. Archbutt, Engineer Rear Admiral G. G. Goodwin, R.N.; Professor A. K. Huntington, Assoc. R. S. M.; Mr. J. T. Milton; Mr. A. Philip, B.Sc., Assoc. R.S.M.; Sir W. E. Smith, C.B.; Mr. L. Sumner, M.Sc., and Professor T. Turner, M.Sc., has now been at work for some three and one-half years.

Under its guidance two reports have been published by Dr.

G. D. Bengough, who has carried out his investigations in the Metallurgical Department of the University of Liverpool.

The first report, issued in January, 1911, and published in Vol. V of the *Journal of the Institute of Metals*, was intended to give a general review of the knowledge as to the corrosion of non-ferrous metals both in its practical and scientific aspects, and is the most comprehensive survey of the subject that has yet been published. On the basis of this report the committee decided to initiate an experimental investigation of the cause or causes of the corrosion of condenser tubes in marine engines.

An appeal for funds was issued and the response was sufficient to enable a two years' research to be undertaken, which formed the subject matter of the Second Report to the Committee presented at the Ghent meeting of the Institute of Metals held at the end of August in the present year.

CORROSION RESISTING TUBES

The experimental work has been carried out both in a special condenser and by means of laboratory experiments. The condenser plant was erected with the object of reproducing as accurately as possible the conditions to which condenser tubes are submitted in actual practice, and the laboratory experiments were designed to interpret the observations made with the larger apparatus, a procedure which has been signally successful in its results. In both cases it was found possible to observe and study "dezincification" which is the usual cause of the failure of tubes in practice. Tubes of four different compositions have been examined under strictly comparable conditions. The ordinary 70:30 tube of commerce has been shown to be far surpassed in resistance to dezincification by tubes containing 70 per cent copper, 28 per cent zinc, and 2 per cent lead, and by those of "Admiralty" composition, i.e., containing 70 per cent copper, 29 per cent zinc and 1 per cent tin.

Muntz metal, 61 per cent copper, 39 per cent zinc, has been shown to be unsuitable for use as condenser tube material.

GREAT IMPORTANCE OF TEMPERATURE

The influence of a variety of "deposits" and "particles" which have hitherto been supposed to cause localized dezincification has been shown to be of only secondary importance, and dezincification has been found to start and proceed rapidly in their absence. *In all cases the rapidity of the failure of the tubes was found to depend principally upon the temperature to which they were subjected.* If corrosion is to be prevented, or at any rate minimized as much as possible, *the temperature in the condenser must be kept low.*

Preliminary experiments have been made on the mode of action of electrochemical protection. This has been found to be intimately connected with the deposition of a continuous layer of calcium carbonate upon the inner surface of the tubes.

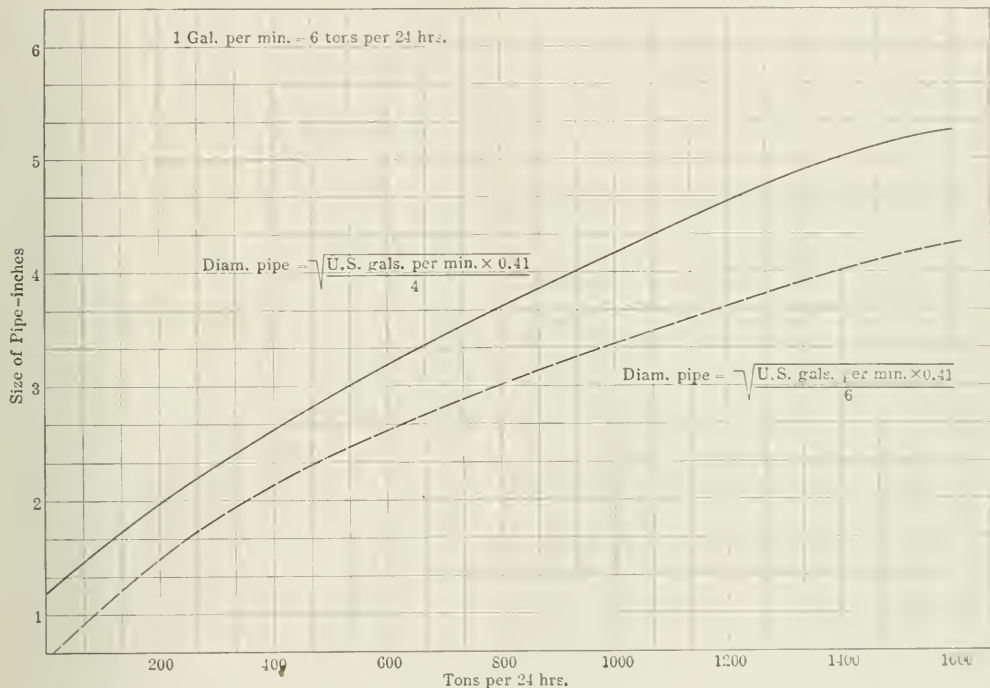
SCHEME OF NEW YORK

The committee have met recently and having obtained sufficient funds to continue their investigations for another two years they have planned a further scheme of work which is now well in hand. In deciding upon the lines of future investigation they have been careful to take into consideration a number of valuable suggestions and criticisms which have been made on the Second Report both in the discussion at Ghent and in the comments of the technical press. The scheme includes a complete time-temperature survey of the conditions under which dezincification can take place and a more detailed study of the temperatures existing in their own and other condensers. Special attention will also be directed to the conditions under which electrochemical protection can be effectively maintained, and in this connection a comparative study of all the principal methods of holding tubes in tube-plates will be carried out. A systematic series of experiments will also be initiated with a set of tubes of a copper-aluminum alloy from which specially interesting results are anticipated.

The foregoing scheme by no means exhausts the lines of investigation which in the opinion of the committee might profitably be undertaken, and if further financial support is forthcoming they will gladly extend their researches in other equally vital directions.

Sizes of Pipe for Cyanide Mills and Industrial Plants

The accompanying curves will be found useful in determining the size of pipes to be used for water and solution in cyanide mills, or in industrial plants where it is necessary to convey definite volumes of liquid through pipes from one point of a process to another. The present chart was drawn by Mr. Justin H. Haynes, and used by him in the design and construction of the Junta mill, near Telluride, Col. The solid line refers to a velocity of 4 ft. per second, and should be used whenever possible. The dotted line is based on a velocity of 6 ft. per second and is not normally used.



SIZES OF PIPES FOR CYANIDE MILLS AND INDUSTRIAL PLANTS

An acid-proof lining for tanks can be prepared from the following formula: 10 per cent litharge, 20 per cent short-fibre asbestos and 70 per cent sand, mixed to a plaster with a 40° Be. solution of silicate of soda. The plaster can be applied as thick as desired. An application of acid after drying will "set" the cement and render it acid-proof.

Refractories in Montana.—Among the raw materials used by the Anaconda Copper Mining Co. for making refractory brick are (1) a plastic clay from Arrington; (2) a flint clay from Lost creek; (3) silica, and (4) "Dillon" rock. The silica (3) used for silica brick contains about 97 to 98 per cent SiO_2 ; lime is used as a binder. The "Dillon" rock runs 99 per cent SiO_2 , and is used for sanding molds; it is a fine, dense, white sandstone, not fit for making silica brick. Analyses of the two clays are given below.

	Plastic	Flint
SiO_2	55.7%	76.9%
FeO	1.1	0.0
MgO	0.5	0.0
Al_2O_3	30.8	17.3
CaO	0.2	0.0
Alkali	2.1	—
H_2O , etc.	10.2	—

Ozone

By A. Vosmaer, Ph.D.

In ordinary oxygen two atoms are combined to form a molecule, but it takes three atoms to form a molecule of ozone. Different names have been given to this particular gas. Some people have called it activated oxygen, others have given it the name of condensed oxygen, some have spoken of electrified oxygen and the name polymerized oxygen has also been suggested.

None of these suits the purpose in a strict way. It is better to distinguish sharply between substances of different properties. In fact, Ostwald who defines a body as a state of things

differing from its surroundings, considers allotropic modifications as different substances and this seems very sensible. It is really too bad to have different sets of properties for one and the same substance in different modifications, such as red and yellow phosphorus, or diamond, graphite and carbon, or sulfur in all its varieties. Is not it much more plausible to consider them as really different substances?

The fact that chemically they closely resemble each other does not entitle us to consider them as one and the same, it is not only the chemical properties which determine a substance, and even in that respect we are not always so very sure about it.

The names "activated" and "condensed" oxygen are wrong altogether. Ozone is more active than oxygen because one, or all three of its atoms, are holding each other more less firmly than the two in oxygen. The well-known oxygen in *oxygene nascenti* is none the less as active as ozone. One should not lose sight of the fact that if we speak of oxygen we have in mind the gas in its molecular, (two-atomic) state and not in its atomic state. Atoms are altogether different in their behavior from molecules, so we must guard ourselves against mixing them up. This consideration also condemns the word "condensed," because there has been no condensation of O_2 but of O .

The prefixes "electrified" and "polymerized" do not please me

either, the first because it is too vague, the second for the same reason as above, that ozone is no polymerization of oxygen, is not (O_3)¹ but is O_3 .

What other name shall we give it then? Well, if we call it ozone there is no need for another name. Its composition has been fixed to be O_3 its molecular weight being 48.¹

A great difficulty for philosophical consideration is that we know all substances by their molecular state while in chemistry we deal with them in their atomic state. This cannot be altered for the present. It has to be endured as long as it cannot be cured.

MANUFACTURE OF OZONE

Ozone is difficult to make. It is an endothermic compound, which means that it absorbs heat when forming. How much has not been settled yet. The amount seems to be in the neighborhood of 30,000 gram calories per gram molecule. But unfortunately the temperature of formation and that of decomposition lie very close together, so that, although heat is wanted, it should be kept away as well. In other words as soon as the ozone is formed it should be taken out of the heat zone.

Practically this means that ozonators work best at the lowest possible temperature. There is always quite enough heat in the discharge itself for the formation of the ozone and all that is formed must be quickly drawn away, out of the influence of the discharge which also has the tendency to desozone.

These circumstances present interesting cases of equilibrium reactions. When working with air it is hardly possible to get a higher concentration than about 30 grams per cubic meter (containing about 260 grams of oxygen) and this figure does not run up so very much higher when using pure oxygen instead of air.²

Ozone may be made chemically, mechanically, and electrically and in different ways, but most of them have not the slightest importance from a commercial viewpoint.

By way of example we call attention to the chemical method of making oxygen by means of sulfuric acid and permanganate of potassium; oxygen thus made contains some ozone. By way of curiosity we mention the fact that violent mechanical disturbance of the air may cause some very little ozone to be formed. It has been reported that when grinding wheels were being tested for bursting speed and when they were flying to pieces the odor of ozone was noticed.

Among the electrical methods there are two; one in the foreground, one in the background. The latter one is the electrolytic when electrolyzing strongly acidulated water, or rather diluted sulfuric acid (sp. gr. 1.1), the oxygen liberated is strongly ozonized; up to 17% (according to others up to 25%) of the oxygen may be ozone, the process being carried out at low temperature and with a high current density. If we take something like 70 or 80 grams of oxygen per kw-hour, as a practical figure, the yield in ozone comes up to 20 grams under the very best conditions. The yield is low but not so low as to cut out competition, because the other products, oxygen and hydrogen, may have their share in the business and may considerably reduce the cost.

For the present moment, however, the other method, that of electric discharges through air or oxygen is ahead of all other methods on account of the yield being several times that of other methods. Commercial ozonators yield up to 70 or more grams of ozone per kw-hour.

The subject of ozonators is one of importance but is too large to be treated in a few words.

For our purpose it will be best to consider only the very best make. Other systems may have scientific interest, but have no chance of permanent application.

None of the systems in which no solid dielectric is used, has been able to make its way while the system based upon the use of a solid dielectric has succeeded in holding its original position.

The principle underlying is very simple. Any two conductors with sufficiently high potential difference (say above 6,000 volts) and separated by some solid dielectric will discharge the electricity in the form of the so-called dark, silent or brush discharge. This is the kind that possesses the remarkable property of converting oxygen into ozone. Although this principle is simple enough in itself, the details of the method by which it is carried out into actual being, are very important. Numberless patents have been taken out for different arrangements, but, most of them are stone dead and only few are of value.

Those patents are not basic. They all refer to some arrangement, form of electrodes, shape of dielectric and so on. We shall not discuss them here because this article is intended to show the importance of ozone rather than that of some particular make of ozonator.

As said any arrangement of electrodes separated by a dielectric will do. Not all will do equally well. Some apparatus are excellent in the laboratory, others are good in technical work. Some people want ozone of very high concentration, others again want it at least cost. In some cases reliability is of first importance, for instance, when using ozone for the sterilization of drinking water. In other cases high output per kw-hour is the principal requirement. As we intend to devote a future article to ozonators, we shall not enter into details now.

If we compare the theoretical equivalent of ozone in kilowatt-hours with the actual yield, a very great discrepancy is found. We ought to obtain over 1000 grams and at the most we get 100 grams per kw-hour, that is 10 per cent at the very best, and yet there is little hope for further improvement unless some genius strikes a happy thought. For the present moment one has to be content with such low figures as 10 per cent and in most cases not more than 5 per cent efficiency.

PROPERTIES OF OZONE

The pre-eminent property of ozone is its extraordinary oxidizing power, which from a chemical standpoint, can be best judged from the fact that it oxidizes silver.

Popular pamphlets enumerate a great many substances that will be oxidized by ozone, including such substances as ferro compounds, mercury compounds, etc. Now this is not worth mentioning, since if we once know the tendency of ozone to oxidize, it goes without saying that certain reactions will occur. We need not give such a list which anyway could be enlarged to any length.

What is particularly interesting about this property of ozone is that when it has fulfilled its oxidizing function there remains nothing but oxygen, which, being a gas, can easily be got rid of if so desired. This point should be borne in mind when we have to carry out some oxidizing process because in many cases we want to oxidize but do not want any other reactions such as are likely to occur when we use one of the more common oxidizers.

Peroxide of hydrogen has a similar advantage in so far as what is left in its case is water. It is, however, not so easy to get it in a pure state. It does not keep very long unless in acid solution, but it answers the purpose if pure.

We need not insist upon the fact that very few chemical reactions take place when we just drive oxygen gas into some solution. With ozone this is quite different. Here we have oxygen *in statu nascendi* exempt from other chemicals. It will, therefore, prove a very valuable agent in many chemical reactions.

It is disputable whether or not its remarkable bactericidal power is the same as the foregoing, or if it is another property.

If we stop to realize what it means to sterilize water containing thousands and thousands of bacteria per cubic centimeter by using a quantity of one in a million (that is one gram of ozone to sterilize 1,000,000 grams of water or 1 cubic meter or about 260 gallons), we feel inclined to attribute this abnormal power to some definite and specific action on bacteria. As matters are presently I would not dare, however, to claim such a specific property for ozone. It may be just nothing but ordinary oxidation.

¹ Leaving out of consideration, for the present moment the facts that partly it may be of more complicated nature, i. e., partly O_3 , partly O_4 .

² Professor Warburg obtained values of over 200 but on very small laboratory scale.

Ozone is said to cause phosphorescence in water. It may be so, I have never seen it. But I happened to watch another very curious phenomenon which went on for a whole day. But afterwards I have never been able to get it again. This unfortunate circumstance has prevented an explanation or even a suggestion.

On one of the hundreds of days that I have been working with ozone and water, once it happened that the sterilizing apparatus, which was made of glass so as to permit watching the operation, turned a violet color.

The sterilizing apparatus was a glass cylinder, 1 ft. in diameter, and 30 ft. high (built up in sections). During all the particular day the regular appearance of that column was changed to a violet color, about the shade of a faint amethyst or the reflection of arc light on a wet street. Probably the phenomenon has no bearing upon ozone but is merely a physical effect due to an intimate and very finely divided mixture of gas and water. The same color may sometimes be observed at the edge of the water line when going at a good speed in a calm sea.

That much for the chemical properties of ozone. The physical properties are of importance when it comes to the making of ozone. Ozone is a colorless gas with a sharp penetrating odor which has often been compared to that of chlorine or phosphorus. However, chlorine smells absolutely different from ozone, and comparing it with phosphorus means comparing it with itself, since the peculiar smell of phosphorus is probably due to slow combustion which is said to generate ozone. Here again I would rather say that ozone smells like ozone.

The gas is very unstable. It takes very much trouble to have that third atom join the two others and it is a very loose connection. It decomposes slowly at ordinary temperatures, but rapidly at high temperatures. Above 270 deg. C. it cannot exist under ordinary circumstances.

At a pressure of 125 atmospheres and a temperature of minus 103 deg. C. ozone becomes liquefied, it is then a dark blue mobile liquid of notable magnetic susceptibility.

Ozone is an endothermic compound, that is it requires heat for its formation, which, however, does not mean at all that the hotter an ozonator the better. On the contrary, the cooler the better.

The property of being very unstable prevents us from being able to store ozone. We hardly can expect even to be able to sell ozone as a liquid in steel bottles like carbonic acid or so. But there is no need for this, since the market offers many kinds and sizes of ozonators and with them any amount and any concentration can be obtained that may be wanted in the industries.

USES OF OZONE IN DAILY LIFE

The utility of ozone culminates in the purification and sterilization of drinking water. This is a subject of such importance that it is rather difficult to do justice to it while confining the discussion to a few lines, as is necessary in this short article.

It can be done when we are allowed to leave out the proofs of what we are saying. Be it sufficient to say that all statements are based on personal experience, except the experiments with pathogenic bacteria, which have been carried out by the German Imperial Board of Health.

To be very brief, we may just summarize what is known by saying: Ozone when of proper concentration and when judiciously applied will kill any amount of bacteria in water. We tested up to as many as 600,000 per cubic centimeter. Ozone acts selectively in so far as the pathogenic, which are the least resistant, are killed first. The German Board of Health tried mixtures up to 30,000 cholera and as many typhus bacteria, added to ordinary river water and none of those were left after treatment.

Some people do not realize the size and nature of bacteria and feel uneasy about their carcasses, but a little figuring will reassure them.

Suppose some water to contain 100,000 bacteria per cubic

centimeter, which is an abnormally high figure. There are 375,000,000 in a gallon, their size is about .00004 in their cubic size, 0.00000000000004 cubic inch, so that the tremendous number, 375,000,000 bacteria per gallon occupy only 0.00000004 cubic inch for the whole lot. Now consider a bacterium that consists of about 90 per cent of water, and 10 per cent solid matter. A solid may contain something like 0.1 per cent of mineral matter, mineral matter than said 0.000024 cubic inch of bacteria yield 0.0000000024 cubic inch of "ashes." Thus it is evident that there is so little matter in bacteria that it matters little whether or not we take it into consideration. Moreover, one has to remember that water usually contains some mineral matter so that after all the last figure should have been expressed in percentage increase and that shows the absurdity of thinking of remnants or cadavers of bacteria, which moreover are not an animal but of vegetable substance.

There is overwhelming evidence as to the absolute bactericidal power of ozone in water, but one *conditio sine qua non* has to be considered, viz., that of competent planning and expert management. Both conditions have very often been overlooked and failure could not but result.

Besides its bactericidal power, ozone has also that it takes away all discoloration, even that caused by organic ferric compounds (the precipitated iron has to be removed afterward by suitable means). It also takes out any disagreeable smell and taste, giving a taste like fresh spring water, which is probably due to the excess of oxygen.

Everybody who really has had thorough experience with the ozonization of water will have to acknowledge the absolute superiority of ozone as a purifier to any other method. Ozone is the ideal. Nothing more is left to be desired, not even the cost. We may mention here that as 1 gram of ozone can treat 1 cubic meter, or say 250 gallons of water of medium quality, and as one gram ozone of sufficient concentration per cubic meter of air to do the work can be produced by 20 watt-hours, there are required 80 kw for 1,000,000 gallons of water per hour. The cost of this amount of energy *delivered 24 hours a day* is a figure which varies from place to place, but is by no means prohibitive, as is well known to readers of this journal.

We wish particularly to draw attention to this extremely low figure of cost because many people repeat the verdict "too expensive" without even knowing the cost and certainly without realizing the advantages.

Another reason why we mention the price is that, after all, ozone is expensive in itself. It is because so little of it is needed for *this* purpose that the cost comes so low. For many other chemical processes it is out of the question because of its price. Ozone should never be used where other means are just as good. Its use should be restricted to those cases where other means fail to do the work or fail to do it as satisfactorily.

The next best thing after pure drinking water is pure air and this is another achievement of ozone. While it does not sterilize it when dry, yet it takes away nasty odors, like that of tobacco smoke and partly masks, partly oxidizes products of respiration, perspiration, and other volatile matter. Ozone treated air cannot compete with real pure air from the mountains or at sea, but it invigorates when being breathed in adequate dose, by all means not too strong. A proportion of about one in a million is quite enough for ordinary purposes.

It does make some difference to the men in an over-crowded office whether or not the air breathed by every one of them several times over is ozonized or not. Anyone who actually has experienced the great difference in sensation will be able to appreciate fully the beneficial effect of ozone. It has been a mistake, however, to attribute to ozone what it really does not do, viz., sterilize the air.

The beneficial effect is unquestionable. Moreover, the experiment is very easy to try. There are many air ionizers on the market. The energy which these small apparatus require is hardly worth speaking of, as it is of the order of magnitude of that of an incandescent lamp.

USES OF OZONE IN THERAPEUTICS

We must be very short on this subject however important it

may be, as there is very little reliable information to be had on this matter.

There is some evidence that ozone is very beneficial to phthisis patients when in the first stadium. It should certainly be not stronger than in the proportion of one in a million or one in ten million parts of air, when intended for continuous inhalation. This point seems of enough importance to be taken up seriously by some official medical society.

When we consider obesity as the result of inadequate power of oxidation, the body building up no further than to fat instead of going to a higher step of oxidation, for instance, fibre, tissue, flesh, and so on, ozone might be of help.

The therapy of "eating" oxygen in the form of peroxide of barium and other peroxides seems very poor indeed; it is very probable that through inhaling ozone, of course, in very weak concentration, additional oxygen may be brought into the system.

It is dangerous for a layman to discuss matters of a medical nature. It will be better to make one statement which is easily controllable and this is that the inhalation of ozone causes a rise in the percentage of oxyhemoglobin when this happens to be below the normal, and this rise accounts for the supposed beneficial effects.

Cases of chlorose, and all that is connected with the blood in a direct way, may be treated successfully with ozone. But my own personal experience does not go further than the increase mentioned in the percentage of oxyhemoglobin, which suggests or even indicates possible favorable results from proper ozone treatment.

INDUSTRIAL USES OF OZONE

Let us bear in mind what we said about the price of ozone that there never is a chance for ozone where cheapness is the very first question to be considered. But cheapness is not always the main factor. Very often quality comes in as a more important item.

As a bleaching agent ozone has to compete with chlorine. For bleaching paper pulp, however, there are other factors to be considered, like the size of the fibre, which is much better if ozone is used. In fact, when comparing ozone-bleached pulp with chlorine-bleached pulp the difference is striking, the individual fibre being about three or four times as large in the former case. This is worth while considering, for the longer the fibre the more "stuff" can be added to make paper out of it.

The bleaching of wax has been carried out with fair results, but not so striking, as to think much of it. The bleaching of sugar molasses succeeds easily, but at the expense of power of crystallization. This fact does not surprise us, when thinking of the hyper-sensitiveness of sugar to turn to the non-crystallizing kind.

The treatment of milk is a great problem of excessive importance, but the trial of ozone for this purpose leads only to disappointment, since the very tender organic compounds contained in milk, are destroyed long before any of the bacteria are destroyed and this is not surprising.

It may be mentioned here that it was the desire to sterilize milk which in Holland led to the pioneer work of Tindal (Schneller patents). When he saw that ozone would do nothing but harm to milk Tindal was the first to recognize the possibility that ozone might be beneficial to water. I dare say that to Tindal is due the present status of water sterilization by ozone. All that has been done—and that is a great deal—has been the result of his showing the way and full credit should be given to him although usually it is not customary to acknowledge the merit of those who did not succeed themselves but gave the impetus to the success of others. Tindal died some years ago.

The treatment of flax can be carried out with great success with ozone. The process known as rotting consists of having the glue of the stalk destroyed through the development of certain bacteria. This result is desired because when combing the raw flax, particles of glue prevent it from being done so effectively or may cause loss of material.

Ozone-treated flax does not contain any glue or other sticky

matter and thus said treatment improves the product to a great extent, and more especially so the yield of fibre.

It was not until very much later that I found the reason of this favorable result.

I found it by chance when trying the effect of ozone on glue, the purpose being a bleaching of it.

It turned out that ozone bleaches glue effectually but, after having been thus treated, glue did not possess any more of sticking property. For gluing purposes it is no longer suitable, but the same phenomenon explains the beneficial effect of ozone in treating flax.

Another vast field of possibilities is found in the industry of oils and fats. When making varnish out of linseed oil, discoloration is hardly to be avoided on account of the higher temperature used. If, however, the oxidation is performed by ozone it can be done entirely at ordinary temperature. The result is a perfectly transparent varnish of high quality—and of higher price, as may be added.

For the manufacture of surrogates for butter use is being made of cottonseed oil. This oil, however, possesses a rather strong and nasty taste; to take that taste away has been tried and it can be done.

Another problem has been the bleaching of santal oil, a very expensive oil, the bleaching of which is done by aid of sunlight or peroxide of hydrogen.

I have tried ozone for this oil and obtained a very good result.

However, these questions are rather more commercial than technical, for the problem turns out to be this: Does it pay to spend extra money on the purification so as to get a higher price for the product? Such questions cannot be treated in a short condensed summary like this.

There are numberless other possibilities for the use of ozone. Very many have been suggested and perhaps some have been tried. But as I did not want to make any statement in this article unless backed by my own personal experience I cannot give information on those other possibilities, not knowing them from actual tests.

Now that the time has arrived that chemical laboratories can have ozonizers of convenient size, and in case of successful results, can have large ozonizers of any size, it is to be hoped that earnest investigators will set to work in their particular lines and will contribute to the development of the ozone industry.

No antimony was produced in the United States last year from domestic ore, but a considerable quantity was saved in the form of antimonial lead, which is obtained at the smelters of precious metals in the course of their operations, and large quantities of antimonial alloys are recovered from secondary sources, such as scrap bearing and type metals, solder and antimonial lead drosses. From antimonial lead of both domestic and foreign origin, but smelted in the United States, according to figures compiled by the United States Geological Survey, 1949 short tons of antimony were produced, while from old alloys, scrap, dross, etc., 2506 short tons of antimony were recovered.

The Yampa smelter in Bingham canon, Utah, has been offered for sale. It was built in 1904 and for several years treated low-grade copper ores from the vicinity of Bingham. Later the company made an ore contract with the A. S. & R. Co. and the plant was closed, never to be reopened.

Gogo juice is used by the natives in the Philippines in panning gold-bearing sands. The juice is obtained from the bark of a plant and is neither alkaline nor acid. It "sickens" mercury and destroys its amalgamating power; but it is supposed to cause fine gold to settle quickly in a pan and not float away. According to experiments outlined in the Philippine *Journal of Science*, gogo juice has no qualities which would be beneficial in the concentration of gold, but the effect of a spray of the juice on the surface of water in a pan is to cause floating particles of gold to sink. Water sprayed on the pan would have the same effect.

A New Electric Steel Casting Plant

A Stassano Furnace Installation in California

By Ernst M. Schmelz

The first electric furnace of the pure Stassano type in the United States has just been installed by the Warman Steel Castings Company in their foundry at Redondo, Cal., and is

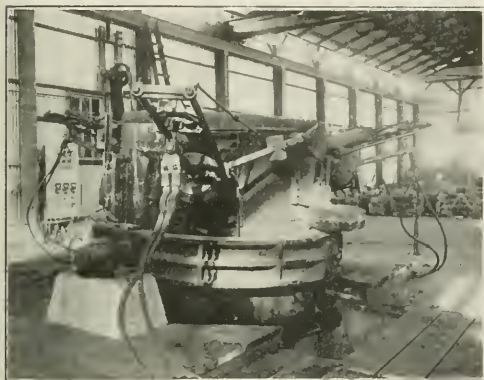


FIG. 1.—ELECTRIC STASSANO STEEL FURNACE

now in daily operation producing small and medium-sized steel castings.

This firm, which was organized three years ago, has hitherto been using Milwaukee-type crucible furnaces with a capacity of 3 tons per day. While the comparatively low price of fuel oil in California would seem to favor this process, this advantage is more than off-set by the higher cost of crucibles and labor.

One of the principle considerations in changing to the electric furnace was the fact that the firm was anxious to extend their trade to produce castings weighing 1600 lb. which is almost impracticable in a small crucible plant. An extended investigation of the various types of electric furnaces, with their particular requirements in view, decided the management on the Stassano type. An important feature influencing their decision was the adaptability of the Stassano 1-ton type for three-phase alternating current, doing away with the expense of a rotary transformer, the use of which is unavoidable if three-phase currents are used with a single-phase furnace.

The distinguishing feature of the Stassano furnace is that the metal is fused by the radiant heat from an electric arc over the metal and in a closed chamber providing a neutral atmosphere for the chemical reactions which take place in the process of refining. This permits of greater accuracy and thoroughness in closely following specified analyses in the production of castings.

The furnace in Redondo, shown in Fig. 1, is built according to the latest Stassano patents. It combines the well-known features of a completely closed melting chamber with a neutral atmosphere, the arrangement of a three-phase electric arc above

the metal, preventing the carbon and all other electric impurities from entering the metal, and allows a thorough mixing of the bath during any desired length of time.

This mixing is accomplished in a way entirely different from the older types of Stassano furnaces. Fig. 2 will explain the construction. A pair of bevel gears placed underneath the furnace and driven by a small electric motor causes the rotation of a shaft which is affixed to the center of the furnace bottom and thereby keeps the axis of the furnace always inclined at an angle to the perpendicular. The furnace is suspended by two trunnions in a horizontal steel ring, and this ring, in turn, swings by means of another pair of trunnions from supports in the same plane, but 90 deg. around from the first pair of trunnions.

This double suspension, which is the same as is used in a ship's compass, transforms the rotating movement of the furnace bottom into an oscillating movement of all furnace parts above the plane of the trunnions. In this way the water connections for the cooling of the electrodes and for the electrode hydraulic control, as well as the cable connections between switchboard and furnace, are extremely simple, while the old type of rotating furnace was rather complicated in this respect.

The furnace has a capacity of 2000 lb. to a charge and can be tilted to pour either the entire charge at once into a bull ladle or as required into small handle ladles for the pouring of small and thin-walled castings.

The electric current is supplied by the Southern California Edison Company, in the form of three-phase alternating currents at 10,000 volts and 60 cycles, and will eventually be brought down to 100 volts and 150 volts respectively by three single-phase transformers, arranged in the so-called interconnected star. Temporarily, however, until these transformers are ready, two sets of three single-phase transformers are installed, each having a ratio of 10,000 volts to 2200 volts and 2200 volts to 110 volts respectively.

As raw materials are used the mis-run castings, risers and gates, which accumulated in great quantities from the crucible process, and a small percentage of shearings, etc., supplied by local machine shops

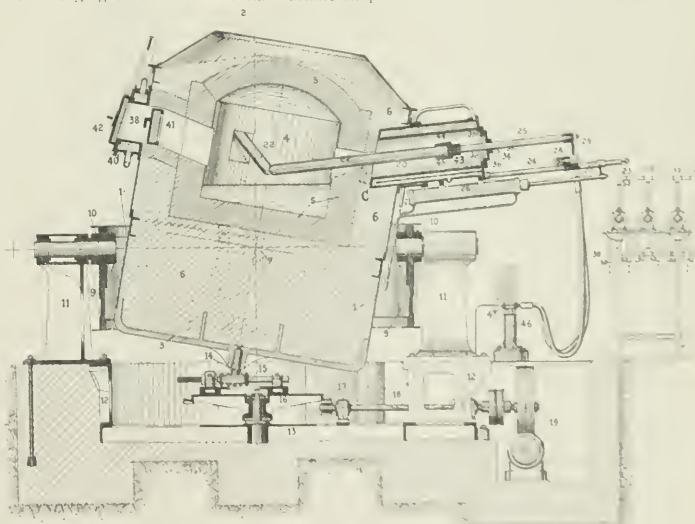


FIG. 2.—VERTICAL SECTION OF STASSANO FURNACE

The first heat from the furnace was taken on Oct. 27, and showed remarkably good results. The wattmeter installed by the power company was unfit for the control of the operations, its smallest unit being 200 kwh, but there is a recording wattmeter in the line which keeps an exact record of the operation.

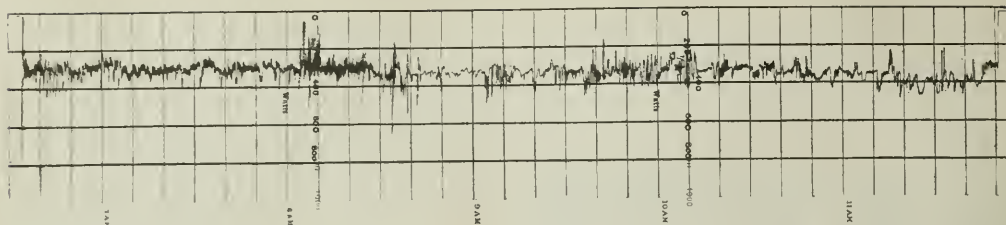


FIG. 3.—POWER CURVE DURING THE TWO FIRST CONTINUOUS

The diagrams of two consecutive heats taken off on Oct. 27 are reproduced in Fig. 3. They show, in the first place, how steady the demand of the furnace is. In spite of the fact that during the whole time the regulation of the furnace was operated by unskilled men, who until the day before had never seen an electric furnace, the demand varied hardly more than 10 per cent above or below the average.

The current consumption per ton, taking into consideration the transformer loss, figures 964 kw-hours and 1009 kw-hours per ton of charge respectively, and the time for each heat starting from a cold charge was 5 hours and 30 minutes and 5 hours and 1 minute respectively. Considering the lack of skill on the part of the men, and the fact that the current-saving arrangement of the new transformers was not yet installed, this record must be called a very good one. The writer hopes in the near future to report a further reduction, both in time and current consumption at this plant.

The metal, which proved to be extremely hot, poured to the last ladle in small castings for automobile parts. Neither a chemical analysis nor a test of physical properties has been made at the time of this writing, but the fractures at the gates showed a good grain and the castings appeared to be sound. Unfortunately the writer was unable to make further tests, being called away for the completion of another plant of the same kind in an Eastern foundry, which is expected to start operation before the end of the year.

Los Angeles, Cal.

Cost of Transportation to South America

Foreigners contemplating the purchase of American goods want to know the cost of the article delivered, and the inability or unwillingness of American exporters to quote c. i. f. prices has lost them many orders. To assist exporters in ascertaining delivery charges, the Bureau of Foreign and Domestic Commerce has published a bulletin compiled by Commercial Agent F. J. Sheridan dealing with transportation rates to the west coast of South America, this section of the foreign field being selected because of the trade activity that is expected to follow the opening of the Panama Canal. The main part of the bulletin is a series of tables giving the freight rates on over 800 articles from inland points in the United States to inland points in Ecuador, Peru and Chile. Freight rates per 100 pounds, in carload and less than carload lots, are given to New York from Chicago, St. Louis, Cincinnati, Indianapolis, Cleveland, Detroit, Buffalo, Pittsburgh, Boston, Providence and Hartford; ocean freight rates per 100 pounds and per cubic foot from New York to Guayaquil, Callao, Mollendo, Antofagasta and Valparaiso, via the Straits of Magellan and via Panama, and freight rates per 100 pounds inland from Guayaquil to Quito, Callao to Lima, Mollendo to Arequipa and Valparaiso to Santiago. Lighterage, transfer and other charges at the port of New York and at South American ports are shown, together with the cost of marine insurance, and data are also given as to consular fees and regulations and steamship requirements. Comparative tables give the competitive rates from European ports and from New York to South America. In addition to these features, statistics are furnished to show the character of the goods sold by the United States to Ecuador, Chile, Peru and Bolivia and for each of

these countries general information is given as to area, population, language, currency, postage rates, foreign trade and distance from New York. Tables of price comparisons give the equivalents in the currency and measures of these countries of prices stated in the money and measures of the United States. The bulletin, in short, is a compendium of practical information for the assistance of merchants who are engaged in the export trade or who contemplate entering the foreign field. Copies of the bulletin (Special Agents' Series No. 72) may be obtained from the Superintendent of Documents, Washington, for 10 cents each.

A New Type of Electrical Furnace for the Reduction of Ores

By Francis Louvrier

In view of the many well known advantages of electricity as a source of heat, the slow development of electrometallurgy is rather surprising. Of course, numerous obstacles had to be surmounted in this development.

Most electric furnaces in use at the present time can use only

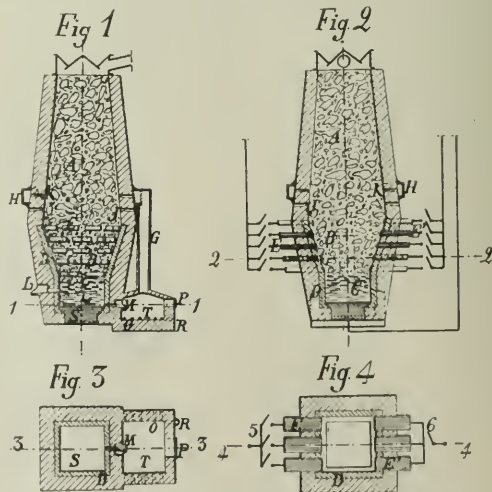
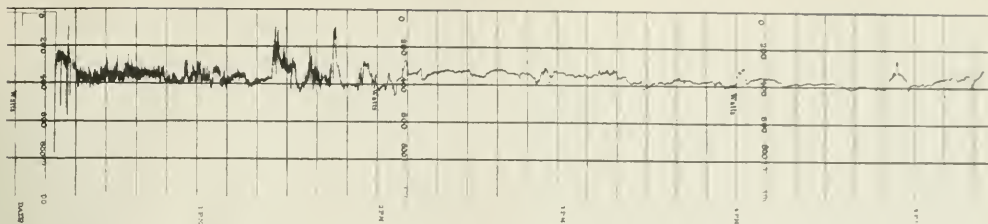


PLATE I.—SECTIONS OF FURNACE DESIGNS FOR IRON, COPPER, AND LEAD ORES

a small power, and their output is limited mainly on account of the shape of the furnace and the arrangement of the electrodes.

Temperature regulation is difficult, if carried out, by means of movable electrodes, as these are heavy and usually immersed in the charge. While it may not be easy, yet it is always possible to remove the electrodes further away from each other, so as to lower the temperature in the furnace. But the reverse operation—bringing the electrodes nearer together in order to increase the temperature—may be impossible because the charge is in the way.

In certain furnaces the electrodes are put on a slant and have to bear the weight of the charge. There is the danger



RUNS OF THE ELECTRIC STEEL FURNACE

of breaking at any moment. Further, the thermic efficiency of many electric furnaces actually in use is unsatisfactory: their consumption in electrode material is too high, and so on. Difficulties of this kind have held back the development of electrometallurgy. To overcome them has been the purpose of the furnace design which is the subject of this article.

This type of furnace, which is the invention of F. Louvrier and C. Louis, is characterized by the use of electrodes which are practically fixed, and the regulation of temperature is accomplished by simple manipulation of switches, by means of which the number of electrodes in circuit or the position which they occupy toward each other can be varied.

The general arrangement of the furnace is indicated by Plate 1, which represents a polyphase furnace.

Fig. 1 is a vertical section along 3-3 of Fig. 3. Fig. 2 is another vertical section along 4-4 of Fig. 4. Fig. 3 is a horizontal section along 1-1 of Fig. 1. Fig. 4 is a horizontal section along 2-2 of Fig. 2.

The furnace proper consists of three principal parts—the shaft *A*, wherein the progressive heating of the ore takes place, the bosh *B*, which is the zone of reduction, and the crucible *C*, where the molten metal accumulates. The crucible and the reduction chambers are lined with very refractory and non-conducting material *D*. The bottom *S* of the furnace is conducting and is connected with one of the phases of the current. It is made from varying material to suit each particular case; for instance, for iron ore reduction it is made of a piece of cast iron, with some cooling arrangement.

A series of openings are horizontally disposed at various heights in the two opposite walls of the bosh. Through each of these openings passes a well-fitting electrode. These electrodes form two symmetric series, opposite to each other. Each of the series corresponds to a phase of the current and each electrode in the series can be connected with its corresponding phase by means of a switch.

In smaller furnaces each electrode takes up the whole width of the furnace; in larger furnaces two or three electrodes are arranged at the same horizontal height side by side, as shown in Fig. 4, so that the size of each electrode remains within reasonable limits.

The electrodes of each series are insulated electrically from the other series by the furnace wall, which is non-conducting. The electrodes are pushed through the openings into the furnace so that they touch the charge without being immersed in it. The small space between the furnace wall and the electrode is luted in order to avoid leakage of the gas formed inside the furnace.

The top of the furnace is provided with the charging arrangement, a safety valve and the necessary pipes for the removal or recovery of the furnace gases.

When the carbon monoxide which develops from the reduction of the ore is to be burned in the furnace itself, the hot air necessary for the combustion is blown in through the tuyeres *J*.

The electric current passes through the charge between the two series of lateral electrodes and between these and the bottom electrode. The charge acts as resistor, and as all parts of the charge are traversed by the electric current, a practically uniform temperature is obtained.

Evidently the current passing through the charge and consequently the temperature in the furnace depend on the number of lateral electrodes that are connected to the circuit and on the relative position which they occupy.

The temperature is always easily regulated by a simple movement of the electrode switches. The electrodes themselves are, therefore, immovable or nearly so, all that is necessary is to push them in from time to time in order to compensate for their consumption. As air cannot enter the furnace and as no hot gases circulate around the electrodes, this consumption is relatively small and the compensation need only take place once or twice in a week.

The crucible *C* is provided with two tap holes, one for the slag and the other, *M*, for the metal.

For operation with single phase or continuous current the furnace is arranged in the same way. One possible arrangement is a conducting sole connected to one pole, and a series of lateral electrodes occupying two opposing walls of the furnace and all being connected to the other pole of the circuit. Another possible arrangement uses two lateral opposite series of electrodes connected to the two opposite poles of the circuit, without a conducting sole.

In plate 1, Fig. 3, a refining crucible *T* with accessories (tuyeres *O*, chimney *G*, etc.) is shown, this is intended for the refining of copper.

This furnace has the following advantages: Simplicity of construction; ease of regulation of temperature by a single movement of the electrode switch; the possibility of using a high voltage, such as 120 volts in small furnaces, 240 volts in medium-sized furnaces, and 350 to 400 volts in large furnaces, according to the ore under treatment. These are the usual supply voltages, so that the electric furnace may be connected directly to the lighting and power circuit. These furnaces can be built in large sizes, even above 40,000 hp, according to the inventors. For instance, a large three-phase furnace may have a reduction chamber *B* 5 meters (16 ft.) wide on top and 3 meters (10 ft.) at the bottom. There are above each other five horizontal series each of three electrodes with dimensions 0.25 meter and 1.10 meter (0.8 and 3.6 ft.). The section of the electrodes per phase is therefore $25 \times 110 \times 3 \times 3 = 41,250$ sq. cm., which allows of the passage of a current of at least 50,000 amp (if we allow 120 amp per square centimeter).

If in such a three-phase furnace the voltage between the phases is 400 volts, the power is $50,000 \times 400 \times 1.732 \times 0.925$ watts or 43,000 hp, the power factor of the furnace being assumed to be 0.925.

A similar furnace operated with single-phase current would have a capacity of 50,000 hp and with continuous current it would exceed 54,000 hp.

A furnace of 40,000 hp could produce, according to the inventors, 500 to 550 tons of pig per day; it could handle 220 to 280 tons of copper ore, or 600 tons of 50 per cent. concentrate.

Another advantage of the design is that the furnace is completely enclosed so that no gases can escape nor is the outside air able to enter.

The consumption of electrode material is considerably reduced since the hot gases do not circulate around the elec-

trodes and these latter can only be used up at their ends. With the exception of the ends the electrodes are out of reach of the high heat and of combustible gases, air and oxygen.

Considerable reduction in manual labor results from the large capacity of the furnace and from the facility of regulation: on account of the small consumption of the electrodes these have only to be readjusted in their positions once or twice a week.

Heat losses are avoided, except the loss by radiation, and this is small on account of the large size of the furnace.

The heat carried upward by the gases is utilized in the upper portion of the furnace for preheating the charge.

The gases, as far as they are combustible, can be burned in the furnace itself and their inherent heat may thus be utilized. Or it is possible and easy to recover completely the gases in case this is desired.

On account of the advantages just mentioned the efficiency of the furnace is high.

A uniform temperature is obtained in the whole part *B* of the furnace so that the reduction zone comprises the whole of part *B* and is not limited to a small space between two or more electrodes of small cross-section.

A uniformly progressive rise in temperature is obtained from the top of the furnace toward the lower part of the reduction zone.

The operation of the furnace is absolutely continuous, without interruption for the replacement of electrodes.

REDUCTION OF IRON ORES AND PRODUCTION OF PIG IRON OR OF CRUDE STEEL

For the reduction of iron ores the furnace is used as shown in plate 1, with or without supplementary crucible. This crucible can be installed where the direct production of cast iron or crude steel is intended.

The upper part of the furnace may be provided with tuyeres *J* for supplying the hot air necessary for the combustion of the carbon monoxide which rises upward from the reduction zone in the furnace.

The consumption of electrical energy in a Louvriér-Louis furnace which treats 50 per cent ore should not exceed 14-hp year per ton of pig iron produced, and would be reduced to 15-hp year when the carbon monoxide is burned inside the furnace.

The expense for electrodes should not exceed 8 cents per ton of pig produced.

The advantages may thus be summed up:

1. Possibility of transforming smelting furnaces in actual use into electric furnaces
2. Output as large as in blast furnaces: a large electric furnace can easily produce 600 tons of pig iron per day.
3. Better quality of product, since the electrical energy does not introduce any impurity (sulphur, phosphorus, etc.), which is always contained in the coke.
4. Possibility of directly producing crude steel, by having the molten metal during its reduction in contact with a smaller amount of carbon, and by employing pure ores.
5. Reduction of treatment cost, electrical energy, electrodes, reducing carbon, flux, manual labor, general expense, upkeep, amortization, etc., which, as estimated by the inventors, can be reduced to \$5.20 per ton of pig in countries where cheap water power is abundant and where reducing carbon can be obtained at \$7.50 per ton.
6. Possibility of using the heat of the carbon monoxide which rises during the reduction of the ore.

Before adopting the furnace just described, Messrs. Louvriér and Louis had patented and recommended another type of furnace which was characterized by its movable vertical electrodes between the charging stacks. This type of furnace was offered by its inventors in 1908 to the Noble Electric Steel Company for its Heroult plant at Pitt, Shasta County, California.

This now ancient Louvriér-Louis type is in actual use at the Heroult plant at Pitt, with some slight modifications in

the shunting of the current, which were introduced by Mr. R. E. Frickey (see *Metallurgical and Chemical Engineering*, July, 1913).

This type of furnace with charging stacks has been abandoned by its inventors since 1910 and has been replaced by the furnace above described, which gives much superior results.

REDUCTION OF COPPER ORES

The general arrangement of the furnace is exactly as shown in plate 1. The lower part of the stack *A* is fitted with tuyeres *J* which supply hot air which is required for the oxidation of the ore and the combustion of the sulphur.

A crucible *T* serves for the refining of the matte. This crucible, which is built close to the furnace, communicates with it by an ordinary tap hole *M*, which can be opened or closed from the outside. This crucible is provided with a door *P* made from refractory material. This door also serves for introducing into the crucible the materials (silica, etc.) which are required for refining the matte.

Tuyeres *O* are arranged in the lower part of the crucible for blowing in hot air into the center of the matte.

The hot gases resulting from the refining can be led off into the upper part of the furnace by the chimney *G* into the annular space *H* and through the openings *K* in order to utilize their heat.

The crucible works intermittently while the furnace operation is continuous. It is also possible to use two crucibles as described and placed symmetrically one on each side of the furnace.

The consumption of electrical energy per ton of ore (with 5 to 6 per cent of copper) should be between 0.06 to 0.03-hp year, according to the nature of the ore and its sulphur contents. The cost of electrodes should amount to about 2 cents per ton of ore treated. The inventors claim the possibility of treating in one furnace 2000 to 2500 tons of medium-grade ore per day with a considerable reduction of operating costs. Including general expense and amortization these are estimated by the inventors to amount to \$1.20 per ton of ore treated in the case in which the smelter produces its own electrical energy from water power.

TREATMENT OF LEAD ORES

The arrangement of the furnace is almost the same as described for copper ores. We will, therefore, refrain from further description.

REDUCTION OF ZINC ORES

Plate II shows the arrangement of the furnace in this case. The interior of this furnace is divided into two compartments by a horizontal refractory partition with a central opening for the passage of the ore. The lower chamber forms the actual furnace, in which the reduction of the ore takes place, while the upper chamber serves for the drying and the preheating of the charge. The partition also serves to economize space and facilitates the withdrawal through the lateral openings *O* of zinc vapors and of the carbon monoxide produced in the furnace.

These openings *O* lead into side chambers *R* and *R'*, built onto the walls of the furnace (Figs. 2 and 3 of plate II). In these chambers the vapors of zinc are separated from the carbon monoxide before they pass into the condensers. The separation takes place by gravity only: the heavy zinc vapors sink to the bottom while the carbon monoxide goes to the top and passes into the upper portion of the interior of the furnace.

The use of these side chambers is further intended for the lowering of the temperature of the zinc vapors to a degree intermediate between the temperature of their production inside of the furnace and between the liquefaction in the condenser.

In order to effectively separate the carbon monoxide gas and the zinc vapor it is necessary that the quantity of gases and vapors entering is equal to the quantities escaping at any time.

For this purpose a valve *V* regulates the escape of the zinc vapors to the condensers through the tubes *H* and *K* (Fig. 2) and the adjustment of the valve is based on the indications obtained through the two taps *N* and *Z*, which are placed at

convenient heights in the lateral chambers. When the furnace works normally, the upper tap *X* must give carbon monoxide on opening; the other tap *Z* must show zinc vapors.

When the furnace works normally the proper adjustment of the valve *V* is easily found by experiment, for the production of gas and zinc vapors inside the furnace remains fairly constant.

The temperature of the condensers is uniformly maintained between 600 and 700 deg. C. The consumption of electrical energy per ton of 50 per cent zinc ore amounts to 0.2-hp year.

The expense for electrodes amounts to about 8 cents per ton of ore.

These advantages are claimed by the inventors:

Zinc is obtained as molten metal and not as "blue powder"—the proportion of blue powder does not exceed 3 per cent of the metal.

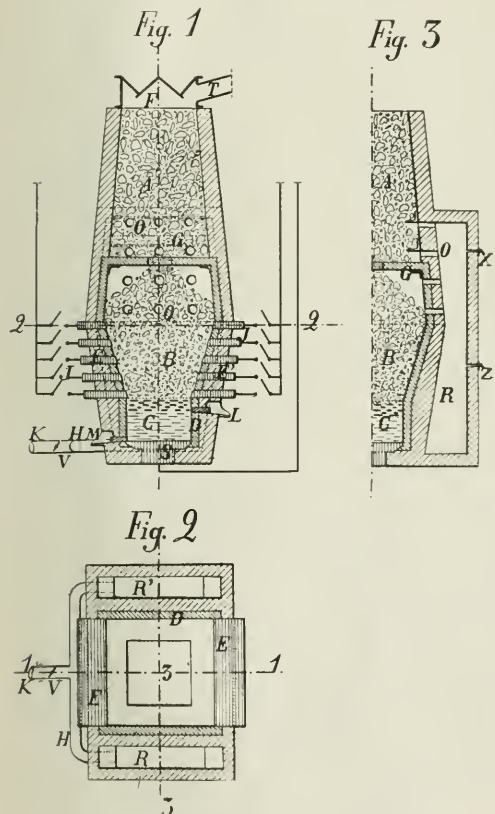


PLATE II.—SECTIONS OF ELECTRIC ZINC FURNACE

The better condensation of the zinc vapors is facilitated by their separation from the gases of reduction and by the previous lowering in temperature in the side chambers.

Most of the faults are suppressed which cause losses in metal so that a yield of at least 96 per cent of the zinc contents of the ore is obtained.

The chemical reactions are not disturbed by foreign factors, like air, moisture, etc., which may be contained in the charge, as these are completely removed before the ore arrives in the reduction zone.

The operation is continuous and adapted to large-scale working—600 tons can be treated per day by a single furnace.

Further advantages claimed are improved thermal efficiency, reduction in cost of handling, diminution in the amount of

reducing carbon and the possibility of roasting the ore without expense for fuel, by utilizing in the upper part of the furnace the heat developed from the combustion of carbon monoxide which rises upward from the reduction zone.

These advantages result in a reduced operating cost. The total cost of treatment, including general expense and amortization is estimated by the inventors at \$5.25 per ton of ore treated in case the electricity is generated by means of water power.

In a paper presented at the recent meeting of the American Electrochemical Society in Denver Messrs. Dorsey A. Lyon and Robert M. Keeney arrived at the following conclusions: "The chemical conditions leading to the formation of blue powder are due largely to the speed of reduction of zinc oxide in the electric furnace and to the presence of volatile impurities in the charge. . . .

"In the electric furnace, the temperature directly beneath the electrode in either the arc or the resistance furnace is so high that the reduction proceeds so rapidly that the large amount of carbon dioxide formed does not have a chance to be reduced to monoxide by the hot coal of the charge before entering the condenser.

"Hence there is oxidation of the zinc.

"By the use of electrodes of large cross-section it is probable that this concentrated heating could be reduced so that, rather than a very high temperature at one place there would be a uniform comparatively low temperature all over the charge. . . . Or the current could possibly be distributed more evenly by flat electrodes of rectangular cross section."

It can easily be seen that the Louvrier-Louis furnace fulfills all the conditions mentioned by Messrs. Lyon and Keeney.

In a paper presented at the same meeting Mr. Woolsey McA. Johnson stated that the electric reduction of zinc ore would be solved if the following conditions could be properly fulfilled:

"1. A first-rate preheater, mechanically strong and machine operated. Preheating *per se* has no advantage save to reduce kw-hours per ton of ore. Preheating must be done right, not too hot and not too cold, with a progressive application of metallurgical means.

"2. The flow of heat through the charge in the electric smelting furnace and slag must be under perfect control. The electricity must be controlled and care must be taken that the zinc globules in eductor assume no static charge otherwise coalescence of the spherulites is hindered.

"3. A regular flow of pure zinc gas must be delivered at proper temperature to the condenser.

"4. The condenser must be durable and under good metallurgical control."

Finally, Mr. Peter A. Peterson at the same meeting summed up his conclusions on this important problem in the following manner:

"It is necessary to have the charge absolutely dry. The presence of water even in small quantities is serious, due to the oxidizing of the zinc vapor by decomposition of the water. What is still more serious is the instantaneous generation of steam and other gases, causing a great increase in the velocity of gases, thus disturbing the condensation of the zinc in the condensers and thereby causing a loss of zinc and also a loss of silver and other metals, due to furnace charge being blown outward into the condenser.

"If it is not possible to mix the charge with hot calcines from the roasting furnace and then immediately feed it to the electric furnace, there must be a preheating furnace to heat the charge before introducing it into the electric zinc furnace."

All, or nearly all, of the conditions mentioned by Messrs. Lyon and Keeney, Johnson and Peterson are practically fulfilled in the Louvrier-Louis furnace.

Most of these conditions have already been mentioned by the writer in the article entitled "Causes of the Practical Non-success of Electric Furnaces in Treating Zinc Ores," which appeared in *Metallurgical and Chemical Engineering* of November, 1912.

Notes on Chemistry and Metallurgy in Great Britain

(By our Special Correspondent)

The New Headquarters of the Institution of Civil Engineers

The new building of the Institution of Civil Engineers has been almost completed. It is a very elegant structure, with a frontage of about 170 feet to Great George street, and 140 feet to Princes street when the whole design shall have been carried out.

The building is constructed of steel framework with an exterior of Portland stone, with the main entrance, which is ornamented with some very fine sculpture, in Great George street. The north and south reading-rooms are on the ground floor, and the latter apartment gives access to the council chamber and committee room; and a small hall also occupies part of the western side, while the eastern side is devoted principally to offices.

The main library is on the front of the first floor, has a length of 147 feet and a width of 28 feet 6 inches, and the old oak fittings and panelling from the building recently demolished have been used as far as possible, whilst the new panelling required, in consequence of the larger size of the new library, has been made to correspond in style. On the second floor is a similar upper library.

The chief apartment of the building is the great hall, which has a length of 100 feet and is also 45 feet in height on the Princes street side. Its mural decorations are in marble, and the ceiling is finely decorated. Three large hinged panels divide the hall from the lecture theater, which is almost exactly on the lines of the theater of the old building, and is of the same dimensions. The old oak dado was removed and has been utilized in the new theater which very closely resembles, even to the dome lighting, the theater of the old building so familiar to engineers from all parts of the world.

Electric lighting is used throughout the building. The heating is mainly by steam radiators, and adequate ventilation has been provided everywhere.

Oil-Shale in Skye

From details recently published by the Geological Survey it appears that the shale discovered in the Island of Skye is of considerable extent and reaches from fully five miles north of Portree to about the same distance south of that place. How far the seam extends westwardly cannot be ascertained at present, but for a good way inland it is at a workable depth and the dip is slight. No outcrops have been located inland, and the deposit is below strata of other rocks. The outcrop south of Portree shows evidence of the action of heat in most places; but to the north more than a mile and a half of the seam has been found free from this defect.

The thickness of the shale-seam at the outcrops varies from 7 feet to 10 feet. Analysis of the shale from the coast of Skye gave 12.8 gallons of crude oil and 7.4 lb. of ammonium sulphate per ton; whilst a sample from the Island of Ramsay—where the first discovery of the shale was made—gave 12 gallons of crude oil and 6.2 lb. of ammonium sulphate per ton.

The chemist who made the analyses found the specimens to be so excessively weathered that he did not expect more than a very small yield, and the true value of the shale cannot be stated until borings are made to procure unweathered samples for analysis. It is certain that new samples from the bores would show a great improvement on the figures given by the outcrop specimens; but the profitable working of the seam depends on its being rich enough to compensate for the increased cost of coal, chemicals and transport which the situation of the deposit must entail.

The Supply of Nitric Acid in England

It is now several years since some of our leading chemists warned us that England should be made independent of foreign sources for the production of nitric acid and nitrates, both for fertilizing purposes and for the manufacture of ex-

plosives in war time; and at length there is a prospect that this country will soon be in a position to manufacture all the nitric acid which will be required.

Steps are being taken for the erection of works at Dagenham, on the Essex shore of the Thames, which will be capable of producing 12,000 tons of nitric acid a year from cyanamide, and to establish a plant with a similar capacity at Trafford Park, Manchester; in addition to which it is proposed to put down plant in Scotland to turn out 9000 tons of nitric acid a year, and in Ireland for an output of 3000 or 4000 tons. The production of cyanamide will be such as will permit heavy reserves being kept in store, but in case of necessity any waste products such as ammoniacal liquors could be utilized. It is understood that the whole of these works will be controlled by the Nitrogen Products & Carbide, Ltd.

The British Aluminium Company

The British Aluminium Company has entered into an agreement with the Burgh authorities of Burntisland to acquire some 40 acres of ground for the establishment of new works close to the town on a site outside the Borough boundary and near the road to Aberdeen, and also for the possession of further ground for waste products and ulterior developments. The new factory will be in proximity to the main line of the North British Railway Company and easily accessible from Burntisland Harbor. It is estimated that the new works and plant, which are to be completed within a year and a half, will involve an outlay of about £200,000, and that from 300 to 400 hands will be employed.

Engineering Imports and Exports

The returns issued by the Board of Trade show that for the nine months ended on the 30th September there were fair increases in the values of imports and exports of engineering materials and products as compared with the corresponding period of last year, but the statistics for the month of September are not so satisfactory, and exhibit a general decline in exports except for machinery and new ships. For the nine months the imports of iron and steel, including manufactures, amounted to £11,192,013, an increase of £1,950,693; and exports totalled £40,962,292, an increase of £6,413,021. Other metals, including manufactures, were imported to the value of £24,193,890, an increase of £1,962,947; and were exported to the value of £9,927,750, an increase of £1,288,405. Imports of machinery are put at £5,444,611, a rise of £448,782; and exports went up to £27,489,564, an improvement of £3,433,170. New ships reached £27,031 for imports, and £9,139,800 for exports, showing increases of £1,723 and £4,706,767 respectively.

Taking the figures for the month of September alone, imports of iron and steel increased £21,445, but exports decreased £531,817; imports of other metals rose £284,733, whilst exports fell £162,765; in electrical goods and machinery the imports were £27,429 higher, and exports were £109,052 lower; machinery showed an increase of £33,265 in imports, and an increase of £172,895 in exports; the value of new ships imported was greater by £2,407, and exports improved to the extent of £100,005.

The New Engineering Laboratories at Dundee

In the middle of October the new engineering laboratories at University College, Dundee, were opened by Sir A. B. W. Kennedy. In the north block, which is two stories high, are situated the hydraulic laboratory, drawing offices, lecture rooms, the library and rooms for the contemplated architectural department. The tank which furnishes the water supply for the hydraulic laboratory has a capacity of 10,000 gallons, and is placed in the central tower 80 feet in height, which contains the appliances for the ventilation of the building. The water pressure available corresponds to heads of 65 feet, 140 feet and 600 feet. The equipment of the hydraulic laboratory includes a flume 45 feet long by 3 feet wide, an inward flow pressure turbine, a high-speed Oddie-Barclay reciprocating pump, an electrically driven centrifugal pump throwing 450 gallons per minute and a Pelton wheel.

The eastern wing provides for the boiler-house, the heat engine laboratory, workshop, physical testing laboratory and a separate room for cement testing. The engine room contains a horizontal tandem compound steam engine of 100 h.p. at 100 r.p.m. with a boiler pressure of 250 lb. per square inch. Steam is supplied from a Babcock & Wilcox boiler and superheater with a Green economizer evaporating about 1500 lb. of water per hour under ordinary conditions; and an electrically driven Worthington surface-condensing plant operate in connection with the engine. There are also fans for forced draft and a carbon-dioxide recorder.

The National Gas Engine Company contributed a 30 h.p. gas engine and a suction gas producer; and there is a 14-16 h.p. Argyle Petrol engine. Space is reserved for a Diesel engine and a steam turbine which will probably be provided in the near future. The equipment of the physical testing and cement rooms includes a 50-ton single-lever Buckton testing machine, an alternating stress machine, elasticity and rigidity apparatus, and a cement testing machine. The total cost of the building and equipment amounts to £17,000, toward which the Carnegie Trust for the Development of the Scottish Universities contributed £10,000.

A Rapid Method of Estimating Phosphorus in Steel

Accurate results are claimed for the process devised by Haripada Bhattacharyya (*Journ. Soc. Chem. Ind.*, 1913, 32, pp. 738-739). The steel is dissolved and the yellow precipitate of ammonium-phospho-molybdate is obtained and filtered off in the usual way, and the precipitate is washed with 1 per cent. nitric acid until quite free from iron: a 1 per cent. solution of potassium nitrate is then employed to completely wash out the nitric acid, when the filter paper with the precipitate on it is placed in a wide-mouthed 200 c.c. flask and 20 c.c. of N/10 soda solution are added. The excess of soda is estimated with N/10 hydrochloric acid, using phenol-phthalein as an indicator. The sodium hydroxide reacts with the phospho-molybdate according to the equation $(\text{NH}_4)_3\text{PO}_4 \cdot 12\text{MoO}_3 + 24\text{NaHO} = (\text{NH}_4)_3\text{PO}_4 + 12\text{Na}_2\text{MoO}_4 + 12\text{H}_2\text{O}$.

Market Prices

October, 1913.

Tin opened at £186, 10s. and fell away to £184, 15s. on the 17th rose to £185, 12s. by the 9th, dropped a little and again rose, reaching £186, 15s. on the 16th, and has since remained lower at £184, 10s. or about. Closes at £182.

Copper.—Opening at £74 the price gradually fell until the 15th, when it was at £71, 15s., it then recovered by degrees, rising to £75, 2s., 6d. on the 21st, and after a loss of 15s. again reached £75 on the 27th. It closes at £72, 15s., od.

Haematite opened at 66/6 and kept this price throughout the month, closing, however, 63/3.

Scotch Pig opened at 60/6 and fell away to 59/1½ on the 9th; recovered to 58/9 on the 10th and fell again to 57/7½ on the 13th; after reaching 58/- on the 22d it again fell to 57/6 and closes 56/9.

Cleveland opened 54/6, but had reached 53/7½ on the 7th and 51/7½ on the 13th, it then rose a little, but weakly, and was down to 51/10½ on the 17th, and 51/7½ on the 27th, and closes 50/9.

Lead opened at £20, 15s., 0. and fell slightly to £20, 5s. by the 8th; recovered next day to £20, 10s. and to £20, 15s. on the 15th, remaining at the latter price, but closing £21.

	£. s. d.
Aluminium, ingots, ton.....	06. 0. 0
Alum, lump, loose, per ton.....	5. 10. 0
Antimony, Star Regulas.....	20. 0. 0
Borax, British refined crystal, cwt.....	18. 6
Copper, sulphate, ton.....	24. 0. 0
Copper ore, 10 to 25%, unit.....	12/9 to 13. 3
Ebonite rod, lb.....	4. 6
Hydrochloric acid, cwt.....	5. 0
India-rubber, Para, fine.....	3. 1½
Mica, in original cases, medium.....	3/6 to 6. 0
Petroleum, Russian spot, gal.....	0½

Quicksilver (Spanish), bottle.....	7. 5. 0
Sal ammoniac, lump, firsts, ton.....	44. 0. 0
Sulphate of ammonia, f.o.b. Liverpool, ton.....	13. 5. 0
Sulphur recovered, ton.....	5. 10. 0
Shellac, cwt.....	4. 14. 0
Platinum, oz., nominal.....	9. 5. 0
Tin ore, 70%, ton.....	£113 to 115. 0. 0
Zinc, Vieille Montagne, f.o.b. Antwerp.....	25. 10. 0

DIFFERENCES

Higher.	£. s. d.	Lower.	£. s. d.
Copper, ton.....	5. 0	Tin.....	7. 5. 0
Lead, ton.....	10. 0	Antimony, ton.....	1. 0. 0
		India-rubber, lb.....	5½
		Tin ore, ton.....	10. 0. 0
		Shellac, cwt.....	4. 0
		Zinc, sheet.....	1. 0. 0
		Haematite, ton.....	4. 0
		Scotch pig.....	3. 3
		Cleveland.....	4. 0

Recent Chemical and Metallurgical Patents

Gold and Silver

Solvent for Precious Metals.—Halogen cyanides have been found good solvents for gold and silver, and the bromo-cyanide process is extensively used in Australia. Care must be exercised in the preparation of bromocyanide, or bromine will be wasted. A new method of generating halogen cyanide in connection with an oxidizing agent has been patented by Dr. HANS FOERSTERLING, of the Roesler and Hasslacher Company, of Perth Amboy, N. J. The inventor melts sodium bromide and sodium cyanide in the proportion of one molecule of each, and allows the mixture to solidify. No decomposition takes place in this process. In order to convert this into bromocyanide, it is treated with an acid solution of hydrogen peroxide, and the reaction proceeds as follows:

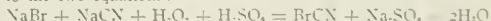


If an acid was not used, sodium hydrate would form, and destroy the bromocyanide. The rapidity of formation of bromocyanide according to the above equation depends on the concentration of the solution. The formation of the corresponding chlorine and iodine cyanides according to the same formula, is much more rapid than the formation of bromocyanide.

In the practical use of this invention in ore treatment, the mill operator would receive the correctly proportioned mixture of sodium bromide and sodium cyanide, and all that would be necessary would be to dissolve the salt and add an oxidizing agent. An acid solution of hydrogen peroxide is preferred for this purpose, because it is stable when formed by dissolving sodium peroxide in an excess of acid. Three parts of sodium cyanide should be used with one part bromocyanide in order to dissolve gold, thus:



Sufficient cyanide can be added originally to the alkali bromide to give a mixture that will act in a manner equivalent to the two equations:



The largest part of the alkali halogen is not lost in practice, and only so much need be added as to make up the small loss.

Combined Pulp Agitator and Thickener.—In the cyanide process, the steps of thickening and agitating the ore pulp in solution are carried on separately. In Fig. 1 is shown an apparatus in which it is proposed to perform the two operations within the same tank, thereby reducing the cost of apparatus and its maintenance. The device is the patented invention of Mr. J. V. N. DORR, and embodies the principles of his continuous thickener and an air-lift. A partition in the form of a truncated cone divides the tank into two compartments in which the different processes of decantation and agitation can be carried on without interfering with each other.

A flat-bottom tank A, with overflow launder H, contains a

partition E in the form of a truncated cone, dividing the tank into the compartments Y and X. Within the inner compartment is an air-lift C supplied with compressed air from pipe D. A central shaft K is operated by the mechanism ML, and carries at its lower end four radial arms J fitted with rabblers which tend to rake settled solids toward the center.

The proposed operation is as follows: The pulp is fed through launder B to an annular distributor G. The solids tend to settle, while the clear solution flows from the tank at the peripheral launder H. The settling solids are raked to the center by the arms J, where they are acted upon by the air lift, raised and discharged within the compartment X. Thus the solids, which require aeration and agitation to cause dissolution of their precious-metal content, are given the additional treatment required. A portion of the thickened and agitated pulp is

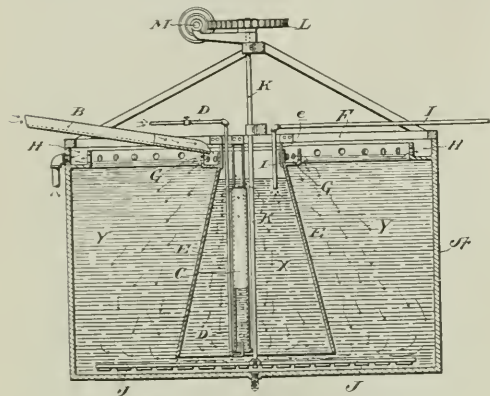


FIG. 1.—COMBINED THICKENER AND AGITATOR

constantly drawn off through the pipe I. Several modifications of the apparatus can be designed, introducing several concentric cone partitions, with a plurality of air-lifts. (1,076,666, Oct. 28, 1913.)

Copper

Apparatus for "Amphidizing" Ores.—In this journal, April, 1913, page 218, reference was made to the Bradley process for "amphidizing" copper ores, being a method of rendering copper soluble, as sulphate. In a recent patent granted to Mr. CHARLES S. BRADLEY, of New York City, the inventor gives a description of the "amphidizer" or oxidizing roaster used to convert the copper ores into soluble condition. The device is essentially a rotary kiln with interior passages so arranged that the ore passes from one end to the other and back again before it is discharged with its copper in soluble form. Heat is supplied through a central passage, and the ore coming in is heated continuously and in a regenerative manner due to the fact that the returning ore transfers some of its heat to the incoming charge. (1,077,036, Oct. 28, 1913.)

A chemical filter, specially adapted to the filtration of acid, corrosive pulps, is the basis of another patent granted to Mr. BRADLEY. As shown in transverse vertical section in Fig. 2, it is a flat filter bed with a foundation to and side walls 11 and 12 composed of concrete. The filtering medium is porous silica brick molded with suitable channels and passage ways connecting with the conduits 19, the latter being connected with sources of vacuum and pressure as desired. The pulp to be filtered is delivered into the channels 15 and 16 in the side walls, and flows onto the bed by way of the ports 18. Vacuum being applied, the solution is drawn through the brick filter into the passage ways and finally to the channels 19. The filtered pulp remains on the surface, and is removed by a traveling scraper which is set to remove all but a thin layer of the material. After this is accomplished, air may be forced through the filter bed from beneath, and the thin layer of

solids blown from the pores of the filter. This becomes mixed with the next charge, and the bed is in condition for another filtering operation. (1,077,037, Oct. 28, 1913.)

Metallurgical Furnaces

Improvement in Ore Roasting Furnaces.—In that type of ore roasting furnaces used for desulphurizing blende or pyrite, having an annular rotating hearth and stationary annular arch to which rabblers are attached, it has been discovered by

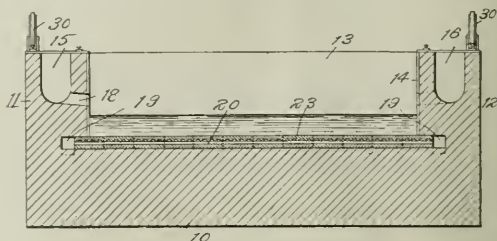


FIG. 2.—FILTER FOR ACID PULPS

NAVIER DE SPIRLET, of Brussels, Belgium, that incipient fusion of the sulphides may occur on the hearth at the point of feeding. M. de Spirlet proposes to obviate this difficulty by arranging ventilating passages in the roof or arch of the furnace and the suspended rabblers, through which air or gas can be passed for the purpose of removing some of the heat concentrated at the point of incipient fusion. (1,077,010, Oct. 28, 1913.)

Air-Cooled Rabble Arm.—The rabble arms used in roasting furnaces of the superimposed-hearth type are usually cooled by allowing air to pass through them by way of the hollow central shaft. An improvement in the manner of cooling has been patented by MAURICE VAN MARCKE DE LUMMEN, of Cologne, Germany, as shown in Fig. 3. The arms are made in one piece in the form of a hollow body containing a longitudinal partition *k*, which divides the interior of the arm into two compartments. The passage or compartment *l* registers with a central opening in the hollow shaft at *o*. Each passage *m* terminates at the central part of the arm in an opening that communicates with lateral passages *q* in the hollow shaft. By this arrangement the cooling air rises through the central part of the shaft, traverses the upper and lower passages of the arms, and makes its exit through the lateral passages in the shaft. (1,076,297, Oct. 21, 1913.)

Heat Regenerator.—In the use of the cheaper grades of gas fuel in metallurgical furnaces, difficulty is sometimes experienced in maintaining a sufficiently high temperature in the furnace to produce a neutral or reducing action. The object of an invention of Mr. THEODOR L. HOLLE, of St. Louis, Mo., is to overcome this defect by preheating the air introduced into the furnace to a high temperature. This is accomplished by a plurality of circuitous passages or flues alternately arranged

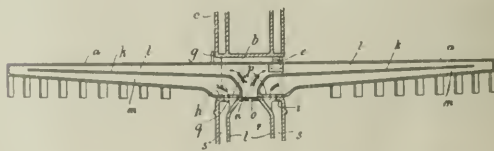


FIG. 3.—AIR-COOLED RABBLE ARM FOR ROASTING FURNACES

for the passage of air and gases of combustion in opposite directions, so that the outgoing products of combustion passing from the furnace to the stack will highly heat the flue walls, and the incoming air traveling in the opposite direction in adjacent flues will receive heat in proportion to the temperature of the walls. A relatively large number of flues are arranged in a small space, and special structural units are used. (1,076,426, Oct. 21, 1913.)

Water-Cooled Port for Open-Hearth Furnace.—In Fig. 4 is shown a longitudinal section through one end of an open-hearth furnace, equipped with water-cooling means invented by Mr. ANSON W. ALLEN, of Birmingham, Ala. The furnace is of the usual construction, having a gas port 10 and air ports 11. These are separated by an arch 12 of refractory material extending to the point where the best combustion is obtained. Beneath the refractory arch is another composed of metallic

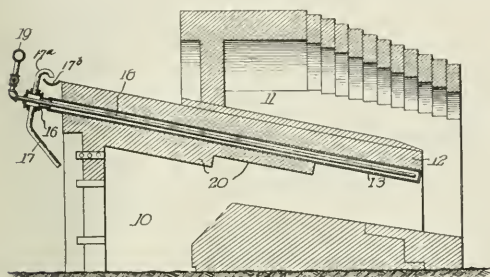


FIG. 4.—WATER-COOLED FURNACE PORT

units 13, these being trapezoidal in cross section and closed at their ends with plugs. The upper plug is apertured to receive a fitting 16, connected with an overflow pipe 17. An additional overflow pipe is provided at 17a, delivering into a trough 17b. Water is supplied separately to each unit through the header 19 and pipe 18. The cooling units are removable without delaying the operation of the furnace. (1,076,540, Oct. 21, 1913.)

Metallic Alloys

Copper-Aluminium.—An improvement in the manufacture of aluminium bronze is suggested in a patent granted to Mr. EDWARD D. GLEASON, of New York City. Poron is used in the preparation of the alloy. Into a fused bath of three parts calcium fluoride and one part fused boracic acid, ingots of aluminium are introduced. When the mass is molten it is stirred and cast into ingots. This mixture is then added in an amount slightly in excess of 10 parts to 90 parts of pure copper melted under a cover of charcoal and black oxide of manganese. The resulting alloy is free from oxide of aluminium, which is said to appear in the ordinary aluminium bronze. The alloy can be made in different degrees of hardness with the same percentage of copper by adding aluminium with more or less boron. (1,076,973, Oct. 28, 1913.)

Lead-Copper-Tin.—Plastic compositions of lead-copper and lead-copper-tin, such as are used for bearing metals, sometimes develop defects due to the lack of affinity of the lead for the copper or copper-tin, resulting in segregation or sweating of the lead. According to an invention of Mr. EDWARD D. GLEASON, such alloys can be made perfectly homogeneous by incorporating a non-metallic derivative of lead, such as the sulphide mineral galena. For example, he may melt 60 lb. copper and add thereto 5 lb. galena, and then gradually add to this mixture 30 lb. lead and 5 lb. tin. (1,077,698-699-700-701, Nov. 4, 1913.)

Malleable Zinc Alloy.—According to an invention of Mr. THOMAS A. BAYLISS, of Warwick, England, a malleable and tough alloy is produced from zinc, aluminium and lead, when the ingredients are present in proportions between the following limits:

Zinc	90.100% and 99.0%
Aluminium	0.001 and 0.1
Lead	0.010 and 0.0

The metal thus obtained is said to be extremely tough and strong, and is malleable and suitable for stamping. (1,074,234, Sept. 30, 1913.)

Hard Copper Alloy.—The following formula gives the ingredients of an alloy patented by Mr. WM. F. NOLAN, of St. Paul, Minn.: Copper, 98.15 per cent; manganese, 1.48 per cent; vanadium 0.25 per cent; aluminium 0.12 per cent. The claims

for the alloy include increased durability, and a degree of hardness which makes the metal useful for cutting tools. After the alloy is mixed, and while yet at dark red heat, it is plunged into oil.

The temper can be removed by heat, but the alloy can be hardened again by heating and immersing in oil. (1,074,285, Sept. 30, 1913.)

Alloy for Metal Tape.—A lead alloy suitable for use as metal tape in winding electrical conductors is patented by Mr. CHARLES P. MCCONNELL, of New York City. The alloy contains lead, antimony and tin in the following proportions:

	Preferred	Maximum	Minimum
Lead	95.00%	96.75%	93.25%
Antimony	4.50	6.00	3.00
Tin	0.50	0.75	0.25

A satisfactory tape made from this metal need be only 0.03 in. thick and can be 1 in. wide, and at the same time have as great or greater strength than tape made entirely of lead 0.006 in. thick and 0.75 in. wide. A material saving of lead is accomplished which is in excess of the cost of the metals added to make the alloy. (1,075,661, Oct. 14, 1913.)

Manufacture of Sodium Alloys.—EDMUND CHARLES RESSITER of Birmingham, England, alloys sodium with other metals in the nascent state. The temperature need never rise above 750 Centigrades during the whole process. To prepare an alloy of sodium and lead, for example, sodium hydrate is fused and reduced with carbon in the form of charcoal. The pot is covered and filled with dry coal gas or other suitable gas. The lead is added in liquid form before or after the reduction takes place, or the mixture of sodium hydrate and carbon is poured into the molten lead. (1,073,523, Sept. 16, 1913.)

Refractories

Refractory Material.—WALTER ARTHUR of Schenectady, N. Y., assigns to the General Electric Co. two processes of manufacturing a refractory material of valuable heat-insulating quality which will withstand temperatures of from 900 to 1100 centigrades or even higher. It has a thermal resistance of from 900 to 1200 thermal ohms per inch cube, the temperature coefficient being very small. It contains mainly silicon, carbon, oxygen, small amounts of metallic impurities and aluminium in one patent, titanium in the other. The charge consists of silica, coke, and either fire clay as an example of aluminium silicate or rutile in the second case; this is subjected to the smothered are of an electric furnace. A volatile product, representing the insulating material, condenses in the form of soft, coherent, somewhat felted masses of grayish or yellowish color; when containing manganese the color is a bluish gray. The apparent density is 0.60 to 0.16, the real density about 2.50. A typical analysis is 45.51% Si, 7.57% C, 40.13% O, 1.07% Ti, .32% Mn, 1.03% Al, 1.08% Fe. (1,072,413 and 1,072,414, Sept. 9, 1913.)

Composite Silicon Carbide Refractory Lining.—Silicon carbide is excellent as a furnace lining for its infusible and refractory nature, but has the disadvantage of high heat conductivity, being a conductor of heat five times better than firebrick. Mr. FRANK J. TONE, of the Carborundum Company, Niagara Falls, has devised a lining in which the working portion exposed to high temperature and requiring stability and strength, is formed of silicon carbide of high refractivity and low thermal resistivity, and backs this layer up with silicon carbide of a form which, although comparatively lacking in strength and refractivity has very high thermal resistivity. For the first named exposed or working portion he uses bricks made of dense silicon carbide, obtained by subjecting a mixture of silicon carbide and carbon to the action of silicon-containing vapors at high temperature. For the second portion he uses porous silicon carbide, obtained by subjecting porous carbon to silicon-containing vapors at a high temperature. This composite silicon-carbide lining may be backed up by ordinary means such as the steel furnace shell or by walls of less refractory brick. (1,078,525, Nov. 11, 1913.)

Sintering

Sintering.—Messrs. QUINCY BENT and EDWIN BARNHART, of Sparrows Point, Md., and JAMES B. LADD, of Ardmore, Pa., patent improvements in the preparation of ore for sintering. The sintering process consists in passing air through the materials, after ignition, thereby causing combustion through the mass to fuse or partly fuse materials in it and to get the mass of fine material into a more or less solid mass suitable for treatment in smelting furnaces. The combustion is supported either by combustible material originally in the fine ore or by carbon purposely added. The new point is that in sintering these fine materials, which must be in a moist state, much better results are secured by separating into small parts the more or less coherent mass of material and by lightly and evenly sprinkling or depositing these small and separate parts so that the entire mass to be treated shall lie loosely and be of uniform density and consistency. (1,078,988, Nov. 18, 1913.)

Iron and Steel

Vanadium in Iron.—Mr. JOHN KIRBY, of Pittsburgh, Pa., incorporates vanadium in muck iron to increase the tensile strength by the following process. He feeds into a heat of muck iron in a puddling furnace about the time the cinder is boiled off and before the iron is ready to ball, a composition including vanadium oxide with a reducing agent and an agent to increase the combustion of the reducing agent, so that the vanadium will be reduced and thoroughly incorporated with the iron. Such a composition may contain 32 parts of vanadium oxide, 8 to 16 aluminium, 8 iron rust, 2 potassium chlorate, 1 black oxide of manganese and 1 barium dioxide. (1,079,129, Nov. 18, 1913.)

Electric Furnaces

Melting Resistor Furnace.—Mr. ALBERT E. GREENE, of Chicago, Ill., patents a resistor furnace for melting. Claim 2 relates to "the combination with a furnace chamber, of electric resistors constituting the primary circuit for an alternating electric current flow, electric resistors arranged in inductive relation to said primary circuit and thereby constituting the secondary circuit, said electric resistors being arranged in proximity to said furnace chamber; whereby heat developed in said resistors is transformed to the charge in said furnace. (1,078,619, Nov. 18, 1913.)

Electrode Holder.—In Fig. 5 is shown a top view of a holder for electrodes used in arc furnaces. A side view is given in Fig. 6. It is designed to provide an easily adjustable support, and one which will make a good electrical contact with the electrode so that hot spots will be avoided. It is the invention of Mr. JOHN ORR, of Schenectady, N. Y., and the patent is assigned to the General Electric Co. The holder con-

sists of an adjustable collar consisting of a number of contact plates joined by links at the adjoining ends, and adapted to make a good contact with the electrode surface when drawn together. A bracket 1 forms a support for the electrode and is attached to the furnace framework. It is longitudinally adjustable by a screw 2 moving in a guide through a block 3. At

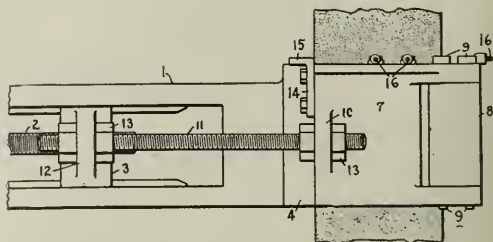


FIG. 6.—ELECTRODE HOLDER

the end of the bracket is a contact plate 4, which forms one part of the electrode holder. The other contact plates 5, 6 and 7 are adjustable with respect to the fixed contact plate 4, and preferably consist of some good conductor such as copper. The adjacent ends of the adjustable electrodes are connected by links 8 movably attached to the plates by pins 9. In order to give a good contact with the electrode, the contact

plates have the same configuration, round or square, as the electrode. When the plates 5 and 7 are drawn toward the fixed plate 4 by tightening the screws 11, they exert pressure on the enclosed electrode and provide a good contact. (1,076,054, Oct. 21, 1913.)

Induction Furnace.—LESLIE E. HOWARD, of La Grange, metallurgist of the Simmonds Mfg. Co., describes a new electric melting furnace for bronzes, brass mixtures, white metal, Babbitt, aluminium, iron and steel. The furnace is of the induction type and is characterized by the arrangement of the secondary circuit in form of a fused metal bath of wide cross section, which forms about one-third of the secondary, while two-thirds consist of a solid mass of metal having a very large cross-section relative to the cross-section of the molten portion of the secondary. This construction allows to approach as much as possible open hearth conditions for the molten metal and further to place primary and secondary circuit very close to the magnetic field with the resulting advantage of reducing magnetic leakage. The patent is illustrated. (1,076,887, Oct. 28, 1913.)

Electrolytic Furnaces

Electrolytic Furnace.—DAVID J. HAUSS designs an electric furnace adapted to produce metallic alloys or chemicals or compounds with separate regulation of the temperature at the anode and cathode. It also contains a separate chamber in

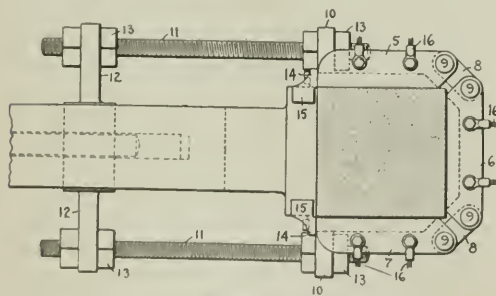


FIG. 5.—ELECTRODE HOLDER

sists of an adjustable collar consisting of a number of contact plates joined by links at the adjoining ends, and adapted to make a good contact with the electrode surface when drawn together. A bracket 1 forms a support for the electrode and is attached to the furnace framework. It is longitudinally adjustable by a screw 2 moving in a guide through a block 3. At

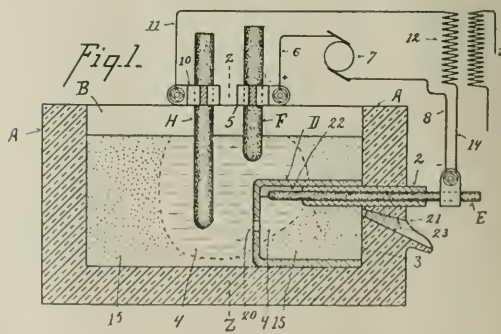


FIG. 7.—ELECTROLYTIC FURNACE

which the product accumulates and is preserved from alteration. The main feature of the patent is the combined application of alternating and direct current. Fig. 7 shows the electrodes E and F (the former one installed in the special chamber) connected to a source of direct current, while the electrodes H and I supply the alternating current for heating.

purposes. For the production of alloys the electrode F may be subdivided into two connected in parallel, one being carbon, and the other the alloying metal, a rheostat regulating the proper distribution of the current (1,060,839, May 6, 1913).

Metallic Magnesium.—For the production of metallic magnesium Messrs. ROGER WILLIAM WALLACE and EUGENE WASSMER, of London, England, electrolyze a fused mixture of magnesium chloride and magnesium sulphide in a vessel acting as one electrode with another electrode in the center. One of the advantages is that the nascent chlorine combines with the sulphide so as to form sulphur chloride which does not attack the vessel and electrode. The voltage is 8 to 10. The process is carried out in a closed vessel with a nitrogen atmosphere. (1,079,079, Nov. 18, 1913.)

Electrolytic Apparatus

Diaphragm.—A new combination of materials for an electrolytic diaphragm has been patented by Mr. HERMAN A. WAGNER, of New York City. It consists of a plate molded of a substance such as manganese dioxide, in which has been incorporated mercury and palladium black. The plate is used between porous walls of a substance such as burnt clay. In making the diaphragm, the manganese dioxide is powdered, mixed with mercuric chloride, and treated with an alkaline solution of formic aldehyde to reduce the mercury salt to metal, which is thus finely disseminated through the mass. Palladium black can then be added, and the whole molded and pressed into any desired shape. (1,077,444, Nov. 4, 1913.)

Electrode.—Metallic tungsten is insoluble in ordinary acid or neutral solutions, and can be used as a cathode in such solutions; but it is not useful as an anode because it does not readily allow the current to pass. Mr. ROYALE H. STEVENS, of Salt Lake City, Utah, has discovered that tungsten can be formed into a useful anode material by means of a very thin plating of a metal of the gold group, such as platinum, iridium, ruthenium or gold. The combination gives electrodes that are non-oxidizable and refractory, especially when used in electrolytic work where the current liberates chemically active gases at the anode. The same effect can be obtained by alloying the precious metal with the tungsten, but the plating is preferable as requiring less of the expensive metals. The plating can be accomplished with a current density of 0.05 ampere per square decimeter of the area to be plated. The electrolyte consists of 5 parts of platinum or iridium chloride, 250 parts of disodium phosphate, 50 parts ammonium phosphate, all in 1000 parts water. The solution is boiled until all free ammonia is expelled. (1,077,894-920, Nov. 4, 1913.)

Oxygen and Hydrogen.—In the electrolytic cell of RENE MORITZ, of Wasquehal, France, for the production of oxygen and hydrogen from water, the chief features are details of design by which the two gases produced are carried away to separate containers through independent and isolated ducts and any leakage from one to the other is prevented so as to insure high purity of the gases. Each electrode is bipolar, hydrogen being evolved on one side and oxygen on the other. Means for circulation of the electrolyte are provided. (13,643, Nov. 11, 1913.)

Mold for Electrotyping Processes.—In order to produce a good and lasting mold for silver plating in cyanide, ARTHUR J. TIZLEY brings strips of softened celluloid of required dimensions into intimate contact with the design to be reproduced. The celluloid mold is preferably made up of a plurality of thin sheets, and wire mesh is embedded into the celluloid as a reinforcing element. (1,063,740, June 3, 1913.)

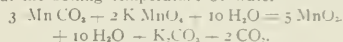
Electric Gas Reactions

Ozonizer.—NEVIL MONROE HOPKINS, of Washington, assigns to the Torpezone Co. of New York the construction of a high-voltage ozonizer. It consists of an inner metal cylinder as one electrode which is filled with a loose permeable mass of copper foil or shavings, an annular passage for the air to be treated, a glass cylinder as dielectric, bearing a heavy metal coating as the other electrode and radiators connected to this

electrode for cooling purposes. The whole apparatus is enclosed in a strong outer glass tube which is perforated in one place for the introduction of a binding-post. Properly constructed caps close the two ends and provide for the in- and outlet of the cooling air and heated air. (1,062,974, May 27, 1913.)

Batteries

Lelanche Cell.—For use as depolarizer in Lelanche cells, Mr. MORDECHAI L. KAPLAN recommends the use of dihydrated manganese peroxide, $MnO_2 \cdot 2H_2O$, produced by treating manganese carbonate with a highly diluted potassium permanganate solution at the boiling temperature of water



This dihydrated manganese peroxide consists of a fine, comparatively dense powder of a grayish blue color. Cells made up with it "show 1.8 volt and are superior with respect to yields and recuperative powers." (1,078,788, Nov. 18, 1913.)

Synopsis of Recent Chemical and Metallurgical Literature

Gold and Silver.

Charcoal as a Precipitant of Gold in Cyanide Solutions.

—A study of the causes of precipitation of gold from cyanide solutions by means of charcoal is presented in a paper by Mr. MORRIS GREEN, in *Bulletin* No. 109, Inst. Min. & Met. (London). The author considers three possible explanations of the precipitating effect of charcoal: (1) A chemical action in which the carbon participates; (2) a reaction due to osmotic pressure; (3) a reaction due to occluded gases. The results of investigations suggest that the first two assumptions are fallacies, and that precipitation is due to the latter cause.

On the general effect of charcoal, the author presents experiments in which he added freshly ignited charcoal to different portions of the same gold-bearing cyanide solution and agitated the mixtures for varying periods. After agitation for 3, 6, 9, 12, 15 and 18 hours, the various solutions were practically identical, as also were the amounts of gold carried by the charcoal. It is concluded that either the solution is incapable of dissolving the precipitated gold, or that precipitation and dissolution proceeded at the same rates. The charcoal will not precipitate unlimited quantities of gold. Gold which had been precipitated by charcoal was not readily dissolved by fresh cyanide solutions.

The effect of different forms of charcoal was determined. The table shows the percentages of gold precipitated by equal weights of different forms of charcoal, acting under similar conditions on equal quantities of the same gold-bearing cyanide solution.

Precipitating Agent

Precipitating Agent	Per cent gold precipitated
Freshly ignited wood charcoal.	90.3
Wood charcoal exposed in lumps for about 6 mos.	50.0
The above charcoal reheated to incipient redness	84.2
Sugar charcoal (by heating in closed retort)	62.1
Sugar charcoal (by burning in contact with air)	5.0
Sugar charcoal (by H_2SO_4 and washing out acid)	0.0

The effect of the presence of charcoal in a cyanide solution during the dissolution of metallic gold was to precipitate some of the gold, and establish a condition of equilibrium between precipitated and dissolved metal. This equilibrium was not changed after agitation for a number of hours.

All of the foregoing results establish, in the opinion of the author, the fact that carbon *per se* does not participate in the reaction. He further shows that graphite not only does not precipitate gold from cyanide solutions, but aids solution by the formation of an electric couple where gold and graphite meet.

A number of experiments were performed to test the possi-

bility or reaction due to osmotic pressure, but with negative results.

In testing the action of occluded gases, it was found that the removal of the loosely-held gases was not sufficient to destroy the precipitating effect of the charcoal; but that when charcoal was subjected to the action of a powerful vacuum, with the addition of heat, its precipitating effect was decidedly diminished.

Nature of Charcoal

	Per cent gold precipitated
(1) Charcoal exposed to air for about two weeks after manufacture	93.3
(2) The same charcoal (1) heated and exhausted of gases	17.0
(3) Charcoal exposed to air for about one year....	41.2
(4) The same charcoal (3) heated and exhausted of gases	14.8

"It appears, therefore, that the power of charcoal as a precipitant of gold from aurocyanide solution is due to occluded gases." Carbon monoxide and hydrogen were found in the exhausted gases; but since hydrogen was found in quantities too small to account for the observed effects, carbon monoxide is considered to be the active reagent.

Use of Lime in Cyaniding.—The almost universal adoption in cyanide practice of crushing ore in cyanide solution instead of water has introduced new metallurgical problems, one of which is the best method of adding lime to the ore. Mr. A. M. MERTON discusses the general use of lime in cyaniding in *Mining Science*, September, 1913. He makes the rational criticism that few mill designers make adequate provision for handling lime in the mill or for the manner of its addition to the ore. The result is the ineffective use of an expensive reagent.

The functions of lime in cyaniding are classified thus:

- (1) Neutralizes acids formed by the oxidation of sulphides, etc.
- (2) Breaks up metallic salts, forming innocuous hydrates.
- (3) Protects the cyanide solution against carbon dioxide.
- (4) Furnishes alkali to break up silver minerals.
- (5) Protects cyanide solution against latent cyanicides in the ore.

The effect of actions (1) and (2), and to a certain extent (5), becomes apparent during crushing and grinding. The amount of lime to be added for these purposes cannot be determined in the laboratory. The amount of protective alkalinity to be carried must be determined experimentally in the mill. In the case of Mexican silver ores it is common to have the solution at all times a concentrated lime solution.

Lime losses occur as follows:

- (1) Lime-consuming constituents in the ore.
- (2) Lime destroyed by CO_2 in air used in agitating.
- (3) Mechanical loss in solution discharged with tailing.

The author makes a point of the fact that the mere addition of lime to the ore in quantities indicated by laboratory tests will not suffice always to protect the cyanide solution. In the case of an all-sliming treatment, for instance, although the pulp going to the agitators is alkaline, the tube mill grinding classifier sand may be working in acid solution because the lime added at the head of the system is passing from the classifier in the slime, leaving an unprotected sand to be reground. In such circuits it is necessary to see that the solution is properly alkaline at all points, and for this purpose it may be necessary to add the lime at different points instead of at the beginning.

Lime added at the coarse crusher may be only partly effective owing to the impossibility of neutralizing the acid conditions in the larger pieces. If only a part of the lime were added here, sufficient to act on the finely crushed part, and the balance at the secondary grinder where the ore is further comminuted, a saving may be effected and greater efficiency obtained at the same time. Thus in treating an ore requiring 16 lb. lime per ton, the most effective way of adding it was as follows: At the coarse crusher, 3 lb.; at the stamps, 9 lb.;

at the tube mill, 4 lb. In order to get the correct lime conditions for this ore it was necessary that the proper amount be present at each stage of the treatment, and it would have been impossible to get such distribution by adding the entire 16 lb. at the coarse crusher.

The proper addition of lime to cyaniding ores becomes a difficult problem in the case of custom milling, where small lots of different ores are constantly going through the mill. Equal difficulty ensues in the case of ore from one mine in which the acidity fluctuates often. The most satisfactory solution for such problems is to bed the ores, thus making as homogeneous a mixture as possible. An alternative is to keep different classes of ore in separate bins and draw from each to make a uniform mixture.

Attention is called to the varying quality of lime, and to the necessity of buying lime on the basis of its "available" alkaline content. The test used for this purpose is that recommended by the Mines Trials Committee, of South Africa, and is as follows: To 2 gr. of lime crushed to 60-mesh, add 50 gr. of crystallized sugar and 800 c.c. of water. Shake thoroughly and allow to stand for a few hours with occasional shaking. Make up to 1000 c.c., shake and take out 50 or 100 c.c. with a pipette. Titrate with standard oxalic acid using phenolphthalein as an indicator. The standard oxalic acid can be prepared easily by dissolving 22.5 gr. of the crystallized acid in 1000 c.c. of water. In such a solution 1 c.c. = 0.01 gr. CaO.

Standardization of Metallurgical Terms.—Much confusion occurs in technical writing owing to the lack of uniformity in the use of technical terms. With a view to clarifying the situation, the Council of the Chemical, Metallurgical and Mining Society of South Africa has adopted certain standard terms and definitions for use in the society's *Journal*. Some of the terms refer mainly to Rand practice, but most of them will find acceptance in other countries.

1. The following words shall be treated as collective nouns, and therefore used in the singular, unless it shall be necessary to use the plural to make careful distinction: Sand, tailing, fine, black sand, residue, ash, pyrite, concentrate, slag, sulphide, telluride, slime, chloride.

2. The following shall be used in the plural form: Shavings, skimmings, steamings, sweepings, drillings, reserves.

3. Preference should be given to the singular form in: Precipitate, bin, cost, profit, value.

4. The following abbreviations shall not be printed in the plural: Oz., lb., dwts., gr. (grain), c.c., cm., kg., in., gal., m., gm. (gram), mm.

5. As far as possible the following terms (local expressions) shall be avoided: Dirt, mullock or muck (broken ore or waste), kokopan (truck), cheesa stick (firing stick), lasher (shoveler), lashing (shoveling).

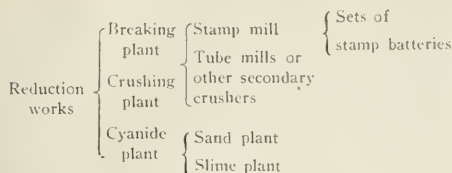
6. The following spelling shall be used consistently: By-product, shoot (geological term), chute (opening through which material is guided), by-pass, hydraulicking, siphon, kafir, gram, dike, rise.

7. "Vat" shall be used instead of "tank," "cyaniding" instead of "cyanidation"; "breaking and sorting" instead of "crushing and picking"; (Note: "crushing" equals "stamp-milling" plus "tube-milling"); "ore slime" to distinguish from "gold slime"; "extraction" to refer to metal precipitated from solution; "recovery" to refer to metal won in a form suitable for market.

8. In referring to screening, it should be stated whether the number of holes refers to the square or linear inch, and the diameter of the aperture in decimals of an inch should be given; battery screen should have the gage of wire in decimals of an inch and the percentage of discharge area, mentioned, whenever possible. Temperatures should be expressed in degrees Centigrade, with the equivalent in Fahrenheit in parenthesis. Cyanide and alkali strength shall be expressed in terms of specified reagents, thus: 0.05 per cent KCN; 0.02 per cent NaOH; 0.02 per cent CaO.

9. A ton shall be 2000 lb. A fluid ton shall mean 32 cu. ft.

10. The following scheme illustrates the nomenclature of Rand reduction works:



Fuel.

Pulverized Coal for Metallurgical Furnaces.—It is only within the last 15 years that means for producing initially a fire of high efficiency have advanced to their present state of perfection. In a paper published in the October *Bulletin* of the A. I. M. E., Mr. H. R. BARNHURST presents a case for the use of pulverized coal in metallurgical furnaces. The cement industry was the first in which pulverized coal was extensively used to supply heat for the chemical reactions taking place in the operations, but the production of heat in this way has since extended to other forms of chemical and metallurgical industry. Coal properly ground will burn thoroughly if 85 per cent passes through 200 mesh and 95 per cent through 100 mesh, the measuring screen apertures squaring 1/400 and 1/200 in. respectively. The author has found that such properly ground coal will contain a percentage above 70 capable of passing through a 300-mesh screen, squaring 1/600 in. on the side.

Undoubtedly the proper method of firing pulverized coal is to admit with the fuel the exact quantity of air necessary to the result to be attained, as shown by observation, and to maintain the relationship between the fuel and air quantities as long as conditions desired are being fulfilled. The control of the two factors, fuel and air, is and will be at the root of all success with pulverized or sprayed fuel in metallurgical processes. Pulverized fuel has a great advantage in that the eye of the intelligent operator can tell him whether he is getting the heat to serve his purpose. Also, it is best prepared for the absorption and evolution of heat in the combustion process, and in addition the air is properly prepared for joining in the process.

The author submits the following data on the use of pulverized coal in metallurgical furnaces: In roasting carbonate ores of high sulphur content, the carbon has been driven off and the sulphur reduced within permissible limits by the use of less than 7.5 per cent of fuel upon the weight of the charge. This problem involves the maintenance of a low temperature, about 2100 deg. F., to prevent agglomeration of the fine ore. The same practice obtains in the roasting and nodulizing of ores and flue dust, where the heat is sufficient to cause the material to form nodules but not stick to the furnace.

In open-hearth practice with pulverized coal, steel is being made with this fuel at the rate of from 450 to 500 lb. per net ton of product. This is an average of 45 heats, the fuel and product being carefully weighed. The melts were obtained in less time than with oil, which had been used previously. Following are analyses of slags with each fuel:

	Oil	Coal
SiO ₂ per cent.....	16.0	16.5
FeO, per cent.....	22.0	18.2
MnO	7.4	6.7
P ₂ O ₅	1.7	1.0

The sulphur in the final steel ranged from 0.025 to 0.035 per cent with oil, and 0.035 to 0.04 per cent with coal. The coal contained from 1 to 1.15 per cent sulphur.

In puddling furnaces it is safe to say that iron can be puddled at an average expense of 1200 lb. pulverized fuel per gross ton of muck bar produced; in fact, less than 1000 lb. of coal per gross ton of bars has been shown in practice during periods favorable to high economy.

The average consumption of pulverized fuel in heating furnaces seems to be from 500 to 550 lb. of fuel per gross ton. Busheling furnaces require from 550 to 600 lb.

"To obtain such results, however, the furnaces must be properly proportioned and equipped and in good condition. It must

not be expected that the results obtained by simply squirting coal of greater or less degree of pulverization into a furnace with an unmeasured jet of air, will equal the practice here shown. The figures given are authentic and may be relied upon. Success insists upon dry coal, fine pulverization, proper air supply. Another factor is that the attendants should be interested in the production of high results."

"With the proper predetermined proportioning of the separation, the operation will be elastic or adjustable to a wide range of performance under a very nearly constant percentage of efficiency. This is unattainable in an installation which may or may not be proportioned to the production of high efficiency at any particular duty. For this reason the ease with which pulverized fuel may simply be burned does not assure that the best results are being obtained."

Fuller's Earth

Bulletin No. 71 of the United States Bureau of Mines on the subject of "Fuller's Earth" has just been issued. Charles L. Parsons, the author, in the introduction says:

"The United States produces all of the fuller's earth used for refining petroleum within its borders. On the other hand, most of the fuller's earth used in bleaching edible oils has been imported from England. Recently a few of the refiners of cottonseed oil have begun to utilize domestic fuller's earth, whereas others have been unable to substitute it successfully for English earth in their practice. Many samples of American earth distinctly superior in bleaching power to the English earth have come to the attention of the Bureau of Mines. For these reasons an investigation of the mining, preparation, and use of fuller's earth in this country, especially in its application to edible oils, was conducted in order to ascertain why our own raw material has been deemed inapplicable to our needs.

"During the calendar year 1912, according to figures of the United States Geological Survey, there were imported into the United States 1,070 tons of unground fuller's earth, valued at \$11,619, and 17,130 tons of ground fuller's earth, valued at \$133,718, these values being based on the wholesale market price at the port of origin. The addition of transportation charges, commissions, etc., make the price to the American refiner \$14.50 to \$16 per ton. According to the latest figures compiled by the United States Geological Survey, the United States in 1912 produced 32,715 tons of fuller's earth, valued at \$305,522 or \$9.34 per ton, at the mine. Most of this domestic production was from three plants in Florida and one in southwestern Georgia and was used almost wholly for decolorizing petroleum. A small amount from other localities was used in the refining of edible oils. No figures definitely differentiating fuller's earth from other clays are kept in regard to our exports, but it is certain that several thousand tons of domestic earth was exported to Germany, and it is also true that German refiners of edible oils have used and are using large quantities of American fuller's earth and have a higher appreciation of its merits than our own refiners.

"As a result of the investigations made, the Bureau of Mines believes that the United States has fuller's earth far better suited for refining edible oils than any imported, and that to assure the almost universal use of this earth by American refiners there is required only a careful and intelligent technical control of the preparation of the output and its application to the bleaching of oils."

Copies of this bulletin may be obtained by writing to the Director, Bureau of Mines, Washington, D. C.

The International Oxygen Company announces that the steadily increasing demand for the L. O. C. system of oxygen and hydrogen generating plants has made necessary the removal of their executive offices from 115 Broadway, New York City to their works at Newark, N. J. Additional buildings have been erected to house the executive offices and provide additional manufacturing room for the increased demands made upon the plant. The general sales offices will remain at 115 Broadway, New York, as heretofore.

Dry Concentration and Separation of Minerals.

The Plumb Pneumatic Jig (Editorial Correspondence)

Processes for concentration in the dry way have been experimented with for many years. The goal which has attracted inventors has been the evolution of a machine that would take the place of wet concentrators, particularly in localities where the water supply was insufficient to meet the requirements of wet ore-dressing. Some of the more recent proposals offer the possibility of direct competition with wet methods. The fact that only a few of the dry concentrators invented have ever attained commercial prominence, suggests the probability that the great majority reached only the patent-office stage of development.

Richards¹ classifies pneumatic concentrators under three heads: (1) those in which a continuous air-blast acts on the ore particles and yields graded products; (2) those in which the ore particles are projected through the air by some other force, and are graded according to their momentum; and (3) air jigs.

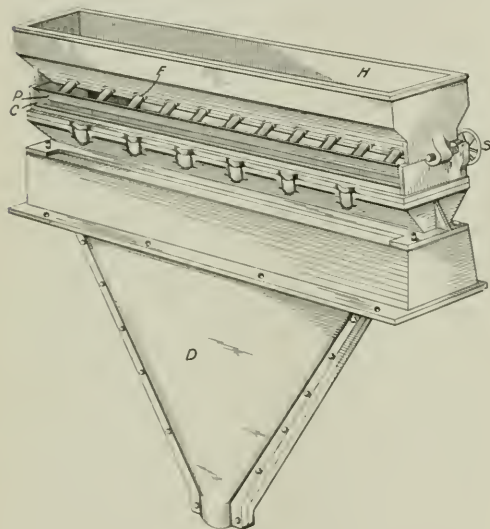


FIG. 1.—PERSPECTIVE VIEW OF JIG

or those in which a bed of ground ore is subjected to an upward pulsating current of air, resulting in a stratification of the constituent minerals and gangue according to their specific gravities. The pneumatic jig under consideration, which is the invention of Mr. A. M. Plumb, of Denver, Colo., falls naturally within the third class.

A suggestion which led to the development of the Plumb pneumatic jig was obtained from the operation of the Bonson dry concentrating table for iron ores. In this machine an adjustable deck is covered with two sets of riffle cleats or guides, one above the other, running diagonally to each other and at angles with the sides of the deck. When an ore mixture is fed onto the table, to which a reciprocating motion is imparted, a stratification of mineral and gangue occurs; and as the mass progresses over the deck a skimming action is produced by the upper set of cleats, whereby the lighter gangue is diverted in one direction while the concentrate is directed by the lower cleats to another point of discharge.

This process of skimming a stratified pulp by means of a guide or partition extending vertically downward into the mass, is one of the features of the Plumb jig. The method of

securing stratification, however, is comparable to that used in hydraulic jigs, more particularly the Richards pulsator jig. A bed of dry ore on a screen is subjected to upward pulsations of air, without intermittent suction.

The construction and method of operation of this jig are best explained in connection with the accompanying drawings. Fig. 1 shows the jig in perspective, and Fig. 2 in cross-section.

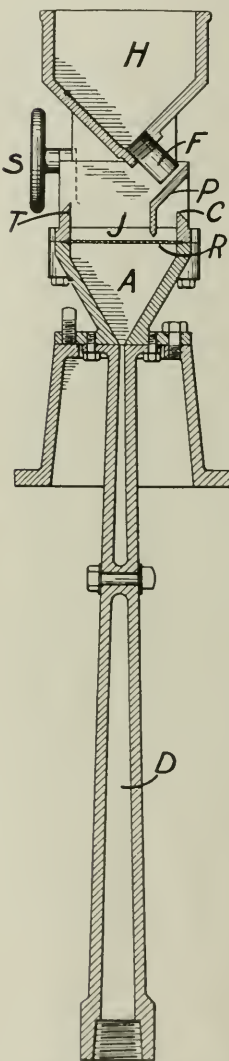


FIG. 2.—CROSS-SECTION OF JIG.

An ore hopper H is provided with a number of feed pipes F, having their lower ends in proximity to the feed plate P. The hopper is adjustable with reference to the feed plate by means of screws S at each end. There is a screen at R on which the ore bed is supported. T is the line of tailing discharge, and C the line for delivery of concentrate. The jig bed J is divided by a vertical extension of the feed plate. There is an air chamber A below the screen, with sides converging to a narrow slot that registers with a slot of similar size in the triangular-shaped air distributor D. The latter is to be connected with a source of compressed air, and a rotary valve is placed in the air line near the jig.

The following dimensions have reference to a jig of commercial size. The base plate is 6 in. by 26 in. The inside dimensions of the jig bed J are 3 in. wide, 24 in. long and 1 in. deep, the last being the height above the screen of the tailing discharge. The concentrate compartment, separated from the main jig bed by the vertical partition, is $\frac{5}{8}$ in. wide, 24 in. long and 0.87 in. deep, the last being the height of the concentrate discharge. The partition between the jig bed and concentrate compartment extends to within $\frac{3}{16}$ in. of the screen. The 24-in. slot for admission of air is $\frac{1}{16}$ in. wide. The screen is 150 mesh, supported between two 20-mesh screens. These dimensions may suggest a toy or model rather than a commercial concentrator, but the jig as outlined shows a capacity ranging from 5 to 10 tons per day, varying with the ore.

No less surprising is the fact that a single form of construction, with no adjustable parts save the feed hopper, should suffice for the treatment of practically all concentrating ores, both with regard to variations in the size of feed and mineral content. A fixed relation has been established between the heights of tailing and concentrate discharges, that will serve for the treatment of a wide range of ores. The conditions governing this fixed form of construction are given below, reference being had to Fig. 3. The figure represents a cross-section of the jig bed, drawn full size. When the jig is in normal operating condition on a given ore, the concentrate and tailing assume relative positions as shown. The tailing plus con-

¹ *Iron Dressing*, Vol. 11, pp. 344-347; and Vol. 111, pp. 1553-1555.

concentrate in the jig bed will balance the concentrate in the other compartment.

Designate the heights of the tailing and concentrate strata in the jig bed as x and y , respectively, and the height of clean concentrate in the other compartment as unity. Assume an ore of the following characteristics:

Sp. gr. of the concentrate, $G = 3$

Sp. gr. of the tailing, $g = 3$

$$3x = 5y, \text{ or } x = \frac{5y}{3} \quad (1)$$

$$5x + 3y = 5 \quad (2)$$

Substituting in (2) the value of x in (1) we have

$$\frac{25y}{3} + 3y = 5, \text{ or } 25y + 9y = 15$$

Hence, $y = \frac{15}{34} = .441$. The general expression for the

$$\text{value } y \text{ is } y = \frac{25 + 9}{G^2 + g^2}$$

Substituting the value of y in (1) we get $x = \frac{2.205}{3} = .735$

Thus the height of $x + y$ for this particular ore will be 1.176, or 117.6 per cent of the height of the concentrate discharge.

For other combinations, for example where $G = 4$ and $g = 2.6$, or $G = 7$ and $g = 5$, the respective values of $x + y$

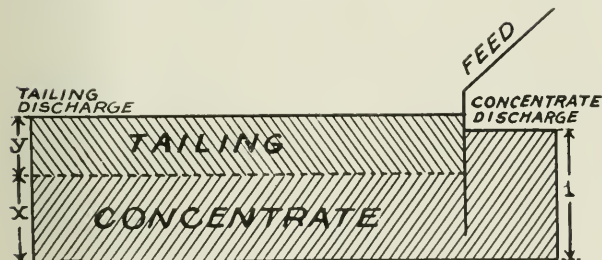


FIG. 3.—DIAGRAMMATIC CROSS-SECTION OF JIG BED

will be 1.157 and 1.137. The average of these conditions and of others likely to be encountered in concentrating ores will give a value of $x + y = 1.15$; so that if the height $x + y$ in the jig bed be made exactly 1 in., the height of the concentrate discharge will be 0.87 in., and these fixed dimensions will be within 2 per cent, plus or minus, of the figures indicated for any of the various combinations of mineral and gangue.

In operation, the dried ore pulp is fed through the hopper onto the screen. Under the action of the pulsating air current the ore bed becomes "light"; that is, the solid particles are separated from each other by the air which is rising through the mass and gradually expanding to atmospheric pressure. A condition of hindered settling in air is produced, which permits a ready stratification of the constituents of the ore, according to their specific gravities. After a few minutes of operation the bed acquires its normal operating condition for that particular pulp, and the jig bed assumes the condition shown diagrammatically in Fig. 3. The constant addition of further feed causes a displacement which results in the discharge of material over the two weirs, concentrate on one side and tailing on the other. The particles of heavy mineral pass down to the screen, flow under the partition, rise on the opposite side and ultimately pass out of the compartment due to the pulsating air. The tailing particles which are too light to sink into the concentrate stratum are constantly skimmed and discharged over the tailing weir. Thus a single machine can make but two products, concentrate and tailing. The latter may contain lighter minerals which will require a further separation.

Experimental work with the jig has resulted in defining very clearly its limitations as to size of feed. It is not well adapted to work on pulps coarser than 12 mesh, and it will not make a satisfactory separation of minerals ground finer than 150 mesh. Between these two limits, however, the machine gives satisfactory results.

It is a generally accepted principle that the feed for pneumatic concentrators should be more closely sized than for machines using water. The necessity for close sizing for the Plumb jig has not proved as great as might be generally supposed. The results of many tests indicate that three or four sizes between 12 and 150 mesh will be ample for good work, and that considerable latitude can be allowed in the sizing efficiency without materially altering the results of separation.

A convenient grading of the feed is $-12 + 20$, $-20 + 40$ and $-40 + 150$ mesh. Some data of operation are compiled in the following table:

Jig No.	Grade	Air Supply			Capacity, tons, 24 hr.
		Pres- sure, per min.	Pulsa- tions per min.		
1	$-12 + 20$	40	60	700	10
2	$-20 + 40$	10	40	500	8
3	$-40 + 150$	3	15	500	4

Thus three jigs will have a combined capacity of about 1 ton an hour, for which the power requirement will be 8.6 hp.

One of the interesting and valuable features of the jig is its automatic regulation under variations of feed. For any given ore a maximum rate of feed will be determined, dependent on the quantity of mineral to be separated. For all variations in rate of feed between this maximum and zero, the jig is self-regulating and will perform its functions without adjustment. Should the feed cease entirely, the jig would merely stop discharging concentrate and tailing, and would continue to work indefinitely without disturbing the ore bed. When the feed came on again the separation would proceed as usual. An excessively high rate of feed would result in loss of mineral in the tailing.

Equally good regulation is obtained automatically when sudden changes occur in the grade of the feed. Thus if normal feed contains, say, 10 per cent galena, and a change should give 15 per cent, the line between concentrate and tailing (Fig. 3) would rise to a new position and remain there as long as the new feed continued. Should the grade fall to 5 per cent galena, the line between concentrate and tailing would be lowered. In the opinion of a competent authority this is the only jig ever built that perfectly regulates the discharge of concentrate and tailing.

The capacity of the jig will vary with the nature of the ore treated. It is limited by the rate at which concentrate can be discharged. Hence the tonnage capacity will be high on an ore with low mineral content, and *vice versa*. The jig shows its best capacity when treating an ore that yields three tons of tailing per ton of concentrate.

The field of usefulness of the jig is limited by the efficiency of coarse hydraulic jigs on the one hand, and slime concentrators on the other. In extremely favorable cases the jig might be the only concentrator needed to make the best commercial saving; but in most instances it will be used as an accessory to other concentrating and separating devices, such as tables and electrostatic separators. In any event, its work will be confined to that part of the ground or between 12 and 150 mesh in size.

The concentrates produced by this jig are always of high grade, regardless of the grade of the feed; that is, two ores containing 2 per cent and 10 per cent galena, respectively, will yield equally high grade concentrates. After the jig assumes normal operating conditions, as shown in Fig. 3, only heavy mineral particles can reach the screen and pass to the concentrate discharge. The production of high-grade concentrate is a great

advantage to the operator for it enables him to secure the highest market price for his products. In the case of a one-mineral separation, for example, the jig offers an opportunity to increase the quantity of high-grade concentrate, and thus produce a greater net profit, even if the total recovery is not increased. Improvement is shown, also, in the treatment of complex sulphide ores, where the total mineral recovery by wet processes usually is low. In some cases of this kind it has been possible by dry jiggling and electrostatic treatment to separate and recover a part of the minerals at 12 mesh in the form of a clean, high-grade concentrate, leaving a tailing which contains the really complex part of the ore that requires fine grinding and separation on wet tables. Excellent results have been obtained in retreating concentrates from ordinary wet-dressing tables.

A few examples of the work of the jig are given below:

Retreating Products from Wet Concentration.—The two tests below show what can be done by dry jiggling and electrostatic separation in raising the grade of products from table concentration. The original concentrate fairly represents the work done in wet-dressing a lead-zinc-iron ore. But there is a considerable quantity of zinc in the product which is without value; in fact, in the second case the zinc entails a penalty at lead smelters. By separating the original concentrate it is possible to secure this zinc in a merchantable product that will add to the net profit. The method of treatment in both of these tests was to jig the original table concentrate dry, getting a high-grade lead product and a tailing containing the zinc and iron with some lead. This tailing was then separated electrostatically on the Huff machine.

	Weight	Lead	Zinc	Iron
	Per cent.	Per cent.	Per cent.	Per cent.
Test No. 1.				
Original table concentrate.....	100	54.50	10.71	9.85
Pneumatic jig concentrate.....	65	77.00	1.97	4.32
Pneumatic jig tailing, separated electrostatically into two products	20	19.80	8.68	14.00
.....	15	4.60	50.90	6.20

	Weight	Lead	Zinc	Iron
	Per cent.	Per cent.	Per cent.	Per cent.
Test No. 2.				
Original table concentrate.....	100	40.86	14.70	10.30
Pneumatic jig concentrate.....	43	78.80	1.90	2.30
Pneumatic jig tailing, separated electrostatically into two products	31	18.75	5.60	26.40
.....	25	4.00	46.80	2.80

Treatment of Crude Ore.—In the following test the ore was crushed through 12-mesh and treated by pneumatic jiggling, electrostatic separation and wet dressing, these different processes being applied to the portions of the ore best suited to the respective methods. The tables below show the weights and metal contents of ore and products, and percentages of metals recovered in different salable concentrates.

	Weight.	Gold.	Silver.	Copper.	Lead.	Zinc.	Iron.
	Per cent.	Per oz.	Per ton.	Per cent.	Per cent.	Per cent.	Per cent.
Crude ore	100.00	0.26	8.56	0.55	24.10	10.00	2.60
Pneumatic jig concentrate	25.75	0.32	18.36	0.16	78.20	2.6	0.78
Electrostatic copper iron	5.80	1.09	36.00	5.55	31.18	13.20	11.50
Electrostatic zinc	14.00	0.14	3.62	0.34	5.30	48.00	1.00
Tailing	47.50	0.12	1.10	0.18	1.33	2.56	2.10
Dust and slime	6.74	0.30	0.40	0.40	16.00	7.40	..

Percentage Recovery in Salable Products

	Gold.	Silver.	Copper.	Lead.	Zinc.
	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Lead concentrates	31.70	55.25	..	83.56	..
Iron-copper concentrates	25.50	25.30	59.5	7.60	..
Zinc concentrates	7.54	5.04	..	3.00	67.56

Total recovery..... 64.74 80.49 59.5 94.25 67.56

The pneumatic jig is not offered by its sponsors as a cure-all for the ills of concentration. Its apparent usefulness is within

a well defined field, the limitations of which are appreciated. Developments up to date, however, indicate that even within its limited field the pneumatic jig can produce marked and favorable improvement in the present methods of concentrating and separating ores.

The jig is manufactured and sold by the Plum Jig Company, a subsidiary of the American Zinc, Lead and Smelting Company. The latter maintains a custom testing plant and assay laboratory at 1422 Blake Street, Denver, Col., where tests can be made on the Plum jig. Huff electrostatic separator and other concentrating machines. The charges for the work are nominal considering the quality and extent of the tests.

Titanium for Rail Steel

The progress which metallographic methods have made in recent years in the metallurgical art is reflected in a characteristic way in three bulletins issued recently by the Titanium Alloy Manufacturing Company, of Niagara Falls, N. Y., and entitled "Rail Reports" Bulletins No. 1, 2, and 3.

A number of samples of standard and titanium-treated open-hearth rails were sent by several different railroad systems to the laboratories of the Titanium Alloy Manufacturing Company. All the samples were from "A" rails (the top rail from the ingot which usually shows the most excessive segregation): in each case both the standard and titanium-treated rolls had been rolled by the same mill, under the same specifications, with the single exception of the use of 0.10 per cent titanium in the treated steels.

The three bulletins give in detail the results of chemical and physical and microscopic tests of the seven different pairs of samples. A convenient summary of the results of the chemical and physical tests for the seven pairs of samples is given in Bulletin No. 3 in form of a long table from which the following general conclusions are drawn.

First. The tensile tests show on an average much better ductility for the treated rails, especially in the heads, and at the same time a slightly better strength than for the untreated.

Second. The impact tests show an average increase of shock resistance for the treated steel, in the head, web and flange, of about 50% over that for the untreated.

Third. The endurance tests on the White-Souther machine show that the treated rails are much less easily fractured by fatigue or by constantly repeated stresses below the elastic limit than the untreated.

Fourth. The chemical analyses, hardness tests, and Landgraf-Turner endurance tests (in which the stresses exceed the elastic limit), all show a much greater uniformity in the treated samples than in the untreated, indicating that the treated rails are freer from segregation and all its attendant evils.

To sum up all these points, it seems certain that these treated rails, when laid in the track and subjected to the strains and shocks of service, will form a safer line than would the untreated rails and that if excessive segregation is dangerous, the use of ferro-carbon-titanium in open-hearth rail steel is imperative.

Interesting as these physical and chemical tests are yet the beautiful series of sulphur prints and microphotographs given in the three bulletins are probably of even greater interest as they are a distinctly novel feature in a manufacturer's bulletin.

The sulphur prints are made by placing photographic paper on the polished cross sections of rail steels. This paper is moistened with a very weak sulphuric acid solution; and the sulphur in the steel reacts with the acid, forming hydrogen sulphide, which combines with the silver in the paper, producing dark spots thereon. The sulphur print is a rapid and accurate method of determining the quality of steel, and excessive segregation shown by this method indicates a weakness, which the detailed physical tests confirm.

Carbon, phosphorus, and sulphur segregate under similar conditions, so that the degree of segregation of sulphur as

shown by a sulphur print, determines the comparative degrees of segregation of phosphorus and carbon as well. Ample evidence is given in the latter of the three bulletins that this parallelism in the segregation of carbon, phosphorus, and sulphur is indeed a reality.

Cross-sections of the rails were also etched with iodine to reveal segregation and the pictures give, in the whole, the same indications as the sulphur prints.

Finally, a beautiful set of *photomicrographs* is given in each of the three bulletins. These photomicrographs were made of sections taken from the top of the head, center of the head, the web, the bottom of the flange, and the end of the flange. Further microphotographs were made of unetched sections as well as of sections under different etching treatments.

In short, these bulletins are exceedingly interesting. Quite apart from the primary object which is to permit the reader to form an unbiased judgment of the value of titanium treatment of rail steel they permit a very suggestive comparison between the different methods of testing—chemical, physical, and microscopic.

As to the value of titanium-treated steel rails in practice Messrs. Robert W. Hunt & Company are quoted in No. 3 as follows: "Through comparison of wearing results as between the standard open-hearth and the titanium-treated open-hearth rail, the latter in each of a number of instances on four railroad systems has demonstrated its superior wearing qualities. We have yet to find a single exception where the claim just pronounced is not true."

Wet and Dry Bulb Recording Hygrometer

Recording hair hygrometers, giving a single record of the relative humidity, have been in use for a long time. Engineers have appreciated the distinct advantage of having a record of both the dry-bulb temperature and the wet-bulb temperature independently but simultaneously on the same chart. Such an instrument has the added advantage in the ease with which its accuracy can readily be checked with a standard thermometer at all times. The importance of proper conditions of temperature and humidity is being more and more appreciated in its effect on health and human efficiency as also on the effect on many products during process of manufacture.

To meet the growing demand for a wet and dry bulb recording thermometer, the Industrial Instrument Company, Foxboro, Mass., have designed and placed on the market the recording hygrometer illustrated in Fig. 1. It consists of two sensitive bulbs mounted in tandem back of the case as illustrated in

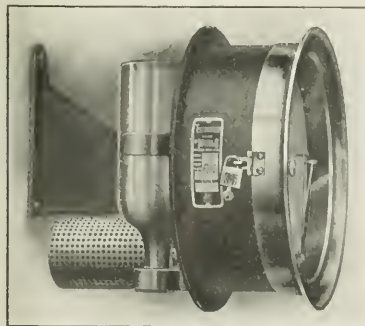


FIG. 1.—WET AND DRY BULB RECORDING HYGROMETER

Fig. 2, the wet bulb being jacketed and kept moist by maintaining water at a constant level in a trough beneath. The pen arms are attached directly to shafts concentric with the helical tube bulbs. The case is mounted on a swivel bracket enabling swinging the instrument at right angles to the wall or support, giving easy access to the inverted glass bottle serving as

water reservoir. It is made in three sizes, viz., 8 in., 10 in. and 12 in., corresponding to the standard sizes of *Barber* recorders and to cover ranges between freezing point and boiling point of water (32 deg. to 212 deg. Fahr., or 0-100 deg. C.).

Fig. 2 illustrates the manner in which the helical tubes acting

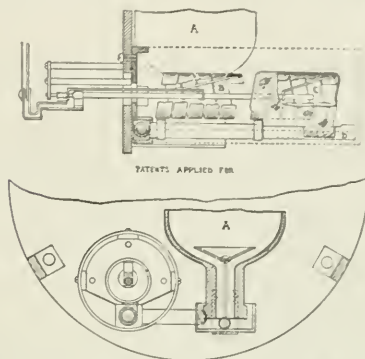


FIG. 2.—CONSTRUCTION OF HYGROMETER

as the bulbs are placed in relation to each other and the reservoir A.

These two bulbs extend beyond the case at the rear and the motion is transmitted to the pens through the actuating shafts. The tube marked B is the dry bulb which records the atmospheric temperature. The tube C is the wet bulb, which is covered with a special jacket, leading down into the trough D.

This bulb is always cooler than B due to the evaporation of the water. This evaporation increases or diminishes in proportion to the amount of moisture in the air. Taking the difference between the two records made by the pens and consulting the table sent with the instrument, the relative humidity can be quickly ascertained.

Personal

Mr. G. F. Brindley, who was works manager of the Niagara Electrochemical Company, Niagara Falls, N. Y., from 1896 to 1906, when he left for Japan, subsequently held the position of consulting chemist and engineer to Dr. E. S. Pearson in Mexico, and has recently returned from a prolonged stay in New Brunswick where he was carrying on an investigation of the oil shales of that place. He is at present doing some experimental work at The FitzGerald and Bennie Laboratories, Niagara Falls, N. Y.

Mr. Franz Cazin, of Denver, Col., has been obliged to return to the United States from Mexico, where he was engaged in the design and construction of a large metallurgical plant. Conditions in the southern republic made further work impossible.

Mr. Robert C. Gemmel, general manager for the Utah Copper Company, celebrated his twenty-fifth wedding anniversary on October 17, at which time the operating heads of the Utah Copper Co. presented Mr. and Mrs. Gemmel with a beautiful silver punch service. The bowl is engraved with a panorama of the great Utah copper mine and with scenes depicting the mining and transportation of ore.

Mr. Jay P. Graves, who has been vice-president and general manager of the Granby Consolidated Mining, Smelting & Power Co., of British Columbia, has retired from the active management of the company. He is succeeded by **Mr. F. M. Sylvester**, who has been assistant general manager.

Mr. John Gross, metallurgical engineer of Denver, Col., has finished the construction of a new mill for the London Mines Company, in Gilpin county.

Mr. John T. Jones, of Iron Mountain, Mich., was in Denver recently en route to Utah and Arizona on professional business.

Mr. Robert M. Keeney, of the Bureau of Mines, Pittsburgh, Pa., was married October 22 to Miss Maud Conrad. Mr. and Mrs. Keeney are at home at the Moyer apartments, Oakmont, Pa.

Mr. Thomas M. Kekich has been engaged by the Spassky Copper Mines, Ltd., Siberia. He recently resigned his position as assistant superintendent of the Great Western S. & R. Company, Chicago.

Sir Robert Mond and Sir Alfred Mond were the guests of honor at a dinner tendered by Dr. L. H. Baelkand at the Chemists' Club in New York before their return to England. Dr. Baelkand acted as toastmaster. Sir Robert Mond spoke impressively of the scientific, industrial and commercial problems inherited from his father. Sir Alfred Mond, M. P., spoke very amusingly of the life of a British politician. The other speakers were Dr. W. H. Nichols, Dr. C. F. Chandler and Dr. J. Takamine.

Mr. H. N. Thompson has resigned his position as superintendent of the Tooele smelter of the International Smelting & Refining Co., in Utah. William Wraith, manager of the plant, will assume the duties of Mr. Thompson, and will be assisted by O. M. Ruchs, who has been chief chemist.

Obituary

Ashmead Gray Rodgers for twelve years superintendent of The Carborundum Company's plant at Niagara Falls, died October 23d, 1913, as the result of injuries sustained through an accident on October 3. Mr. Rodgers had a host of friends and acquaintances throughout the mechanical and chemical world and they will sincerely feel the loss of a man who was so pleasing in personality, democratic in his attitude to others, and so capable in his chosen line of work. Mr. Rodgers was a native of Albany, N. Y., having been born there in 1872. Previous to his coming to The Carborundum Company as superintendent he was superintendent of the Eddy Electrical Company, of Hartford, Conn. His funeral services were held Saturday, October 25, from St. Peter's Church at Niagara Falls and was attended by several hundred of the employees of The Carborundum Company, members of the Niagara Club and other friends. Mr. Rodgers was a member of the American Electrochemical Society, the American Chemical Society, Engineers' Society of New York, Chemists' Club, Niagara Club, University Club and Country Club of Niagara Falls and several other scientific and social organizations.

Book Reviews

Metallography. By Cecil Desch.

This book was reviewed in our November number, but the title erroneously given as "Metallurgy."

New Standard Dictionary of the English Language. Edited by Isaac K. Funk; prepared by nearly four hundred specialists. Large quarto (30 x 22 cm.), xxxviii + 2916 pages, nearly 10,000 illustrations, 450,000 definitions. Price, \$21.50, in better bindings, \$27 and \$35. New York and London: Funk & Wagnalls Company.

The appearance of a new complete dictionary, prepared with the great expenditure of money and brains which have been lavished on this new edition of the Standard, is an event worthy of notice in a technical journal. Confining our remarks to the scientific side of the work, and particularly to that which concerns our readers, chemical terms have been supervised by Dr. H. W. Wiley, geology by Director George Otis Smith, of the U. S. Geological Survey, metallurgy and electrochemistry, by Dr. J. W. Richards, mineralogy, by Dr. F. W. Clarke, gems and precious stones by Dr. G. F. Kunz,

electricity by Dr. Samuel Sheldon, engineering and mechanical terms, by C. A. Munn, mining by Dr. J. C. Bayles, former editor of *The Iron Age*. With equally competent experts in charge of other departments of the book, the result is probably as near perfection as it is humanly possible to make such a colossal work. From the printer's and publisher's standpoint, it is a masterpiece of the book-maker's art.

General Metallurgy. By H. O. Hofman, Professor of Metallurgy in the Massachusetts Institute of Technology. Large octavo (15 x 23 cm.), 909 pages, 836 illustrations; price \$6.00 net. New York: McGraw-Hill Book Company, Inc.

This remarkable book is the equivalent of Schnabel's "Allgemeine Hüttenkunde," brought up to date and written in the spirit of modern American metallurgy.

A rather short chapter on the properties of metals is followed by a longer one on alloys; metallic compounds are then described in general terms, with their heats of formation. Over 300 pages are given to "Fuel"; a short 38-page chapter to refractory materials, and 124 pages to smelting processes in general, including fluxes and slags. Only 8 pages are given to the details of hydrometallurgical processes, and but 20 pages to electrometallurgical methods and apparatus. On the other hand, an imposing chapter of 350 pages covers mechanical operations, such as handling ores, working metals and alloys, pumping and filtering liquids, supplying, heating and drying blast, purification of gases and catching of fume.

There is an extraordinary amount of useful information in this work, rendered doubly valuable by numerous references to the literature, so that the reader is everywhere informed where further information can be obtained. There are also numerous mistakes, such as are perhaps unavoidable in compiling such an enormous work, but which must be corrected in the next edition. The reviewer particularly calls attention to the chapter on "Fuel," in which small calories (cal.) and large calories (Cal.) are frequently confused, and in which flame temperatures are in some cases evaluated on a false principle. These, and rather numerous smaller errors and typographical mistakes, are blemishes on the usefulness of what is otherwise a magnificent work and a credit to American technical literature; assuming these corrected in the next edition, we will then be able to recommend the work to students and practitioners, without reservation, as just the work on general metallurgy that we have for so long desired to have.

J. W. RICHARDS.

Industrial Poisoning, from Fumes, Gases and Poisons of Manufacturing Processes. By Dr. J. Ramboisek, Professor of Factory Hygiene and Chief Health Officer of Bohemia. Translated and edited by Thos. M. Legge, M.D., Medical Inspector of Factories. Octavo (21 x 13½ cm.), 360 pages, 61 illustrations. Price, \$3.50 net. New York: Longmans, Green & Co. London: Edward Arnold.

The author is a medical man, a chemist, and a government official, and has treated the subject in a novel, comprehensive and systematic manner. The translator is a physician, and lecturer on factory hygiene in the University of Manchester, England.

The first part of the book describes briefly the process of manufacture of metals and chemicals, the opportunities for poisoning in each case and the symptoms and effects. Part two deals with industrial poisoning in general, its classification, pathology and treatment. Part three deals with preventive measures in general, and in particular cases. A classified bibliography is given in an appendix. One omission is the peculiar poisoning caused by the dust of vanadium ore and oxide, but this is a rather recent discovery; another is the dangerous effects of osmium oxide vapor. The book is, however, very complete, and so well written as to be really fascinating in its interest.

Every worker or firm manufacturing, handling or using chemicals or metals should get a copy of this book.

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